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Lafigué Project, Côte d'Ivoire

NI 43-101 Technical Report

Definitive Feasibility Study (DFS)

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Prepared by:

Lycopodium Ltd

Address: Level 5/1 Adelaide Terrace, Perth WA 6004, Australia

Phone: +61 8 6210 5222

Website: www.lycopodium.com

Prepared for:

Endeavour Mining plc

Address: 5 Young St, London, W8 5EH, United Kingdom

Phone: +44 203 011 2723

Website: www.endeavourmining.com

Qualified Persons:

Abraham Buys (FAusIMM)

Alex Veresezan (P.Eng)

David Morgan (CPEng, MAusIMM)

David Taylor (CPEng, FIE(Aust)

Francois Taljaard (Pr.Eng)

Geoff Bailey (CPEng, FIE(Aust))

Graham Trusler (Pr Eng, MIChE, MSAIChE)

Lucy Roberts (PhD, MAusIMM(CP))

Silvia Bottero (Pr. Sci. Nat.)

Stuart Thomson (FSAIMM)

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This report was prepared as a National Instrument 43-101 Technical Report (the 'Report') by:

- Mr. Abraham Buys, who is a Fellow of the Australasian Institute of Mining and Metallurgy (AusIMM);
- Mr. Alex Veresezan, who is a Registered Professional Engineer with the Professional Engineers Ontario, Canada;
- Mr. David Morgan who is a Chartered Professional Engineer and a member of the Australasian Institute of Mining and Metallurgy (AusIMM);
- Mr. David Taylor who is a Chartered Professional Engineer with Engineers Australia, and a Fellow of the Institute of Engineers (FIE), Australia;
- Mr. Francois Taljaard who is a is a Registered Professional Engineer, with the Engineering Council of South Africa (ECSA);
- Mr. Geoff Bailey who is a Chartered Professional Engineer and a Fellow of the Institute of Engineers (FIE), Australia;
- Mr. Graham Trusler who is a Registered Professional Engineer with the Engineering Council of South Africa (ECSA), a Fellow of the Water Institute of South Africa, and a Member of the South Africa Institute of Chemical Engineers;
- Dr Lucy Roberts who is a Chartered Professional (CP), and Member of the Australasian Institute of Mining and Metallurgy (AusIMM);
- Ms. Silvia Bottero who is a Certified Professional Natural Scientist with the South African Council for Natural Scientific Professions (SACNASP); and
- Mr. Stuart Thomson who is a Fellow of the South African Institute of Mining and Metallurgy (SAIMM);

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1. SUMMARY

1.1 Introduction

The Issuer, Endeavour Mining plc (Endeavour) in March 2021 engaged various consultants to prepare a Definitive Feasibility Study (DFS) for the Lafigué Project (the Project). The Project is on Endeavour's Lafigué Exploitation Permit (PE 58)/historical Fétékro Exploration Permit (PR 329), and is located approximately 470 km by road, northeast of Abidjan in Côte d'Ivoire (CI).

Endeavour has an 80% ownership interest in PE 58, hereafter also referred to as the 'Lafigué Mining Licence', 'Lafigué ML' or 'Site' and a 100% ownership interest in Exploration Permit (PR 329), hereafter referred to as the Fétékro Exploration Licence or 'Fétékro EL.

Société des Mines de Lafigué SA (SML) is the permit holder for PE 58, whilst the permit holder for PR 329 is La Mancha Côte d'Ivoire SARL (100%) or 'LMCI'.

The Lafigué ML was granted 22 September 2021, whilst the exceptional renewal request for the Fétékro EL was received by the authorising authority on 3 March 2022 (approval pending).

The Project comprises an open pit mine, processing plant (the 'Plant') and supporting infrastructure, with a processing capacity of 4.0 Mt/a (db) to produce some 155 to 251 koz/a[1] of gold (average for years 1 to 12, 212 koz/a)[2], over a <13-year life of mine (LoM).

Key mine features are as note below:

- Enabling offsite infrastructure.
  - 33 km, 225 kV Power Transmission line from Dabakala to Site.
  - Upgrade of approximately 17 km laterite access road to Site (2 km new).
- Open pit mine with attendant waste rock dumps and water management infrastructure;
- 4 Mt/a (db) three stage crushing, milling and Gravity/CIL Process Plant (the 'Plant').
- Downstream construct Tailings Storage Facility (TSF).
- General mine infrastructure:
  - Mine Services Area (MSA).
  - Emulsion and explosives facilities.
  - General administration and plant buildings/facilities.
  - Accommodation facilities.
  - Water harvest and storage dams.
  - Airstrip.
  - Contact water management systems.

[1] Excludes year 13, which is not a full year of operation. Doré grade (93±2% gold)

[2] Years are from first gold pour, Q2 2024.

The mine is to be developed/operated under a hybrid business model, with a number of outsourced operational contracts (Section 19).

1.2 Reliance on Other Experts

Where QPs have relied on others for information used and/or presented in this Report, the persons relied upon and the extent of this reliance is stated in Section 3.

1.3 Property Description and Location

1.3.1 Location

The Issuer's Exploitation Permit (PE 58) and the associated Exploration Permit (PR 329) are located in the north-central region of Côte d'Ivoire (CI), approximately 330 km north-northwest of the port city of Abidjan (approximately 470 km by road). The southwest corner of PR 329 lies approximately 63 km north-northeast of Bouake, the second largest city in Côte d'Ivoire (CI), and 38 km east of Katiola. The northwest corner of PR 329 lies approximately 18 km west-southwest of Dabakala; and PE 58 lies within the boundary perimeter of PR 329 (Figure 1-1).

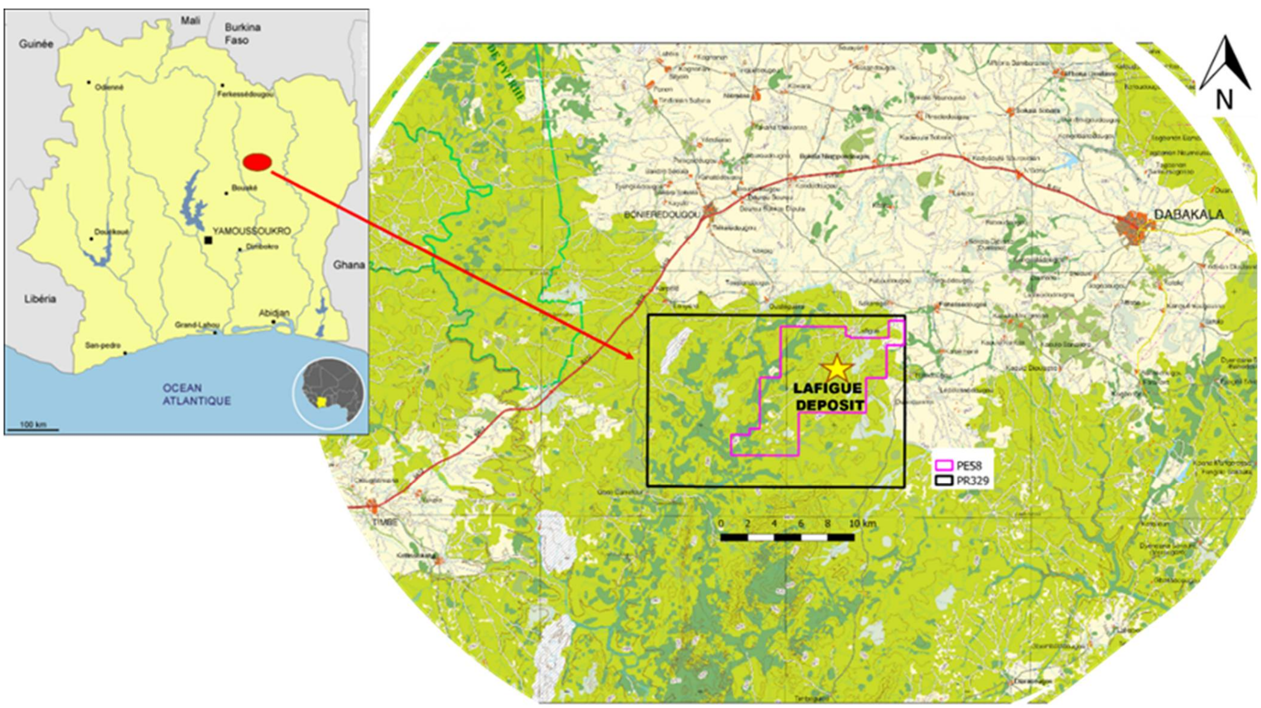

Figure 1-1: Location of PR 329 and PE 58 in CI (Endeavour, 2022)

1.3.2 Mineral Tenure and Title in Côte d'Ivoire

1.3.2.1 Overview

On 24 March 2014, CI's parliament approved Law No. 2014-138 adopting the new mining code (the 'New Mining Code or 'NMC'). A Decree No. 2014-397 implementing the NMC was issued on 25 June 2014 (the 'Decree'). The NMC replaced the former mining code (Law No. 95-553 dated 18 July 1995) (the 'Old Mining Code or OMC').

The NMC reflects the government of CI's desire to attract more investors, particularly in the gold sector, and to better regulate the mining sector as a whole. To this end, several measures have been taken, including:

- the additional profit tax under the OMC, paid by mining licensees has been abolished;
- holders of mining permits sign a mining agreement within sixty working days of being granted a permit (Article 12); and
- the State guarantees the stability of the tax and customs regime to the holder of the mining permit (Article 164).

1.3.3 Exploration Permits (PRs)

In accordance with the NMC, an 'Exploration Permit' 'Permit or PR' is granted by Presidential Decree. The Permit is valid for an initial period of four years and may be renewed for two consecutive periods of three years, with an exceptional renewal for a final two-year period, provided the Permit titleholder, complies with the rights and obligations set out under the NMC. At each renewal, at least 25 per cent of the original area must be relinquished, however the titleholder may elect to maintain the full area, by paying an 'Option Fee'.

The Permit grants an exclusive right to the holder, to explore within the Permit area (not exceeding 400 km^2), and to dispose of the products extracted during exploration activities. However, disposal is subject to a prior declaration to the Ministry and the payment of the applicable mining duties. In addition, the permit holder is automatically entitled to request and obtain an 'Exploitation Permit' at any time during the exploration period, provided that the Permit holder has carried out all its obligations and that a feasibility study has proven the existence of one or several economically viable deposits within the perimeter of the Permit.

1.3.4 Exploitation Permits (PEs)

An 'Exploitation Permit' or 'Permit or PE' is issued for an initial period based on the life of mine stated in the feasibility study submitted for permitting purpose, with a limit of 20 years. At its expiry, a PE can be renewed for successive periods of 10 years maximum. The holder of the Permit is required to sign a mining convention (*Convention Minière*) with the State, within sixty (60) working days from the award of the Permit. The mining convention is valid for an initial period of twelve (12) years (renewable for successive periods of ten (10) years maximum). The purpose of the mining convention, according to the Mining Code, is to stabilise the tax and customs regime.

The holder of the Permit has the exclusive right to:

- exploit the deposits within the limits of the Permit's perimeter;
- transport or to arrange the transport of the extracted ore;
- establish the necessary facilities to condition, treat, refine and transform the ore; and
- trade the ore/product on the internal or external markets (export).

Chapter III, Article 127 of the 2014 Mining Code, prescribes conditions related to occupancy/use of the land. Said occupancy of the land gives the Permit holder the right to:

- use of said land, subject to the lawful occupant of the land receiving fair indemnity (supervision by 'Mines Administration);
- 'cut wood needed for said activity' (not sell); and

- use free waterfalls within the perimeter defined by the mining title.

Article 169, Chapter IV of the 2014 Mining Code, exempts the permit holder from:

- the exploitation levy for the withdrawal of water from the water table as part of mine drainage operations in the perimeter of the permit during the period of validity of the exploitation permit; and
- the felling tax in the perimeter of the permit during the period of validity of the exploitation permit, provided that the woody essences are not sold.

The holder of the Permit does not automatically have the have right to exploit other commodities[3] on the permit, which are not specifically named in the exploitation permit.

The Permit is granted by right, by decree taken in Council of Ministers, to the holder of the exploration permit who has proved that there is a deposit within its exploration permit. Said proof is materialized by a feasibility study.[4]

The applicant must have complied with its obligations under the provisions of the law; and must present an application compliant with the provisions of the implementing decree of the Mining Code, prior to the expiry of the period of validity of the exploration permit under which the application for the exploitation permit is made.

1.3.5 Mineral Tenure, Ownership & Permit Status

Mineral tenure, ownership, and the permit status of PE 58 and PR 329 is summarised in Table 1-1 and discussed in further detail in Section 4.

Table 1-1: Permits, Agreements and Ownership (Endeavour, 2022)

Description	Value	Comments
Exploration Permit:	**Fétékro Exploration Licence**	**'Fétékro EL'(Public/Internal name)**
• Number/Name:	PR 329	Official name on Permit
• Area:	249.8 km^2	Will reduce in size with the 3rd renewal (183.9 km^2)
• Date granted:	06/06/2013 (expired 06/06/2016), Arrêté No. 2013-410 of 6 June 2013.	1995 Mining Code (355 km^2).
• 1st renewal date:	6/6/2016 (expired 6/6/2019), Arrêté No. 090/MIM/DGMG of 11 July 2017.	2014 Mining Code (249.8 km^2).
• 2nd renewal date:	6/6/2019 (expired 06/06/2022) Arrêté No. 00008/MMG/DGMG of 13 January 2020.	2014 Mining Code (249.8 km^2).
• 3rd renewal date:	06/06/2022 (**when granted**, will expire 06/06/2024)	2014 Mining Code (183.9 km^2), exceptional renewal request received by government on 03/03/2022 (Letter reference No. VPE/SB/PK/129/03-2022)
• Applicable mining codes:	2014 Mining Code	
• Historical permit holder	Societe pour le Developpement Minier de la Cote d'Ivoire.	SODEMI
• Permit transfer date:	Arrête No. 00174/MMG/DGMD of 18 December 2020.	Transfer from SODEMI to LMCI

[3] i.e., Aggregate

[4] Whilst a 'feasibility study is stated', the associated level of technical and cost development is not. Thus, in the Issuers case, the Exploitation Permit was granted on a pre-feasibility study.

Table 1-1: Permits, Agreements and Ownership (Endeavour, 2022)

Description	Value	Comments
• Permit holder:	La Mancha Côte d'Ivoire SARL (100%)	'LMCI'
• Shareholder of LMCI:	Ity Holdings Ltd. (100%)	'ITYH'
• Ultimate shareholder of ITYH	Endeavour Gold Corporation (100%);	'EGC'
	Endeavour Mining Corporation (100%); and	'EMC'
	ultimately Endeavour Mining plc (100%)	'Endeavour'
Exploitation Permit:	**Lafigué Mining Licence**	**'Lafigué ML'(Public/Internal name)**
• Number/Name:	PE 58	'PE 58 (official name on Permit)
• Area:	64.08 km^2	
• Date granted:	22/09/2021. Decree No. 2021-538 of 22 September 2021	12-year validity, based on two-year construction period and a 10-year life of mine.
• Expiry date:	21/09/2033	
• Applicable Mining Code:	2014 Mining Code	
• Mining Convention:		Still to be negotiated as of the 'Effective Date'
• Historical permit holder:	La Mancha Côte d'Ivoire SARL	'LMCI'
• Permit transfer date:	12/01/2022	From LMCI to SML
• Current permit holder:	Société des Mines de Lafigué SA	'SML'
• Shareholders of SML:	Lafigué Holdings Ltd (80%);	'LAFH'
	Société pour le Développement Minier de la Côte d'Ivoire SARL (10%); and,	'SODEMI
	Government of Côte d'Ivoire (10%)	'GoCI'
Ultimate shareholders of LAFH	Endeavour Gold Corporation (100%); and	'EGC'
	ultimately Endeavour Mining plc (100%)	'Endeavour'

1.3.6 Surface Rights

As described in Section 4.2.5, SML has the requisite surface rights to develop a mine and the attendant infrastructure required on PE 58, as well as the rights to develop access to said Property. However, this is also contingent on having the requisite permits in place (Section 4.7).

Further, in July 2022, Endeavour approached the Director General of Mines and Geology for the right to blast/abstract non-mineral bearing materials for use in construction. Authorisation was subsequently received, with taxes payable on the material abstracted/used (Ouattara, 2022a).

1.3.7 Encumbrances

All mining licenses carry a 10% free carried interest in favour of the Government of Côte d'Ivoire (CI) GoCI and, as a result, the GoCI holds a 10% interest in SML.

Also, according to the sale agreement of PR 329 entered between LMCI and SODEMI in November 2020, SODEMI holds a dilutable 10% interest in SML; and SODEMI is also entitled to a complementary price of USD 3/oz for every additional ounce of 'Reserves' identified, over and above the Reserves associated with the 2.471 Moz of Measured and Indicated (M&I) gold Resource initially defined on PR 58; and any future mining license issued from the perimeter of PR 329.

1.3.8 Payments

The basis of payments to the State are fully defined in Section 4 and summarised below.

- Free Carry Interest (FCI) - summarised in section 1.3.6.
- Royalties - an 'Ad Valorem' (or proportional) tax is applied to gross sales revenue, after deductions for transport (FOB) and refining and/or smelting costs and penalties. Said tax covers both gold and silver sales.
- Surficial fees - fixed fees based on the granting and renewal of a Permit, and an annual unit rate fee (XOF/km^2).
- Central and Commercial Bank Payments - fees payable on foreign currency payments, not involving EUR or XOF transactions.
- Community Levies - an Ad Valorem contribution of 0.5% of gross sales revenue after deductions for transport (FOB) and refining and/or smelting costs is payable.
- Bonds - a closure bond is payable on the total estimate closure cost, with 20 % of the annual payment made into an escrow account, with the remainder take out as bond with a commercial bank.
- Taxes
  - Overview

    The basis for the application of taxes during construction and production are summarised below. In CI, taxes payable are subject to the definitions outlined in the NMC for 'Production', namely:

    The 'First commercial production date', is the date at which the mine reaches a continued period of production of sixty days at 80% of its production capacity as drawn up in the 'feasibility study' forwarded to the mining administration or the date of the shipment of the mining production for commercial purposes'.

    Importantly, taxes payable by SML if different to the official tax basis outlined in Sections 4.5.9.2 to 4.5.9.13, will be as a result of any amendments to the tax basis in the Mining Convention (not signed as of the 'Effective Date' of this Report).
  - Construction

    During construction, the permit holder is exempt from import duties, except for the Regional/ECOWAS levy of 2.5% CIF (Port). Said exemption excludes duties on chemical products and fuel.
  - Production

    Unless otherwise agreed in the 'Mining Convention', the permit holder will in addition to the 'Regional/Ecowas' levy, be subject to full import duties as defined in the tax code for equipment and consumables, typically 0 to 20%[32] of the CIF value.

Chemical products (including fuel) are exempt of duties and only subject to the Region/ECOWAS Levy of 2.5%.[5]

- Withholding Taxes (WHT)

  Subject to the jurisdiction of the service provider, withholding taxes are applied at a rate of 0 to 20%

- VAT

  Unless agreed otherwise in the Mining Convention, only the Permit holder is VAT exempt for Construction. For Production, the rate will be 18% unless negotiated otherwise in the Mining Convention. The exception being chemical products[6] which are VAT exempt during production.

- Tax on Insurance Premiums

  Subject to the type of Product procured, tax on insurance premiums varies between 0.1 and 25%.

- Dividend Payments

  The policy for the payment of dividends will be as defined in the 'Mining Convention'. In general, a sliding scale is applied to cover the first year of commercial production, the period of repayment of the debt, and the final period after the debt has been repaid.

- Employer Labour Taxes

  The employer is subject to; a payroll tax for expatriates and nationals, and employer; retirement, family, and worker contribution/compensation payments. Said taxes are built into each employee's total cost to company (TCTC).

- Business Tax (Patente)

  Exemption during first 'three years' after 'Production', then 15% payable on the calculated annual rental value of plant and buildings (Rental value is a determined as a function of the gross capital value of fixed assets over a defined term).

- CI Training and Capacity Building

  As per Article 135 of the 2014 Mining Code, an annual; payment of XOF 25 M is payable to assist in building in-country institutional capacity (Endeavour, 2022).

- Corporate Income Tax

  As per the NMC, corporate income tax is set at 25%.

1.3.9 Permitting and Compliance

SML have the required permits to start developing the Project on PE 58. Further the QP is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform the proposed work programme on the Properties held by the Issuer.

Notwithstanding this, it is notable that:

- The third, exceptional renewal Permit for PR 329 is outstanding.

[5] Production equipment only. Non-production equipment (i.e., light vehicles, buses etc, are not exempt)

[6] Excludes fuels used in buses and light vehicles.

- Whilst the key relevant Permits are in place for starting to develop a project on PE 58, the signing of the 'Mining Convention' is significantly outside of the required timelines[7] as defined in the CI 2014 Mining Code. This goes to defining the tax/derogation basis for SML and its contractors.
- The award of PE 58 was based on a pre-feasibility mine plan/production schedule, and as per Article nine of Decree n° 2021-538 of September 2021 granting PE 58 to LMCI, the Issuer needs to notify the Minister of Mines, Petroleum and Energy, that the plan is now different to that proposed.
- The approved ESIA and MRCP was based on the pre-feasibility study results. Whilst the DFS is not significantly different, as per Article 6 of the Environmental authorisation No. 00044/MINEDD/ANDE, dated 18 February 2021, ANDE must be notified accordingly of scope changes to the original ESIA.
- The Mine Closure and Rehabilitation Bond Basis needs to be updated/finalised. Further the:
  - escrow account is to be opened within 20 days following first commercial production; and
  - bank guarantee to be put in place within 120 days from date of first commercial production.
- A Permitting/agreement/stakeholder/notification register is under development for the construction, operational and closure phases of the Project's/Mine's life cycle. Until such time as this is completed and aligned to the construction and operations schedule, it is not possible to say with certainty, that all relevant permits will be in place in time. Notwithstanding this, there is likely sufficient time to address if acted upon expediently.

1.4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

1.4.1 Overview

The Lafigue Mining Licence (PE 58) lies approximately:

- 330 km north-northwest of the port city of Abidjan (approximately 470 km by road), the economic capital of CI and 175 km north-northeast of the political capital of CI, Yamoussoukro (approximately 230 km by road);
- 55 km east of a transnational infrastructure (road, rail, power and fibre) corridor that connects the port of Abidjan, with Burkina Faso to the North; and,
- 80 km north-northeast of CI's second largest city Bouake (approximately 130 km by road).

1.4.2 Political and Economic Environment

CI went through a period of instability from 1999 through to 2011. Since 2012, however, the political situation has been stable, and the government of CI has implemented two successful National Development Programmes (PNDs) between 2012 to 2016 and 2016 to 2020. The third PND runs from 2021 to 2025. These programmes have generated significant Foreign Direct Investment (FDI), both private and intergovernmental, which has contributed to a significant expansion and upgrade of CI's infrastructure, that the Lafigue Project will leverage off.

As a result of the FDI in the local oil/gas industry, agricultural exports and the nascent mining industry, CI has had one of West Africa's highest GDP growth rates since 2012. Further, Moody's affirmed Côte d'Ivoire's Ba3 rating; and changed its outlook to positive from stable and stated that the real GDP growth rate, is likely to be of the order of 7 %/a from 2022 to 2025 (Moody's, 2022).

[7] Should have been signed by 15 December 2021.

CI is also one of the 15 members of the Economic Community of West African States (ECOWAS). This union seeks to create a single economic block and develop infrastructure (road, rail, power and communications) and policies that are to the benefit of all stakeholders.

1.4.3 Country Population Demographics

The population of CI is estimated to be 27.0 M persons, with a population growth rate of 2.5 %/a. The median age of the population is low at 18.9 years, with 51% of the population living in urban areas. Literacy as of the last published population census (2014) was low at 40%.

1.4.4 Local Topography, Elevation and Vegetation

There are low lying hills to the north of PE 58, where the elevation increases to just above 400 mamsl and on PE 58, the elevation varies from approximately 310 mamsl in the northeast corner of the license, to approximately 230 m in the southeast corner. In the area of the proposed pit, the elevation increases to approximately 400 mamsl.

The soils generally located on the lower and high plateaus are iron, aluminium rich ferraltic soils, where mineral alteration is complete. Mineral and organic hydromorphic soils are generally found in the vicinity of streams and marshy areas.

The site is the domain of clear forests and savannas (wooded, shrubby and grassy savannas), with gallery and riparian forests typically running along the seasonal water courses.

1.4.5 Climate

PE 58 falls within the central climate zone of CI, one of three distinct climatic areas within CI. Temperatures are hot year-round, with a mean average annual temperature of 27 °C, with average monthly lows and highs of 19 °C and 36 °C respectively.

The mine will use water harvested from a non-perennial water course that runs diagonally across PE 58. The filling of the water harvest dam and the recharge of ground water is highly dependent on the wet season rains, which peak in September, but are greater than 100 mm/month from April to October. The design average annual rainfall for the Project area is 1119 mm/a, with minimum and maximum design rainfalls of (583 and 1772) mm/a respectively. The mean annual evaporation rate is estimated to be 1373 mm/a.

Winds, are predominantly from the southwest, averaging a wind run speed of approximately 2 m/s over the year. The site is not subject to extreme wind events and is not suitable for wind turbines.

The potential for solar generated power at PE 58 or in the surrounding area is 'good'[8], but not 'excellent'[9].

1.4.6 Seismicity

PE 58 is in an area of very low seismic risk, which means that design costs for plant and facilities are not adversely affected by this factor.

[8] P_{50} = 3.93 kWh/kWp

[9] Values above 4.5 kWh/kWp are considered excellent**Invalid source specified.**

1.4.7 Project/Mine Enabling Infrastructure and Services

The project, being along the A3 infrastructure corridor between the Port of Abidjan and Burkina Faso, has excellent enabling infrastructure in close proximity to PE 58, as described more fully herein.

1.4.7.1 Port

The Lafigue Project/mine will be serviced by the Autonomous Port of Abidjan (APA). The APA is one of the most important ports in West Africa, is a major contributor to the economy of CI, and serves as the primary sea entry point for goods being transported to Burkina Faso, Mali, Niger, Chad, and Guinea. It also serves as a key export point for CI's agricultural and mineral products (manganese ore).

1.4.7.2 Road

The distance by road from the APA/Abidjan to PE 58 is approximately 470 km, of which approximately 453 km is paved. The last 16 km of the route is via a laterite road which whilst public, will likely be maintained by the Issuer. The turn-off from the A3 to PE 58 is at Katiola. The by-pass around the Lafigue village (approximately 2 to 3 km) is funded by the Project.

As part of CI's PND's, the A3 from Abidjan to Burkina Faso is being upgraded and it is likely that by the time the Plant is in operation, 340 km of the route from Abidjan to Bouaké will be via a dual lane highway.

1.4.7.3 Rail

The rail line from Abidjan to Burkina Faso runs through Katiola, and whilst there is a rail station at Katiola, it is primarily used for passengers. While the Lafigue mine can operate without rail, it is recommended that discussions start with SITARAIL and other stakeholders, regarding the possible provision of an expanded service at Katiola (goods, fuel and other).

1.4.7.4 Power

With the PND programmes, CI has upgraded/expanded their transmission infrastructure across the country, to facilitate transnational electricity movement, bringing on new generation capacity (solar and hydro), and increasing power distribution to rural communities.

Whilst there is a 225 km transmission line that runs along the A3 infrastructure corridor from Abidjan to Burkina Faso via Bouake and Katiola, the Lafigue mine will connect to a new 225 km ring main that links Bouake with Ferkessédougou via Dabakala. A new 33 km 225 kV transmission line will be built between Dabakala and the mine by the Issuer. The CI transmission infrastructure has improved in recent years and is considered good/reliable, and no issues are foreseen.

There have been concerns that CI's gas/oil fields would run out within the next 10 or so years, and in CI there has been a strategic focus on expanding hydro and solar generation, and increasing baseload power generation capacity, with a focus on new gas and coal thermal generation capacity. A proposed LNG terminal would facilitate the import of gas, to make-up for any shortfalls in local production. Both the development of the LNG terminal and coal fired power stations are behind schedule, and it is unclear when, and if these will be built. On this basis, it is also unclear whether CI will meet its 2030 strategic generation goals. Further, both hydro and solar have low-capacity factors, with a low seasonal rainfall in 2021, coupled with power plant failures leading to power rationing in CI.

In late 2021, a new gas/oil field discovery was declared, which should extend CI's energy independence for a number of years. However, it is unclear how quickly additional gas generation capacity can be brought online to meet current and projected energy requirements.

Whilst CI exports over 11 % of the electricity generated, additional work is required to understand whether the mine could be affected in future by power rationing and the drivers/factors for electricity price escalation. Since 2016, there has been limited change to electricity pricing. Subject to economics, solar could be considered in CI to achieve the Issuer's business interests, but any installation need not necessarily be located on PE 58.

1.4.7.5 Fuel Supply and Storage

CI has refining capacity in Abidjan, with distribution by road, rail and pipeline. Until such time as a fuel provider is selected, the depot for fuel sourcing is not defined. As policy, the Issuer maintains (15 to 20) days of storage capacity at site to cover supply chain disruptions.

With the newly discovered offshore oil/gas fields, CI will likely stay a net producer of fuel for the foreseeable future.

1.4.7.6 Air

Operations at PE 58 will be serviced the by the international airport in in Abidjan and a newly constructed airfield on PE 58. The airstrip at PE 58 will be used for the movement of both people and gold.

1.4.7.7 Communications

CI has well-developed communication infrastructure (fibre and cellular), which the project will leverage off for both construction and operations. No dedicated satellite links will be required.

1.4.7.8 Location of Towns and Supporting Infrastructure

There are three large regional cities close to PE 58, namely; Bouake (>500 000 people), Katiola (>40 000 people) and Dabakala (>56 000 people), and a number of villages between a few hundred and a few thousand people within a 25 km radius of PE 58, typically located in a region from the north-northwest to the east of PE 58.

Subject to confirmation from the latest CI population census (2021), the population in the immediate area is young[10], with low literacy rates circa 20%.

The economy of the region is largely agrarian in nature, with limited to no heavy industry. Thus, there are likely to be limited technical skills available locally, and few businesses that would support an operating mine. The mine will thus primarily use service providers in Abidjan and abroad to support operations.

As part of the Issuers Social Performance initiatives, the mine will seek to upskill local communities and develop local procurement initiatives.

1.4.8 Land Use and Conditions

From a physiography perspective, PE 58 is considered low risk from a construction and operational perspective, and there is sufficient space on PE 58 for all the required mine infrastructure, including but not limited to: waste rock dumps, tailings facilities, water harvest and storage dams, mine accommodation and plant.

[10] Likely to be slightly older than the CI median age of 19 years (2014)

The highest risk for construction and operations, is the supply of water. Notwithstanding this, the use of a non-perennial water course on PE 58 to harvest water falling in the wet season, supplemented with ground water is sufficient to maintain operations for a 100-year ARI dry and wet season event (Knight Piesold).

1.5 History

The earliest documented exploration at Lafigué dates back to 1935, when ‘Bureau Minier of the France d’Outre-mer’ (Bumiform) conducted geological mapping. It’s successor Bureau de Recherches Géologiques et Minières (BRGM) and Société pour le Développement Minier de la Côte d’Ivoire (SODEMI) conducted airborne geophysical surveys during the late 1960s and early 1970s, before an exploration, development and operating agreement was set up between SODEMI and GENCOR Limited (through its Ivorian subsidiary, GATRO-CI) in 1996.

Through the agreement GATRO-CI completed a series of regional stream sediment and soil geochemistry surveys, exploration pits and trenches, and a small amount of drilling (14 diamond core drillholes and 37 reverse circulation holes), and defined four main targets, including Lafigué.

The wider Lafigué target was later delineated between 1994 and 2002 by exploration works conducted by GATRO-CI, and later by Compagnie Minière Or (COMINOR). Exploration work included; rotary airblast, reverse circulation and diamond core drilling, which demonstrated both lithological and structural controls on mineralisation.

Following a cessation of exploration works during the Côte d'Ivoire civil war, COMINOR recommenced exploration works in 2010.

In 2014, La Mancha Côte d’Ivoire SARL (LMCI) replaced COMINOR in the partnership with SODEMI, leading eventually to a transfer of PR 329 from SODEMI to LMCI in 2020 and the granting of PR 58 to LMCI (and then Société des Mines de Lafigué SA (SML)) in 2021.

LMCI drilling from 2014 onwards, focused on delineating mineralisation in Lafigué North, as well as obtaining structural data to better understand mineralisation controls.

Historical Mineral Resource estimates were produced in 2002 and 2003 (internal use only), followed by publicly reported MREs produced by Endeavour in 2017, 2018, 2019 and 2020, and by SRK Consulting in 2021.

PE 58 has not been mined on a commercial scale, however, there has been significant artisanal mining works primarily targeting the quartz-tourmaline vein-hosted mineralisation. Since September 2021, Endeavour alongside the Dabakala Gendarmes have been undertaking an eviction exercise, whereby the majority of artisanal miners have been removed from demarcated areas within PE 58.

1.6 Geological Setting and Mineralisation

Lafigué is located towards the northern end of the Birimian-age Oumé-Fetekro greenstone belt, a north-south-trending meta-volcano-sedimentary belt comprised primarily of bimodal metavolcanics and clastic metasedimentary rocks.

Lafigué has been interpreted to lie within a compressive relay domain (or transpressive restraining bend), bound by two North-northeast-trending sinistral shear corridors, formed at an angle to regional northwest-southeast directed shortening during the D2 and D3 regional deformation events (Ciancaleoni, 2018). On the deposit scale, gold mineralisation is controlled by a series of east-northeast-trending shear zones dipping at (10 to 40)° to the south-southeast.

Mineralisation is often hosted by quartz-carbonate-tourmaline-pyrite-pyrrhotite-gold veins as well as the associated biotite-tourmaline-sericite-chlorite-carbonate alteration zones, where these veins typically exploit the gently dipping brittle-ductile reverse shear zones. Gold is also hosted within broader zones of altered, stacked shear zones in the hanging wall (and to a lesser degree, the footwall) of the main lithological contacts.

In total, the Lafigué mineralisation spans a strike length of approximately 2 km, trending east-northeast and dipping moderately to the south-southeast to a maximum depth of approximately 440 m below the surface in Lafigué North (approx. down-dip extension of 700 to 900 m). Mineralisation continuity reduces towards Lafigué Centre and to the south and west. The deposit remains open at depth along some parts of its strike length.

1.7 Deposit Type

The Lafigué deposit resembles a typical shear zone-hosted deposit of the West African Paleoproterozoic greenstone terrane (Man-Leo Shield). The deposit is associated with the north-south-trending Oumé-Fetekro greenstone belt, and more specifically, within a Birimian age complex of bimodal metavolcanics and meta-volcanoclastic rocks intruded by a series of felsic intrusions.

1.8 Exploration

Endeavour Mining Corporation commenced exploration activities in 2017, when an airborne vertical tilt-angle derivative ('VTEM') survey was flown across the permit area to better define the regional structures. Drilling targets were further delineated through grab sampling, soil sampling and IP pole-dipole and gradient surveys, mainly conducted between 2017 and 2019. More recent exploration programmes have highlighted six targets which warrant further exploration work across the PR 329 Exploration Permit and PE58 Mining Permit.

1.9 Drilling

Drilling at Lafigué since 2017 has comprised of six separate campaigns aiming to delineate the full down-dip and along-strike extent of mineralisation, as well as increase confidence in the geological and grade continuity through infill drilling. Drilling conducting during Endeavour's ownership of the Project, including reverse circulation and diamond core drilling, has been carried out under the supervision of technically qualified personnel applying standard industry approaches. For all drillholes completed since 2017, collar surveys were conducted using a differential GPS and downhole surveys were completed in each drillhole using a Reflex-EZ track ± EZ-Gyro.

The majority of DD and RC holes at Lafigué have been completed on a 20 to 40 m by 50 m grid, with some areas of closer drilling towards the up-dip portions of the deposit and wider spaced drillholes in down-dip areas. The majority of drillholes dip at 50° or 60° towards an azimuth of either 000° or 335°. Mineralisation typically dips at approximately 20° towards the south/south-southeast, resulting in drilling intersection angles of 90 to 110°. Further drilling is recommended on six exploration targets recently delineated.

1.10 Sample Preparation, Analysis and Security

Field duplicates, blank samples and certified reference materials were inserted into the regular sample stream as part of the QAQC programmes during the 2010 to 2022 drilling campaigns. Overall, it is considered that the majority of sample preparation, analyses and security protocols conformed to industry best practice and do not show any significant indications of contamination or bias in the available sample assay results. The absence of QAQC sample results for the 1997 and 2002 drilling campaigns is noted and as such, assay data from these drilling campaigns present a risk in terms of accuracy and precision of the associated assay grades.

1.11 Data Verification

1.11.1 Geology and Resource

Upon acquisition of the Project in 2016, the issuer implemented a SQL-based database management system ('DBMS'), where all historical data generated from the Fetekro Project was audited during the importation process. No material issues with the historical drillhole data used for the 2022 MRE have been identified. Since 2013 all data acquired across the Fetekro Project area is managed using the built-in data integrity requirements of an industry standard SQL-based DBMS. A total of thirteen 2018 and 2019 drillhole collars were resurveyed by Kouamelan during 2020, with no material discrepancies found. Subsequently, all drillholes completed since 2020 have also had verification collar surveys to confirm their positions. Prior to undertaking the 2022 MRE, Qualified Person, Dr Lucy Roberts and Dr James Davey of SRK verified the position of five drillholes collar locations (from drilling campaigns ranging from 2014 to 2021) during their visit to site between 14 May and 16 May 2021. No twinned drillholes have been completed at Lafigué.

1.11.2 Other

The data verification approaches applied by the other consultants who contributed to this Report, are discussed in Section 12.

1.12 Mineral Processing and Metallurgical Testing

To support the various stages of study development and the economic evaluation of the Project, a series of metallurgical testwork programmes have been conducted on the Lafigué ores through the scoping, pre-feasibility and definitive feasibility study phases. These programmes were designed to evaluate alternative comminution, processing, and gold extraction flowsheets, provide data for design purposes and inputs for estimation of processing costs.

Various comminution options were considered, including; single stage crushing with SAG and ball milling or single stage SAG milling, tertiary crushing with ball milling and HPGR with ball milling. Each of these options was considered with an in-mill gravity concentration circuit. Downstream processing of the gravity tail considered CIL and leach/CIP options as well as thickening options to manage water requirements along with cyanide usage minimisation and control of discharge levels.

The early scouting testwork and subsequent definitive testwork programmes concluded that the ore is free milling with high levels of gravity recoverable gold and high cyanidable gold extraction. Benchmarking of ore breakage characteristics suggested that the fresh ore would be suited to feed preparation using HPGR. A closed circuit HPGR ball milling including primary crushing, secondary crushing, and CIL was selected as the process route for the feasibility study based on lowest comminution energy requirements, resulting in the highest NPV and IRR compared to alternative comminution approaches. This was appropriate given the low fraction of oxide/transition ore (<6% of the mineable reserves) and the extreme competency observed in the fresh ore samples. A leach/CIP circuit was not justified given the additional capital and operating costs with the low leach feed grades following gravity recovery.

During the definitive feasibility study, considerable emphasis was placed on demonstrating HPGR performance for the range of ore host lithology types including waste dilution with fresh ore and the impact of a range of variables such as moisture content, roll speed and pressing force on the HPGR performance. Testwork demonstrated equivalent HPGR performance over a wide range of conditions such that feed ore blending, beyond limiting oxide/transition content to a maximum of 30% of total ore feed, will not be required. The oxide/transition upper blend limit (restricted by low ore competency and higher fines content) will also limit the feed ore moisture to <7%, which has previously been established as limiting in terms of feed slippage reducing throughput and accelerating wear rates.

Further testing of deeper ores from the resource extension drilling programme, demonstrated near identical metallurgical performance/recovery across the major host lithologies and confirmed the high gold recovery and low reagent consumption characteristics of the fresh ores. Localised variability in gold head assays and mineralogy was observed. Although this made direct comparison between tests less conclusive, there was little impact on overall gold extractions or final residue grades. Gold extractions were consistently high across the range of 80 variability samples tested with three slower leaching examples. The mineralogy that caused slow leaching appears to be diluted out in the composite samples, with more consistently high gold extractions from the composite gravity tails cyanidation tests. This phase of testing included additional leach optimisation testing to improve dissolution kinetics and overall gold extraction.

A P_{80} grind size of 106 µm was selected as the economic optimum with leach cyanide addition of 0.035% w/v degrading to 0.03% w/v at a starting pH of 10.5. Efficient removal of the coarse gravity recoverable gold was demonstrated to be essential, with very slow whole of ore leach characteristics and lower overall extractions at even extended leach times. Leaching will be conducted with air sparging as the oxygen demand is very low and little improvement in kinetics was achieved with oxygen. Occasional slower leaching variability examples justified an extended CIL residence time of 36 hours for fresh ore. Metallurgical gold recovery and leach reagent consumption estimates were based on the median results from the variability samples leach tests and include an allowance for soluble gold loss.

Table 1-2: Lafigué Metallurgical Recovery and Reagent Consumption

Composite	Gold Recovery (%)	NaCN Consumption kg/t	Lime Consumption kg/t
Oxide	96.9	0.17	2.85
Transition	96.5	0.17	0.85
Fresh	96.4	0.17	0.32

Table 1-2 notes:

- Lime consumption based on hydrated lime.
- Transition ores were not specifically tested, but the processing requirements are assumed to be the interpolated values shown.

1.13 Mineral Resource Estimate

SRK Consulting (UK) Ltd (SRK) produced the 2022 Mineral Resource estimate for the Lafigué gold deposit. In doing so, SRK conducted a high-level review of the supporting drillhole database and then produced a simplified lithology model, based on a refined lithology logging field, as well as a weathering model constructed using surfaces based on weathering/material type logging, completed by on site geologists.

SRK selected a nominal modelling cut-off grade of 0.30 g/t Au for the modelling of Au mineralisation, using an indicator interpolant with a probability value of 0.4. The indicator interpolant was guided by a structural trend based on a series of surfaces interpreted to be the primary controls on the geometry and distribution of mineralisation (i.e. lithological contacts and associated shear zones). Additionally, a series of vein wireframes were produced based on interval selections in order to accurately model thinner mineralisation domains towards the west of the deposit, where mineralisation continuity is reduced.

Following the generation of the geological models, SRK carried out the following steps to produce the MRE:

- statistical analysis and definition of domains;
- geostatistical analysis (variography) within estimation domains;
- block modelling and grade interpolation using Leapfrog Edge software;
- model validation;
- Mineral Resource classification;
- consideration of reasonable prospects for eventual economic extraction (RPEEE); and,
- reporting of Mineral Resources.

The density database used by SRK includes a total of 2214 measurements (with logged lithology and weathering attributes) taken between 2014 and 2021. Density determinations were carried out using drillcore samples representing the full range of lithologies and weathering intensities present across the Lafigué ML. The average dry bulk density values applied during the MRE are listed in Table 1-3.

Table 1-3: Average Dry Bulk Density Values used for the MRE

Lithology	Number of Measurements	Mean Value (g/cm^3)
Laterite	1	2.00
Saprolite	9	1.66
Saprock	17	2.51
Fresh - Mafic	1205	2.86
Fresh - Felsic	628	2.72

Note, only those density samples with lithology and weathering logging were used in the calculation of average densities for the tonnage model

The 2022 Mineral Resource statement for the Lafigué gold deposit is shown in Table 1-4 following.

Table 1-4: Mineral Resource Statement for the Lafigué Gold Project, Effective Of 15 May 2022*

Classification	Material Type	Tonnes Mt	Au Grade g/t	Contained Metal (Au) koz
Indicated	Oxide	0.7	1.55	36
	Transition	1.7	1.71	94
	Fresh	43.8	2.06	2896
	Total	**46.3**	**2.03**	**3027**
Inferred	Oxide	0.1	1.22	4
	Transition	0.1	2.05	4
	Fresh	1.4	2.11	94
	Total	**1.5**	**2.05**	**102**

*In reporting the Mineral Resource Statement, SRK notes the following:

- The reported Mineral Resources are depleted to a drone survey provided to SRK by Endeavour. The survey was conducted on 17 August 2021 and only accounts for artisanal open pit development at surface. SRK understands that there were further artisanal mining workings underground, but these could not be captured by the drone survey. To account for this, outside of (and below, where necessary) the artisanal open pit workings, to a depth of 5 m below the pre-mining topography, the grades have been reduced to zero. In the absence of any underground survey, and to reflect the uncertainty for these areas, SRK has not depleted the tonnages.
- Since September 2021, Endeavour have been undertaking an eviction exercise whereby the artisanal miners are being removed from site. Endeavour have stated in correspondence with SRK that as of late January 2022, 98% of the artisanal miners were removed. In the absence of an updated survey and groundworks completed at site, SRK highlights the risk associated with more extensive depletion due to ongoing artisanal mining activity in the intervening period and or more extensive workings in the prior period, than is accounted for in this Mineral Resource Statement. A sensitivity analysis is provided in the accompanying report to inform the reader of the associated risks.
- The reported Mineral Resources have an 'Effective Date' of 15 May 2022. The Competent Person for the declaration of Mineral Resources is Dr Lucy Roberts, MAusIMM(CP), of SRK Consulting (UK) Ltd. The Mineral Resource estimate was authored by Dr James Davey, also of SRK;
- Technical and economic assumptions were agreed between SRK and LMCI/Endeavour for mining factors (mining and selling costs, mining recovery and dilution, pit slope angles) and processing factors (gold recovery, processing costs), which were used to run a pit optimisation exercise. These factors were developed as part of the ongoing Feasibility Study for the Lafigué project, as stated below:
  - Mining cost: 2.12 (USD/tore)
  - Waste mining cost: 2.65 (USD/trock)
  - Processing cost: Oxide/Transition: 7.47 (USD/tore); Fresh: 9.13 (USD/tore)
  - Selling cost: 71.8 (USD/oz Au)
  - Mining recovery: 98%
  - Mining dilution: 9%
  - Processing recovery: Oxide = 94.87%; Transition = 94.92%; Fresh = 95.08%
  - Average slope angles: (33 to 51)°, dependent on geotechnical domain
  - G&A cost: 5.60 (USD/tore)
  - Discount rate: 5%
- SRK considers there to be reasonable prospects for eventual economic extraction by constraining the Mineral Resources within an optimised open pit shell using a gold price of USD 1500/oz.
- Mineral Resources are reported within the optimised pit shell using cut-off of grades of 0.4 g/t Au (oxide); 0.5 g/t Au (transition) and 0.5 g/t Au (fresh), which are the marginal cut-off grades for CIL processing determined during the pit optimisation.
- Mineral Resources are reported as in situ and undiluted, with no mining recovery applied in the Statement. All tonnages are reported on a dry basis.
- Mineral Resources are not Ore Reserves and do not have demonstrated economic viability, nor have any mining modifying factors been applied;
- Tonnages are reported in metric units, grades in grams per tonne (g/t), and the contained metal in kilo troy ounces. Tonnages, grades, and contained metal totals are rounded appropriately. 1 troy ounce is assumed to be the equivalent of 31.1034 g
- Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content. Where these occur, SRK does not consider these to be material.

1.14 Mineral Reserves Estimates

1.14.1 Introduction

Section 4 discusses the process followed to derive the Mineral Reserves for the Lafigué Project (the 'Project') in accordance with the Canadian National Instrument 43-101, and adhering to the CIM Definition Standards guidelines (CIM, 2014). The Section will specifically focus on the following essential items:

- Mineral Reserve estimation approach and methodology;
- hydrological conditions;
- mining geotechnics;
- pit optimisation;
- mine designs;
- waste rock dump design;
- Mineral Reserve statements;
- mine production schedule;
- mining strategy;
- mining cost; and
- opportunities and risks.

1.14.2 Mineral Reserve Estimation Approach

As reported herein, the Mineral Reserve statement for the Project is supported by the engineering designs and modifying factors discussed here and detailed in Section 4. A site layout in Figure 1-2 shows the location of the pit relative to the process plant, run of mine ('RoM') stockpiles and waste rock dumps ('WRD').

The open pit is designed with various stage pushbacks, with smaller pits southeast of the Main Pit. The life of mine plan ('LoMp') for Project includes the following key data inputs and activities:

- Resource block model modified to generate the mining block model through re-blocking, which introduces a degree of dilution.
- The pre-mining topographic surface.
- Open Pit optimisation analysis to include:
  - derivation of pit optimisation parameters, among other things: dilution, diluting grades and losses, metallurgical recovery and refining factors, commodity price and operating expenditure assumptions;
  - for unit mining operating expenditures, mining costs were derived from the final financial analysis done during the Pre-feasibility Study ('PFS') (Snowden, 2021) based on actual cost data from similar operations provided by Endeavour Mining Plc ('Endeavour'). Costs include both waste and ore mining costs relationships with reduced level ('RL') elevations coded into the block models as a specific attribute;
  - derivation of marginal economic ore cut-off grades ('COG;) as appropriate; and
  - ultimate pit and staged pit selections.

- Engineering pit design assumptions inclusive of:
  - open pit access, including haul road designs;
  - geotechnical design considerations for pit slopes (inter ramp angles, overall slope angles, batter angles, stacked berm configurations and berm widths), which vary as appropriate with azimuth and depth to reflect the geotechnical domains as established for each deposit;
  - mine planning and production scheduling inclusive of production rates; stockpiling strategies; grade bin selection criteria; production capacities for mining; and processing activities; and
  - Mineral Reserve reporting is based on aggregating all Measured and Indicated Mineral Resource blocks incorporated within the LoMp and reported inclusive of all appropriate dilution, diluted grade, and losses to enable the reporting of Mineral Reserves.

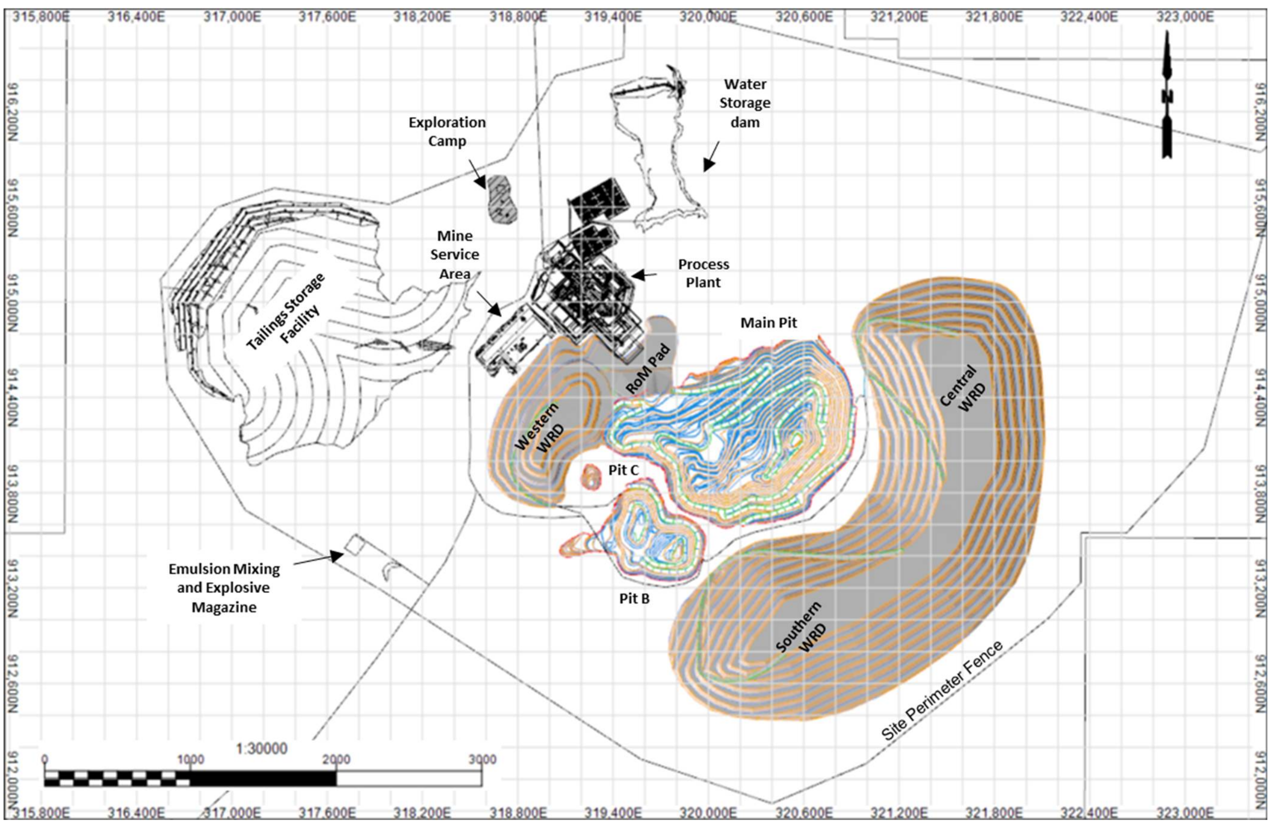

Figure 1-2: Project Site Layout (SRK,2022)

1.14.3 Open Pit Geotechnical Design Criteria

The pit geotechnical design criteria illustrated in Table 1-5 and Figure 1-3 were incorporated during the pit optimisation and design process. Geotechnical data was limited to within the boundaries of the PFS pit design, and only the Oxide zone for the smaller satellite pits. With the updated Mineral Resource and the subsequent extension of the indicated mineral Resource, the updated pit optimisations extended outside the PFS boundary to constrain the Geotechnical zones provided.

Table 1-5: Bastion Geotechnical Pit Geotechnical Design Criteria

Design Sector	Units	Oxide	DS2	DS3	DS4	DS5	DS6	DS7	DS8	DS9
Batter Face Angle	(°)	60.0	55.0	85.0	85.0	85.0	55.0	85.0	85.0	85.0
Batter Height	(m)	10.0	10.0	20.0	20.0	20.0	20.0	20.0	20.0	20.0
Berm Width	(m)	6.0	6.0	12.3	12.3	13.9	11.6	13.9	13.9	13.9
Inter-Ramp Angle (IRA)	(°)	40.3	40.3	55.0	55.0	52.0	38.0	52.0	52.0	52.0
Bench Stack Height	(m)	50.0	50.0	100.0	100.0	100.0	50.0		100.0	100.0
Decoupling Berm Width	(m)	50.0	50.0	100.0	100.0	100.0	50.0	100.0	100.0	100.0
Overall Slope Angle (OSA)	(m)	33.0	33.0	51.0	51.0	48.0	35.0	48.0	48.0	48.0

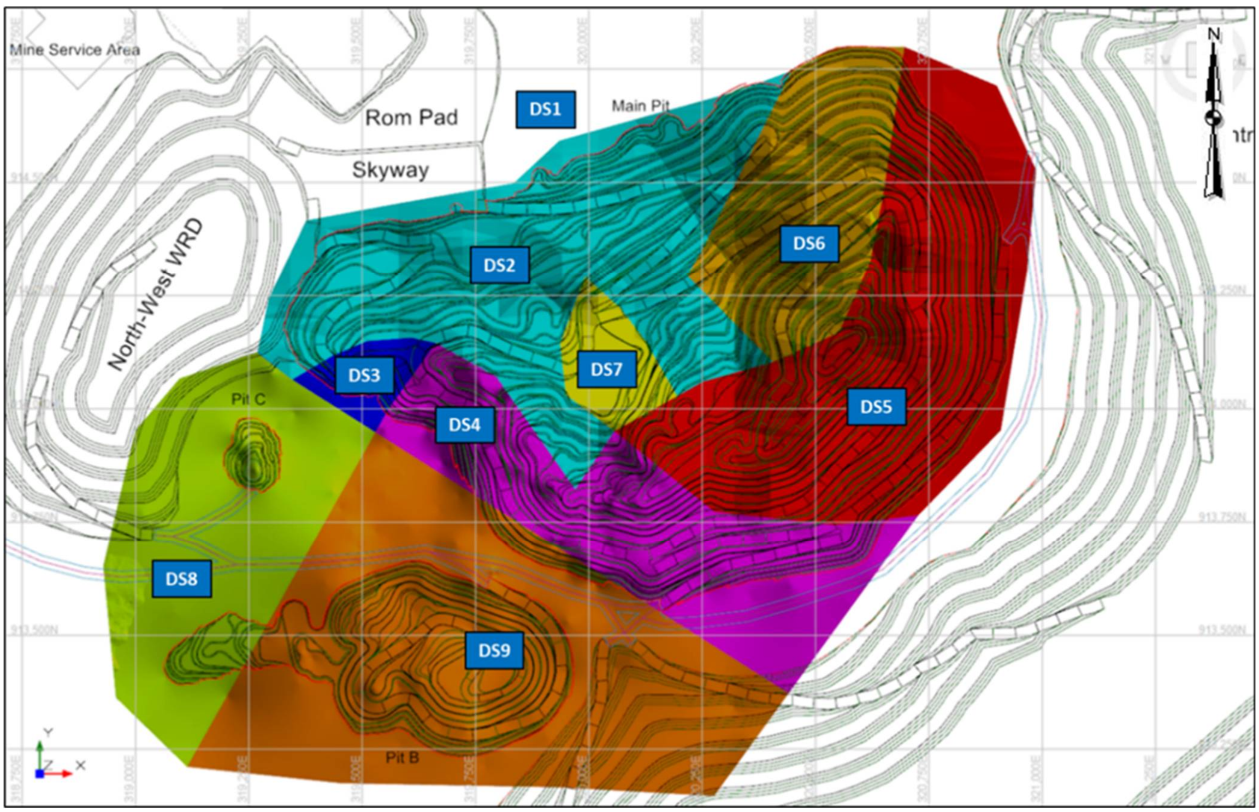

Figure 1-3: Bastion Geotechnical Pit Geotechnical Design Sectors (DS) (SRK,2022)

1.14.4 Mineral Reserve Assumption

1.14.4.1 Block Model

The resource block model was sub-blocked with the parent block size and smallest sub-block sizes 20 m x 20 m x 10 m and 2.5 m x 2.5 m x 1.25 m along the X-direction, Y-direction, and Z-direction, respectively. The resource model was re-blocked to a selective mining unit ('SMU'), size 5 m x 5 m x 2.5 m along the X-direction, Y-direction, and Z-direction respectively, to create regularised mine planning models. The re-blocking process is seen as an industry best practice and one of the most elegant ways of applying proper dilution to the resource models containing too small a block size to be mined selectively. The SMU size was based on ore loading units corresponding in size to a PC2000 and smaller. Therefore, if larger excavators are to be used, the SMU size will need to increase.

The effects of the re-blocking process for Lafigué deposit as percentages of change on the Mineral Reserve ore tonnes, gold content and grade above the set cut-off grades through conversion of the resource model with sub-blocks to a regularised re-blocked model are shown in Table 1-6. In the estimation of the Mineral Reserves, the cut-off grades vary by weathering type 0.4 g/t Au for oxide and transition and 0.5 g/t Au and fresh ore-forming the majority of the deposit.

On the basis that that the Lafigué deposit has not been mined historically, an additional 5% dilution was added on a block-by-block basis. The additional factor is applied in addition to the modifying factors incurred during the regularisation process. The 5% dilution was only applied during scheduling and not incorporated during the pit optimisation.

Mineralogy and by association higher and lower gold grades in the quartz vein versus rock/shear-hosted mineralisation, may be visually discernible at the Lafigué deposit. However, the relationship did not prove to be consistent within the drill core. In addition, the ore/waste contacts will be challenging to visually distinguish, as the deposit incorporates diffuse packages of mineralisation where the grade slowly drops off. Good grade and ore control practices will be required that is supported by internal and external training, on dig polygon design, and adherence to acceptable dilution levels.

Table 1-6: Total applied Modifying Factors, including additional block-by-block dilution

Cut-Off	(g/t Au)	0.4	0.5	0.6	0.8	1.0
Dilution	(%)	112.7	110.8	109.3	107.8	107.1
Recovery	(%)	93.8	92.6	91.5	90.0	89.0
Or						
Tonnes	(%)	12.7	10.8	9.3	7.8	7.1
Grade	(%)	-11.3	-9.8	-8.5	-7.2	-6.6
Content	(%)	-6.2	-7.4	-8.5	-10.0	-11.0

Total applied Modifying Factors on 5 m x5 m x 2.5 m (x,y,z) model (Unconstrained)

1.14.4.2 Optimisation Input Parameters

Table 1-7, following, summarises the optimisation input parameters. For the derivation of cut-off grades and supporting the economic viability of the Mineral Reserves, a gold price of USD 1300/oz was used. The other factors that derive the net sales revenue are the individual government royalty of 4% with an additional community levy of 0.5%, and a transport, vaulting and refining cost of USD 4.0/oz payable. This results in a net gold price of USD 1237/oz payable (USD 39.79/g), and a total selling cost of USD 62.5/oz.

Table 1-7: Pit Optimisation Input Assumptions Summary

Parameters	Units	Value	Source/Basis
Production Rate - Ore	(Mt/a db)	4.00	
Geotechnical			
• Max Surface Elevation	(Z Elevation)	420	
• Laterite & Saprolite (Oxide) & Transition		IRA / OSA	
• DS 1	(Deg)	40.3 / 33.0	END-011-GD DRAFT Report.pdf
• Fresh		IRA/OSA	
• Footwall Zone DS2	(Deg)	35.0/33.0	END-011-GD DRAFT Report.pdf
• Footwall Zone DS6	(Deg)	38.0/35.0	
• Footwall Zone DS7	(Deg)	52.0/48.0	
• Hanging Wall Zone DS3	(Deg)	55.0/51.0	
• Hanging Wall Zone DS4	(Deg)	55.0/51.0	
• Hanging Wall Zone DS5	(Deg)	52.0/48.0	
• Hanging Wall Zone DS8	(Deg)	52.0/48.0	Inputs extended from Zone 5
• Hanging Wall Zone DS9	(Deg)	52.0/48.0	Inputs extended from Zone 5
Mining Factors			
• Dilution	(%)	107%	From Regularisation 5x5x2.5
• Recovery	(%)	99%	
Processing			
• Recovery - Au (oxide)	(%)	94.87%	Lycopodium 2021
• Recovery - Au (transition)	(%)	94.92%	
• Recovery - Au (fresh)	(%)	95.08%	
Operating Costs			
• Wavg Waste Mining Cost	(USD/t_{rock})	2.65	PFS Financial Model
• Incremental Mining Cost	(USD/m bench)	0.0022	
• Reference Level	(Z Elevation)	350.00	
• Wavg Ore/Waste Differential	(USD/t_{ore})	0.81	
• Grade Control	(USD/t_{ore})	1.31	
• Rehandle Cost	(USD/t_{ore})	0.37	
• Off RoM Rehandle	(USD/t_{ore})	0.79	
• Rehabilitation Cost	(USD/t_{rock})	0.06	

Table 1-7: Pit Optimisation Input Assumptions Summary

Parameters	Units	Value	Source/Basis
• CIL - Oxide	(USD/t_{ore})	7.47	Lycopodium 2021
• CIL - Transition	(USD/t_{ore})	7.47	
• CIL - Fresh	(USD/t_{ore})	9.13	
• G&A	(USD/t_{ore})	5.60	PFS Financial Model
• Sustaining Capital (SIB)	(USD/t_{ore})	1.87	PFS Financial Model
• Selling Cost Au (@USD 1300/oz)	(USD/oz)	62.5	EDV 2021 Assumptions
Metal Price			
• Gold	(USD/oz)	1300	EDV 2021 Assumptions
• Gold	(USD/g)	41.80	
Discount Rate	(%)	5%	EDV 2021 Assumptions

1.14.5 Cut-off Grade Analysis

The cut-off grades for the economic and Marginal ore by deposit and weathering types are given in Table 1-8, following. The cut-off grade analysis completed to support the end-2020 Mineral Reserve statements incorporated the various assumptions reported in Table 1-7, inclusive of:

- processing operating expenditures;
- general and administration operating expenditures are assumed at 100% for the economic cut-off grade determination and 60% for the Marginal-Ore ('MO') grade category;
- deposit-specific metallurgical recovery assumptions distinguishing between oxide, transitional and fresh ore;
- transportation, vaulting, and refining charges expressed in USD/oz;
- long-term gold price assumption of USD 1300/oz for reserves; and
- ad-valorem (or proportional) taxes paid on gross revenues from sales after deductions for transport (FOB) and refining and/or smelting costs, comprising:
  - a 4% gold royalty; and
  - a 0.5% community levy.

Table 1-8: CoG for LG and MO by Weathering Type

Description	Units	Value
Economic Operating Costs (Oxide)	(USD/t_{ore})	16.78
Economic Operating Costs (Transition)	(USD/t_{ore})	16.71
Economic Operating Costs (Fresh)	(USD/t_{ore})	16.86
Low Grade (LG) Cut Off Grade		
• Economic Cut-Off Grade (Oxide)	(g/t Au)	0.4
• Economic Cut-Off Grade (Transition)	(g/t Au)	0.4
• Economic Cut-Off Grade (Fresh)	(g/t Au)	0.5
• In Situ Economic Cut-Off Grade (Oxide)	(g/t Au)	0.5
• In Situ Economic Cut-Off Grade (Transition)	(g/t Au)	0.5
• In Situ Economic Cut-Off Grade (Fresh)	(g/t Au)	0.6
Marginally Economic Ore (MO) Cut-off Grade		
• Marginal Cut-Off Grade (Oxide)	(g/t Au)	0.4
• Marginal Cut-Off Grade (Transition)	(g/t Au)	0.4
• Marginal Cut-Off Grade (Fresh)	(g/t Au)	0.4

1.14.6 Pit Optimisation Results

The objective of the open pit optimisation process is to determine a generalised open-pit shape (shell) that provides the highest economic value for a deposit. Pit optimisations were carried out using the Whittle Four-X (Whittle) pit optimisation software. Whittle software is considered the leader and widely used in the mining industry for open pit optimisation and consequently was selected for use in the Lafigué DFS.

The final pit design defines the ore reserve, and subsequently, the LoM production schedule/cashflows. Hence, pit optimisation is the first step in developing any LoM plan. In addition to defining the ultimate size of the open pit, the pit optimisation process also indicates possible mining pushbacks. These intermediate mining stages allow the pit to be developed practically and incrementally while at the same time targeting high-grade ore and deferring waste stripping.

Two optimisation scenarios have been run and will be discussed in this section:

- Measured and indicated classified material ('MI') case. Inferred classified material is treated as waste.

The MI case pit optimisation has been run based solely on the Measured and Indicated classified material to define the optimal pit shell and inventory and supports the estimation of Ore Reserves.

- Measured, Indicated, and Inferred classified material treated as ore ('MIF') case.

The MIF case pit optimisation has been used to evaluate if there is any inferred material in the near-term that might add value or be sterilised by the current MI pit. The MIF case also assists in better understanding the long-term potential.

1.14.6.1 Measured, Indicated (MI) Pit Optimisation Results

The calculation of a Whittle optimisation NPV, the usual criteria for selecting an optimal pit, largely depends on the discount rate and the high-level scheduling methodology applied in Whittle. Whittle then produces nested pit shells with a relative undiscounted cash flow ('CF') and discounted cash flow ('DCF') for each nested pit. Three relative DCFs are presented based on three different scheduling methodologies applied by Whittle:

- Best: The best cash flow is achieved when the nested pit shells are mined in sequence. Although optimal for cash flow, such a sequence is mostly impractical since nested pit shells are often closely layered (like the layers of an onion) and would imply that thin pushbacks could be mined.
- Worst: The worst cash flow is achieved when the selected final pit shell is mined from top to bottom without consideration for nested pit shells or pushbacks. This approach would undoubtedly be practical but usually presents the lowest economic scheduling option.
- Specified: The specified case lies somewhere between the best and the worst. The user selects which cutbacks to mine to gain the advantages of higher cash flow of the Best Case, whilst still being practical and feasible.

Figure 1-4 shows the pit-by-pit metal price sensitivity results from the MI case pit optimisation. Detailed pit optimisation results are provided in Table 1-9, whilst Table 1-10 summarises the selected pit shell results for each pushback and a USD 1300/oz shell. The Reserve Case pit optimisation has been used to determine the final pit extents for the LoM plan.

The ultimate pit shell selection was driven by maximising the pit inventory while considering the economic viability of the open pit, which considers the NPV in conjunction with the overall stripping ratio.

- The discounted open-pit value for each nested pit shows that the 'Best' and 'Specified' scheduling options, start to plateau at USD 1100/oz.
- The 'Specified' scheduling options trend is slightly lower than the 'Best' case, but significantly better than the 'Worst' case. This result indicates that Lafigué will benefit from a pushback-phased mining approach, but this diminishes after a USD 950/oz pit shell, where the 'Specified' case decreases relative to the 'Best' case.
- There are two distinct step changes at USD 725/oz and USD 875/oz, with minor step changes between USD 900/oz and USD 1000/oz. The pit starts plateauing between USD 1150/oz and USD 1300/oz.
- Substantial step changes within tiny gold price margins indicate that concentrated higher-grade zones drive the incremental pits generated by Whittle. This is better represented in Table 1-6.
- Beyond USD 1150/oz on the graph, the Specified scheduling options show a stable but steady decreasing discounted cash flow to USD 1600/oz.

The USD 1175/oz pit shell was chosen as the final optimal pit shell for scheduling, which should deliver 44.6 Mt of ore (diluted) at a Whittle stripping ratio (excluding ramps and bench geometry) of 9.1 (t:t), and a Whittle cut-off grade 0.53 g/t Au for fresh ore. The total metal within the final pit shell is 2.66 Moz. The USD 1175/oz pit shell has a slightly higher specified NPV (USD 7M) compared to USD 1300/oz, but has 100 koz less gold.

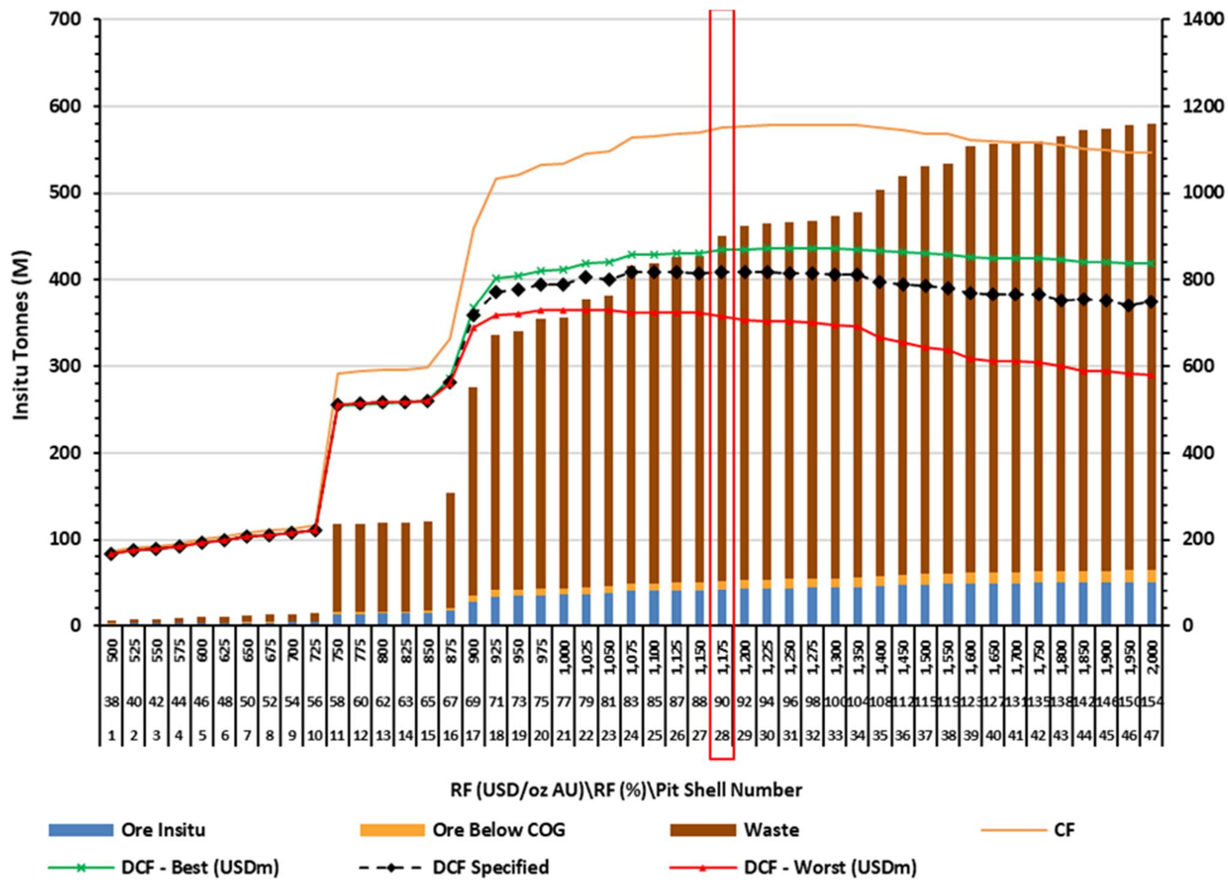

Figure 1-4: Reserve Case (MI) Metal Price Sensitivity

Table 1-9: Reserve Case (MI) Pit-by-Pit Results

Gold Price USD/oz	Total Rock (Mt)	Waste (Mt)	Ore (Mt)	Grade (g/t Au)	Metal (Moz Au)	SR (t:t)	CF (USD M)	DCF – Best (USD M)	DCF – Worst (USD M)	DCF – Specified (USDM)	Rec Metal ((oz Au)	Cash Cost (USD/oz)
600	10.13	6.65	3.48	2.15	0.24	1.91	202.44	194.32	194.32	194.32	0.23	416.71
625	10.54	6.86	3.67	2.10	0.25	1.87	206.77	198.04	198.04	198.04	0.24	424.22
650	12.56	8.64	3.92	2.10	0.26	2.21	217.09	207.36	207.36	207.36	0.25	438.52
675	13.03	8.95	4.07	2.07	0.27	2.20	220.67	210.39	210.39	210.39	0.26	444.49
700	13.93	9.65	4.28	2.04	0.28	2.26	225.90	215.08	215.07	215.07	0.27	453.41
725	15.25	10.62	4.63	1.98	0.29	2.30	233.07	221.86	221.79	221.79	0.28	467.97
750	118.05	103.73	14.32	2.08	0.96	7.25	582.11	508.29	510.72	510.72	0.91	662.34
775	119.39	104.85	14.54	2.07	0.97	7.21	587.33	511.92	514.43	514.43	0.92	663.71
800	120.17	105.40	14.77	2.06	0.98	7.13	591.10	514.50	516.79	516.79	0.93	665.13
825	120.59	105.72	14.88	2.05	0.98	7.10	592.82	515.92	517.84	517.84	0.93	665.81
850	121.65	106.59	15.06	2.04	0.99	7.08	596.05	518.57	519.77	519.77	0.94	667.25
875	154.19	136.21	17.98	2.01	1.16	7.58	663.85	572.32	560.93	562.76	1.11	699.88

Table 1-9: Reserve Case (MI) Pit-by-Pit Results

Gold Price USD/oz	Total Rock (Mt)	Waste (Mt)	Ore (Mt)	Grade (g/t Au)	Metal (Moz Au)	SR (t:t)	CF (USD M)	DCF – Best (USD M)	DCF – Worst (USD M)	DCF – Specified (USDM)	Rec Metal ((oz Au)	Cash Cost (USD/oz)
900	276.64	247.24	29.40	1.93	1.83	8.41	918.44	735.20	690.23	718.73	1.74	772.91
925	337.45	302.24	35.22	1.90	2.15	8.58	1032.63	803.63	718.00	773.37	2.05	795.77
950	342.24	306.39	35.86	1.89	2.18	8.54	1042.09	808.73	720.89	777.22	2.08	797.89
975	356.02	318.92	37.11	1.89	2.25	8.59	1064.36	821.86	728.93	789.06	2.14	803.32
1000	357.41	320.05	37.36	1.88	2.26	8.57	1066.96	823.40	729.67	788.62	2.15	804.09
1025	379.88	341.40	38.48	1.90	2.35	8.87	1091.01	837.57	727.87	805.50	2.24	811.94
1050	383.94	344.70	39.24	1.88	2.37	8.78	1096.92	840.82	728.42	800.41	2.26	814.58
1075	418.34	376.46	41.88	1.87	2.52	8.99	1129.10	857.80	724.40	817.02	2.40	829.63
1100	420.95	378.72	42.23	1.87	2.53	8.97	1131.75	859.13	724.30	817.28	2.41	831.09
1125	428.08	385.23	42.85	1.86	2.56	8.99	1137.08	861.82	723.34	818.08	2.44	834.53
1150	429.40	386.33	43.07	1.86	2.57	8.97	1138.17	862.28	723.11	816.37	2.45	835.39
1175	452.35	407.71	44.64	1.85	2.66	9.13	1150.00	868.34	716.19	819.50	2.53	846.17
1200	464.60	418.87	45.73	1.84	2.71	9.16	1154.96	870.72	705.61	819.05	2.58	852.21
1225	467.54	421.46	46.09	1.84	2.72	9.15	1155.96	871.17	703.48	817.54	2.59	853.96
1250	469.21	422.92	46.30	1.83	2.73	9.13	1156.34	871.34	702.22	814.63	2.60	855.03
1275	469.75	423.32	46.43	1.83	2.73	9.12	1156.46	871.35	701.56	815.18	2.60	855.53
1300	475.70	428.62	47.08	1.82	2.76	9.10	1156.64	871.29	694.92	812.45	2.62	859.26
1350	479.86	432.36	47.50	1.81	2.77	9.10	1156.19	870.92	691.41	811.03	2.64	861.91
1400	505.62	456.49	49.12	1.80	2.85	9.29	1149.56	867.05	667.45	796.05	2.71	876.47
1450	521.65	471.68	49.97	1.80	2.90	9.44	1143.55	863.82	653.76	790.33	2.76	885.35
1500	533.29	482.57	50.73	1.80	2.93	9.51	1137.60	860.67	641.78	786.01	2.79	892.27
1550	536.32	485.33	50.99	1.79	2.94	9.52	1135.61	859.60	637.29	781.80	2.80	894.26
1600	556.71	504.74	51.97	1.79	2.99	9.71	1121.18	852.05	616.27	770.38	2.85	906.29
1650	559.71	507.47	52.24	1.79	3.00	9.71	1118.30	850.58	611.76	767.82	2.86	908.43
1700	560.86	508.53	52.33	1.78	3.00	9.72	1117.04	849.95	610.34	766.46	2.86	909.27
1750	562.63	510.16	52.47	1.78	3.01	9.72	1114.96	848.91	607.70	765.44	2.86	910.57
1800	568.03	515.27	52.76	1.78	3.02	9.77	1109.22	846.03	600.88	752.94	2.87	914.11
1850	575.01	521.88	53.13	1.78	3.03	9.82	1101.45	842.17	589.88	755.66	2.89	918.63
1900	576.82	523.59	53.23	1.77	3.04	9.84	1099.03	841.01	587.34	753.52	2.89	919.94
1950	580.46	527.09	53.38	1.77	3.04	9.87	1094.57	838.82	582.74	741.26	2.90	922.35
2000	581.66	528.21	53.45	1.77	3.05	9.88	1092.80	837.93	580.80	749.44	2.90	923.26

Table 1-10: Reserve Case (MI) Select Pit Shells Optimisation Results

Lafigué DFS	Units	USD 675/oz	USD 875/oz	USD 900/oz	USD 1175/oz	USD 1300/oz
Inventory	(Mt)	4.4	17.1	28.0	42.5	44.8
• Gold grade	(g/t Au)	2.07	2.11	2.03	1.95	1.91
• Contained gold	(koz Au)	294	1161	1829	2660	2756
Modifying Factors						
• Mining Dilution	(%)	From Regularisation 5x5x2.5 with addition 5% dilution				
• Mining Recovery	(%)					
Diluted						
• Inventory	(Mt)	4.6	18.0	29.4	44.6	47.1
• Grade	(g/t Au)	1.98	2.01	1.93	1.85	1.82
• Contained Metal	(koz Au)	294	1161	1829	2660	2756
Quantities						
• Total Rock	(Mt)	15.3	154.2	276.6	452.4	475.7
• Mineral Inventory	(Mt)	4.6	18.0	29.4	44.6	47.1
• Waste + OM	(Mt)	10.6	136.2	247.2	407.7	428.6
• Stripping Ratio	(t:t)	2.3	7.6	8.4	9.1	9.1
Operating Expenditures						
• Mining	(USD/tmined)	2.90	2.71	2.75	2.78	2.78
• Rehabilitation Cost	(USD/t_{ore})	0.06	0.06	0.06	0.06	0.06
• Processing CIL + G&A	(USD/t_{ore})	14.80	15.45	15.63	15.71	15.71
• Au Selling Cost	(USD/oz)	62.50	62.50	62.50	62.50	62.50
• Total Cash Cost	(USD/oz)	2.90	2.71	2.75	846	2.78
Product						
• Au Metallurgical Recovery	(%)	95.25	95.32	95.28	95.26	95.25
• Recovered Metal	(koz Au)	280	1106	1743	2534	2625
LoM Economic Summary						
• Metal Price	(USD/oz)	1300	1300	1300	1300	1300
• Revenue	(USD M)	364	1438	2265	3294	3412
• Mining Costs	(USD M)	44	418	762	1257	1323
• Processing Costs CIL	(USD M)	69	278	459	701	740
• Selling Costs	(USD M)	17.5	69.1	108.9	158.4	164.0
• Cashflow	(USD M)	233.1	663.9	918.4	1150.0	1156.6
• Discount Rate	(%)	5.0%	5.0%	5.0%	5.0%	5.0%
• Mill Rate	(Mt/a)	4.0	4.0	4.0	4.0	4.0
• DCF - Best Case	(USD M)	221.86	572.32	735.20	868.34	871.29
• DCF Specified	(USD M)	221.79	562.76	718.73	819.50	812.45
• DCF - Worst Case	(USD M)	221.79	560.93	690.23	716.19	694.92

Table 1-10: Reserve Case (MI) Select Pit Shells Optimisation Results

Lafigué DFS	Units	USD 675/oz	USD 875/oz	USD 900/oz	USD 1175/oz	USD 1300/oz
• Project Life	(years)	1.4	5.4	8.9	13.5	14.3
Cut-Off Grade						
• OCOG - OPEX CIL	(USD/t_{ore})	24.34	38.71	41.54	43.88	43.81
• ECOG - OPEX CIL	(USD/t_{ore})	14.80	15.45	15.63	15.71	15.71
• OCOG CIL	(g/t Au)	0.6	1.0	1.1	1.2	1.2
• ECOG CIL	(g/t Au)	0.4	0.4	0.4	0.4	0.4
• ISOCOG CIL	(g/t Au)	0.7	1.2	1.2	1.3	1.3
• ISECOG CIL	(g/t Au)	0.4	0.5	0.5	0.5	0.5

1.14.6.2 Measured, INDICATED, and Inferred (MIF) Pit Optimisation Results

For the MIF case, inferred material was monetised with the Measured and Indicated to evaluate if there is any inferred material in the near term that might add value or be sterilised by the current MI pit. Figure 1-5 shows a plan view of the MI USD 1300/oz (red) pit shell to the MIF USD 1300/oz (blue) and USD 1500/oz (purple) pit shell, whilst Table 1-11 provides a summary inventory for each. The inferred inventories are mostly small thin lenses within the main pit at depth, expanding the pit highwall to the south.

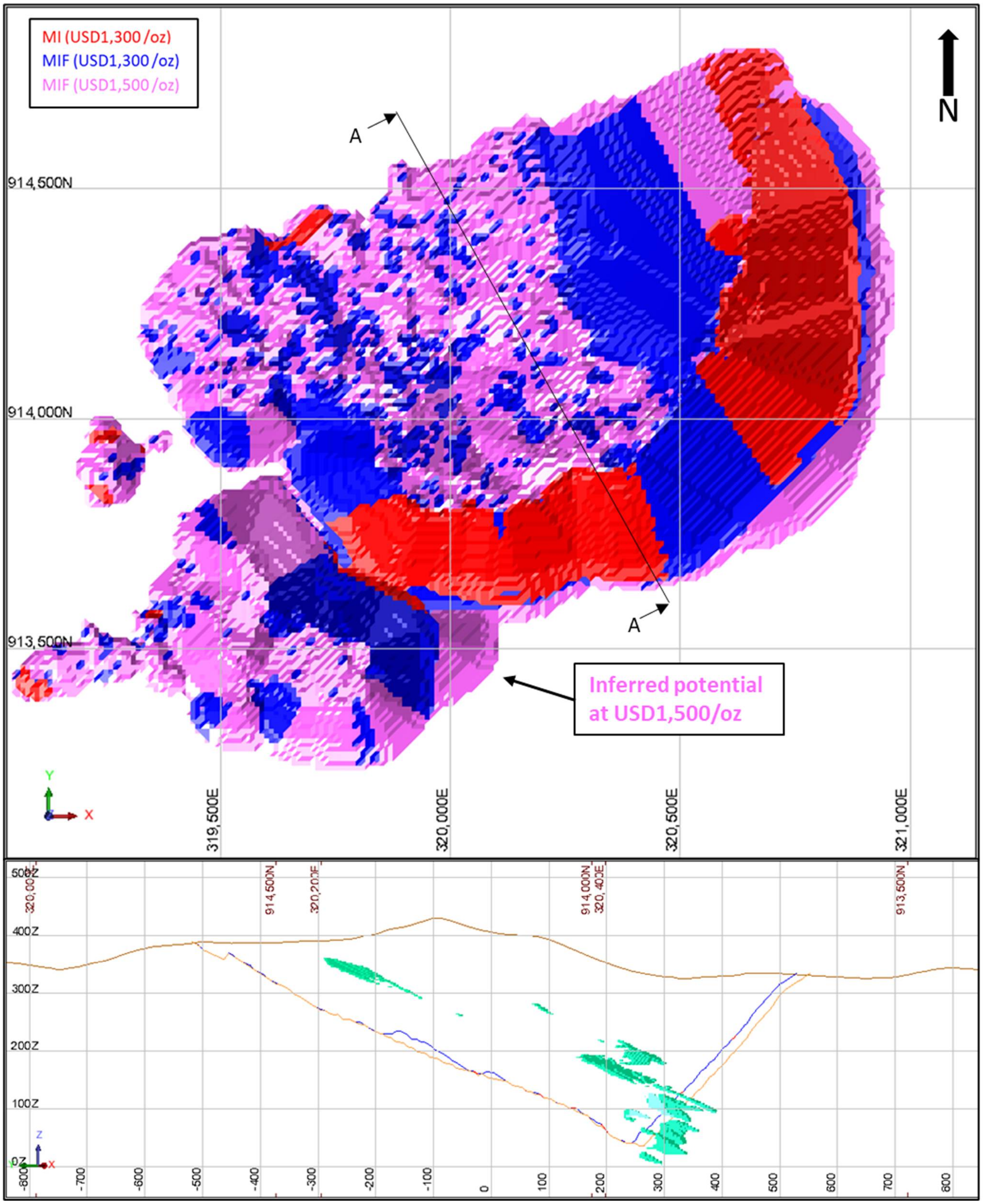

Figure 1-5: MI vs MIF comparison

Table 1-11: MI vs MIF Comparison

Lafigué FS Optimisation Results	**Units**	**USD 1300/oz MI**	**USD 1300/oz MIF**	**USD 1500/oz MIF**
Regularised 5x5x2.5				
In-situ Inventory	(Mt)	44.8	46.7	54.0
• Gold grade	(g/t Au)	1.91	1.91	1.81
• Contained gold	(koz Au)	2756	2860	3136
Modifying Factors				
• Mining Dilution		From Regularisation 5x5x2.5 with additional 5% Dilution		
• Mining Recovery				
Diluted				
• Inventory	(Mt)	47.1	49.0	56.7
• Grade	(g/t Au)	1.82	1.81	1.72
• Contained Metal	(koz Au)	2756	2860	3136
Quantities				
• Total Rock	(Mt)	475.7	495.5	572.3
• Mineral Inventory	(Mt)	47.1	49.0	56.7
• Waste + OM	(Mt)	428.6	446.5	515.6
• Waste	(Mt)	418.0	434.4	505.0
• Inventory (Below Cut-off)	(Mt)	10.6	12.1	10.6
• Stripping Ratio	(t:t)	9.1	9.1	9.1
Operating Expenditures				
• Mining	(USD/t mined)	2.78	2.77	2.79
• Rehabilitation Cost	(USD/tore)	0.06	0.06	0.06
• Processing CIL + G&A	(USD/tore)	15.71	15.60	15.63
• Au Selling Cost	(USD/oz)	62.50	62.50	71.50
• Total Cash Cost	(USD/oz)	859	863	922
Product				
• Au Metallurgical Recovery	(%)	95.25	95.24	95.22
• Recovered Metal	(koz Au)	2625	2724	2986
LoM Economic Summary				
• Metal Price	(USD/oz)	1300	1300	1500
• Revenue	(USD M)	3412	3541	4479
• Mining Costs	(USD M)	1323	1368	1591
• Processing Costs CIL	(USD M)	740	765	886
• Selling Costs	(USD M)	164.0	170.3	213.5
• Cashflow	(USD M)	1156.6	0.0	0.0
• Discount Rate	(%)	5.0%	5.0%	5.0%

Table 1-11: MI vs MIF Comparison

Lafigué FS Optimisation Results	Units	USD 1300/oz MI	USD 1300/oz MIF	USD 1500/oz MIF
• Mill Rate	(Mt/a)	4.0	3.3	3.3
• DCF - Best Case	(USD M)	871.29	894.77	1283.90
• DCF Specified	(USD M)	812.45	809.93	1166.06
• DCF - Worst Case	(USD M)	694.92	707.53	1003.98
Project Life	(years)	11.8	14.9	17.2
Cut-Off Grade				
• OCOG - OPEX CIL	(USD/tore)	43.81	43.50	43.70
• ECOG - OPEX CIL	(USD/tore)	15.71	15.60	15.63
• OCOG CIL	(g/t Au)	1.2	1.1	1.0
• ECOG CIL	(g/t Au)	0.4	0.4	0.4
• ISOCOG CIL	(g/t Au)	1.3	1.3	1.1
• ISECOG CIL	(g/t Au)	0.5	0.4	0.4

1.14.6.3 Whittle Sensitivity analysis

Table 1-12 summarise the Whittle sensitivities for: Mining Cost, Processing Cost, Gold Price, and Slope Angle, run on ultimate shell of USD 1300/oz. The greener colours are the sensitivities with the most significant improvement on earnings before interest tax and amortisation excluding any capital cost (EBITA $_{excl.CAPEX}$) and the red colours with the highest decrease. The brighter the colour, the higher the sensitivity.

Table 1-12: Whittle Sensitivities

Sensitivity (%)	Total (Mt)	Waste (Mt)	Ore (Mt)	Grade (g/t)	Gold (Moz)	NPV (USD M)
Mining Cost						
• +15	452.3	407.8	44.5	2.0	2.66	652
• +10	452.3	407.8	44.5	2.0	2.66	703
• +5	452.3	407.7	44.6	1.9	2.66	753
• 0	452.3	407.7	44.6	1.9	2.66	804
• -5	452.3	407.7	44.7	1.9	2.66	855
• -10	452.3	407.6	44.7	1.9	2.66	905
• -15	452.3	407.6	44.7	1.9	2.66	956
Processing Cost						
• +15	452	409	43	1.99	2.64	766
• +10	452	409	44	1.98	2.64	779
• +5	452	408	44	1.96	2.65	793
• 0	452	408	45	1.95	2.66	804
• -5	452	407	45	1.93	2.67	816
• -10	452	407	45	1.92	2.67	829

Table 1-12: Whittle Sensitivities

Sensitivity (%)	Total (Mt)	Waste (Mt)	Ore (Mt)	Grade (g/t)	Gold (Moz)	NPV (USD M)
• -15	452	406	46	1.91	2.68	849
Gold Price						
• +15	452	405	47	1.87	2.69	1111
• +10	452	406	46	1.90	2.68	1029
• +5	452	407	45	1.92	2.67	917
• 0	452	408	45	1.95	2.66	804
• -5	452	409	44	1.98	2.64	691
• -10	452	410	43	2.02	2.63	577
• -15	452	411	41	2.06	2.60	470
Slope Angle						
• +15	430	383	47	1.96	2.81	939
• +10	431	385	46	1.95	28.84	615
• +5	443	397	46	1.94	2.74	865
• 0	452	408	45	1.95	2.66	804
• -5	454	410	44	1.95	2.64	762
• -10	444	401	43	1.94	2.53	765
• -15	452	410	42	1.94	2.49	729

1.14.7 Pit Design

Figure 1-6 illustrate the Lafigué stage designs. The hanging wall is to the south of the pit, while the footwall is in the north. The number of ramps passing the hanging wall was kept as low as practicable, to avoid lowering the overall slope angle and increasing the waste tonnage. The pit design in the north followed the ore body as closely as possible to reduce dilution and losses in the footwall.

Interim Stages were selected with Stage 1 at USD 725/oz, which includes pushbacks 1 for Pit A, Pit B and Pit C. Stage 2 was designed between USD 850 and 875/oz to accommodate the minimum mining width and expands Pit A to Pushback 2 boundary. Stage 3 included Pit A pushback 3 and Pit B pushback 2 and was designed between USD 900 and 1050/oz to allow consistent ramp access between north and south, transitioning into Stage 4, designed at USD 1175/oz pit shell. Stage 4 mines the final Pit A pushback 4, forming the ultimate pit boundary.

Stage 1 comprises the satellite pit, Pit C, Pushback 1 of the satellite pit, Pit B, and Pushback 1 of the Main pit, and Pit A. Pushback 1 of the Main pit reaches a maximum depth of 90 m at 290 mamsl. Therefore, Pit B is brought in during stage 1 to bring the oxide material forward in the schedule.

Stage 2 extends the Main pit to the east and down to 135 mamsl, resulting in a maximum depth of 230 m. Pit B remains the same during Stage 2 to focus mining on the Main pit. The Main pit's north wall (footwall) forms part of the final pit wall.

Stage 3 reaches a maximum depth of 235 m at the 140 mamsl in the Main Pit A pushback 3, with Pit B and Pit C at final limits. Main pit pushback 3 was limited to 140 mamsl to join up with pushback 4, which extends the Main pit deeper and to the south. This south wall then forms part of the final highwall.

Pit A pushback 3 was adjusted slightly in width and limited to 140 mamsl, to allow continuous ramp access with pushback 4 between (265 and 215) mamsl and below 140 mamsl. There is an opportunity to extend Pit A pushback 3 deeper to mine all the ore tonnes Whittle selected for Pit A pushback 3; however, the strategic schedule indicated that mining deeper at this stage would mainly add material to the stockpile and is not explicitly required to sustain plant feed. In addition, it should be noted that if Pit A pushback 3 is extended to the entire boundary and depth at 140 mamsl, Pit A pushback 4 will require additional ramps with switchbacks to maintain bench access.

Stage 4 extends the Main pit deeper to its maximum depth of 338 m at 0 mamsl. It also advances to the southwest to reach the final pit wall, forming the ultimate pit design.

Table 1-13 summarises the pit design inventories by stage and Table 1-14 by wreathing type. The inventories show that the designs do not precisely follow the selected Whittle shells. Designing to follow the pit shells to operate in isolation from each other, required multiple switchbacks resulting in flatter overall slopes, more waste being mined and a significant increase in cycle times. In addition, the strategic schedule showed that the Lafigué pit quickly extends to its final limits, and pushbacks add limited value.

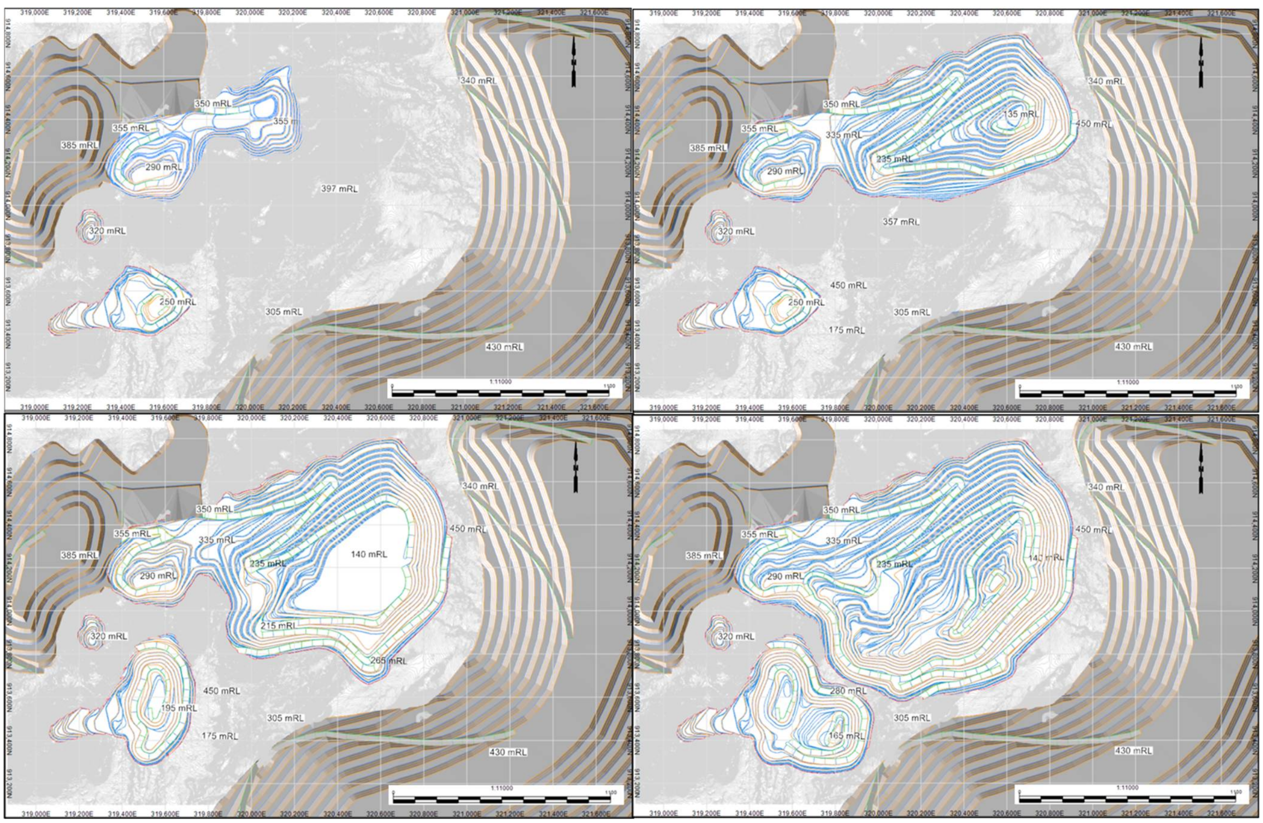

Figure 1-6: Lafigué Stage 1-4 Designs (Plan View) (SRK,2022)

Table 1-13: Pit Design Summary

Deposit	Total (kt)	Waste (kt)	Ore (kt)	Grade (g/t)	Gold (koz)	Total (kbcm)	Waste (kbcm)	Ore (kbcm)
Pit A PB1	23 545	17 674	5871	1.60	301	9486	7169	2317
Pit A PB2	161 670	146 536	15 134	1.80	874	59 307	53 897	5410
Pit A PB3	138 871	131 774	7097	1.57	358	50 048	47 508	2540
Pit A PB4	118 139	99 591	18 548	1.63	973	42 488	35 885	6603
Pit B PB1	5 971	5090	881	1.71	48	2787	2403	384
Pit B PB2	18 621	17 562	1060	1.98	67	6997	6626	371
Pit B PB3	24 361	23 308	1053	2.50	85	9266	8898	368
Pit C	436	268	168	1.51	8	206	133	73
Total	**491 615**	**441 802**	**49 813**	**1.69**	**2714**	**180 584**	**162 517**	**18 067**

Table 1-14: Pit Design inventory by Weathering

Weathering Type	Total (kt)	Waste (kt)	Ore (kt)	Grade (g/t)	Gold (koz)	Total (kbcm)	Waste (kbcm)	Ore (kbcm)
Oxide	22 390	21 523	867	1.23	34	13 773	13 238	535
Transitional	34 248	32 169	2079	1.39	93	13 879	13 036	843
Fresh	434 977	388 111	46 866	1.72	2 587	152 932	136 244	16 689
Total	**491 615**	**441 802**	**49 813**	**1.69**	**2 714**	**180 584**	**162 517**	**18 067**

Table 1-15 compares the pit shell (MI) and the ultimate pit designs. The pit designs mine additional waste to maintain the batter angle and berm width requirements for stability while the pit shell follows the block model. Extra waste is also mined due to ramps in the hanging wall. Further design optimisation is ongoing.

Table 1-15: Pit Design Comparison

Item	Units	Design	Whittle	Variance (%)
Total	Mt	491.61	452.35	8%
Waste	Mt	445.69	407.71	9%
Ore	Mt	45.92	44.64	3%
Grade	g/t Au	1.80	1.85	-3%
Ounces	Moz	2.66	2.66	0%
Strip ratio	Waste:Ore	9.71	9.13	6%

1.14.8 Historical Mineral Reserve Statements December 2020

Snowden completed the 2020 reserve estimate. In addition, Snowden completed pit optimisation and mine design and developed the LoM schedules based on the Endeavour Lafigué deposit block model (fetekro_bm_july202.dm).

The Reserve statement, effective 31 December 2021, reported at USD 1500/oz gold price and a marginal cut-off grade of 0.4 g/t Au is seen in Table 1-16. The Probable Mineral Reserve reported for the Lafigué Project was 32.0 Mt at 2.1 g/t Au, containing 2160 Au koz with an estimated LoM of 11 years.

Table 1-16: Historical Mineral Reserve Statements December 2020

Item	Units	Proved	Probable	Total Mineral Reserve
Ore	Mt	0	32.0	32.0
Gold grade	g/t Au	0	2.1	2.1
Contained gold	Moz	0	2.1	2.1

Table 1-16 note: some numbers may not sum correctly due to rounding

1.14.9 Mineral Reserve Statement as of the 01 June 2022

The reserve estimate is in accordance with the Canadian National Instrument 43-101, and adhering to the CIM Definition Standards guidelines (CIM, 2014). Therefore, public Reports dealing with exploration results, mineral resources and mineral reserves must use only the terms Proved or Probable mineral reserves, Measured, Indicated, and Inferred mineral resources and exploration results as shown in Table 1-17

SRK confirms that the Mineral Reserve statement presented in Table 1-17 has been derived from the Mineral Resource with an 'Effective Date' of 15 May 2022 authored by SRK (31113_Lafigué_MRE_May_2022).

The Mineral Reserve reported by SRK is constrained within an engineered design pit based on the optimised pit shell generated solely on the Measured and Indicated classified portion of the Mineral Resource. Inferred material within the pit design was treated as waste for reporting purposes.

The Reserve statement, Effective 01 June 2022, reported at USD 1300/oz gold price and a marginal cut-off grade of 0.40 g/t Au is presented in Table 1-17. The Probable Mineral Reserve reported for the Lafigué Project is 49.81 Mt at 1.69 g/t Au, containing 2.71 Moz of Au with an estimated LoM of 13 years. The reported mineral reserve is associated with 441.8Mt of waste mining corresponding to 8.9 to 1.0 waste to ore strip ratio on mass.

Table 1-17: Lafigué Project Mineral Reserve 01 June 2022

Classification Category	Material Type	Tonnes (Mt)	Grade, Au (g/t)	Metal Content, Au (Moz)	CoG (Au g/t)
Proved	Oxide				
	Transitional				
	Fresh				
Sub-Total Proved	**ALL**				
Probable	Oxide	0.87	1.23	0.03	0.40
	Transitional	2.08	1.39	0.09	0.40
	Fresh	46.87	1.72	2.59	0.40
Sub-Total Probable	**ALL**	**49.81**	**1.69**	**2.71**	**0.40**
Proved and Probable	Oxide	0.87	1.23	0.03	0.40
	Transitional	2.08	1.39	0.09	0.40
	Fresh	46.87	1.72	2.59	0.40
Total Ore Reserve	**ALL**	**49.81**	**1.69**	**2.714**	**0.40**

Table 1-17 notes:

- Some numbers may not sum correctly due to rounding.
- The statement was depleted based on the depletion surface of end August 2021 ("topo_artisanal_09092021"), including Artisanal mining up to this date.
- Above cut-off grade of 0.40 g/t Au using an Au price of USD 1300/oz.
- Modifying factors for dilution range from (0 to 14) % and mining recovery between (95 to 100) %.
- All figures are rounded to reflect the relative accuracy of the estimate.
- Ore Reserves have demonstrated economic viability.
- The pit inventories were constrained within a pit design. The Ore Reserve comprises a mine life of some thirteen years, with an additional one years of stockpile feed, totalling 13 years.
- The mineral resources and reserves have been estimated and reported in accordance with Canadian National Instrument 43-101, 'Standards of Disclosure for Mineral Projects and the Definition Standards adopted by CIM Council in May 2014. The Ore Reserve is given based on 100% ownership of the property.

1.15 Mining Methods

1.15.1 Mining Strategy

The Project will make use of conventional open pit truck and excavator operation with the production unit operations (drilling, blasting, loading, hauling, and dumping) carried out by contractor mining personnel and equipment. The mining contractor will be responsible for short term production planning, drilling (production and grade control), loading and hauling. Blasting will be carried out by a specialised blasting contractor, that will also be responsible for the supply of explosives.

The saprolite is anticipated to be primarily free-dig potentially requiring ripping with 14% of oxide material planned for blasting. A low powder factor was estimated for blasting the oxide material, with blasting of oxides mainly consisting of fracturing the harder laterite cap with a low powder factor of 0.32 kg/m³. As the rock strengths increase, blasting will be utilised more regularly within the transitional zone with powder factors estimated at 0.59 kg/m³. All the fresh rock will be blasted with powder factors estimated at 0.76 kg/m³. Production drilling of transitional and fresh material will be undertaken by top hammer drills drilling 152 mm diameter holes.

Mining is envisioned to occur in 10 m benches, with double batters to achieve the final 20 m bench. Mining will occur in 3 to 4 flitches depending on required vertical selectivity. This practice decreases dilution by using selectivity practices utilising smaller loading units for ore loading. Ore and waste will be loaded with hydraulic diesel shovels and all material will be hauled out of the pits by diesel powered trucks. The material will be hauled to various destinations as part of the overall mining strategy, namely:

- directly to primary crusher;
- RoM pad stockpile;
- topsoil stockpiles;
- aggregate stockpile; and
- waste dump.

1.15.2 Hydrogeology and Open Pit Water Inflow Review

The Lafigué pit water management system is designed to lower the groundwater table ahead of mining and reduce pore pressures behind pit walls, remove ongoing groundwater seepage from the pit to the extent possible, minimize the impact on mining associated with surface water accumulation in the pit in response to rainfall events, and capture and divert stormwater runoff around the pit.

Groundwater flows will be captured by a network of dewatering wells, sub-horizontal drains (to drain pore pressures behind pit slopes) and open in-pit ditches. Surface water entering the pit will be dewatered through in-pit sumps. The surface water dewatering strategy is designed around managing the 1:100 year 24-hour rainfall event. It is assumed that all rainwater collected during the peak storm event would be evacuated from the pit floor within five days.

Pit dewatering activities will be split with the contractor being responsible for all dewatering activities in-pit and the owner's team for all ex-pit borehole dewatering. All in-pit dewatering costs are accounted for in the mining contractor unit rates. An allowance of USD 500 k was allocated for pre-mining ex-pit borehole drilling and equipping; and, a concurrent baseline aquifer assessment, including water quality assessment and additional groundwater modelling.

The key operational requirements for the contractor include:

- installing and commission in-pit and ex-pit boreholes to provide dewatering capacity both prior to and during mining;
- minimising water flows into the pit;
- maintaining pit wall drainage, including sub-horizontal drains and in-pit ditches; and
- to provide adequate in-pit sumps, pumping capacity and piping capable of handling and removing expected water inflows, including foreseeable extreme events.

1.15.3 Hydrogeology

The open pit hydrogeology study currently has limited characterisation and insufficient monitoring data leading to some degree of uncertainty in the subsequent groundwater modelling analysis and therefore limits the design/validity of the water management plan for the open pit operation.

SRK considers that the pit hydrogeological characterisation and dewatering design has been undertaken to a 'pre-feasibility study level' of development accordingly. However, whilst there are hydrological risks, these are not considered significant relative to the geotechnical risks. Notwithstanding this, further work is required to better define the geological structural model and the associated hydrogeological conditions, and this work should be done during detailed design/FEED and prior to start of mining.

1.15.4 Open Pit Geotechnical Engineering

SRK considers that BG has implemented a diligent review of the pre-existing geotechnical data, identifying gaps and defining confidence levels within the various models feeding into the Geotechnical Model and updating the slope stability analyses. Whilst the analysis undertaken is appropriate, the number of boreholes used to define the updated rock mass conditions and Rock Mass Classification values (4 No.) could be considered a lower bound and may not provide confidence in the spatial distribution of the geotechnical properties within the pit. In addition, drilling orientation bias has resulted in a very limited structural data set (especially for the hangingwall).

Other than foliation, no additional discontinuity sets have been defined, which could impact the achievability of the proposed inter-ramp angles within the hangingwall. As recognised by BG, this can be mitigated through additional geotechnical data collection and verification of the proposed design criteria.

Bench crest loss will be prevalent in the footwall within the design domains affected by the presence of foliation, and a 3D fault model will be critical to understanding the role the identified shear zones will have on any other footwall shears. It should also be noted that BG has used lower bound rock mass strength values (defined from Golder geotechnical logging of boreholes GTLF01 to GTLF07), within their analyses and additional data collection may show upside with regards to rock mass strength.

1.15.5 Mining Equipment

The recommended excavators for the Project are in a weight class of 100 to 200 t that fall in the range of the Komatsu PC1250 and PC2000, supported by 100-t dump trucks Caterpillar 777 or Komatsu 785; however, with the changes in the Resource Model and the subsequent change in process requirements from 3 to 4 Mt/a (db), the equipment selection was revised to a larger fleet more suitable for bulk mining. An owner mining equipment and cost model was developed based on the equipment proposed in the preliminary contractor submission. The contractor proposed 150-t capacity Komatsu HD-1500 dump trucks with Komatsu PC3000 shovel for waste loading and 100-t capacity Komatsu 785-7 trucks with Komatsu PC2000 or equivalent for ore loading.

Other mining equipment configurations are being reviewed with detailed studies; however, the final mining fleet configuration and quantities depend on the agreed mining contractor. Still, the final equipment configuration and quantities should reasonably align with the results obtained from the owner mining equipment model.

It should be noted that the preferred excavator size for the mine, based on the orebody and selectivity requirements, is in the range of the Komatsu PC2000. Therefore, a larger ore excavator will result in an increased SMU; however, the impact would not significantly change the results of the DFS, due to the additional modifying factors already applied.

Based on the simulated productivity figures, equipment requirements throughout the LoMp, based on the required fleet totals are presented in Table 1-18, following. In addition, equipment replacement was calculated based on the direct operating hours in relation to the equipment life cycle hours.

Table 1-18: Equipment Requirements per Period

Equipment	Units	Max.	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032
Waste Shovel	(#)	**5**	-	5	5	5	5	5	5	5	4	4
Ore Shovel	(#)	**1**	-	1	1	1	1	1	1	1	1	1
ROM Loader	(#)	**1**	-	1	1	1	1	1	1	1	1	1
Stock Loader	(#)	**1**	-	1	1	1	1	1	1	-	-	1
Waste Truck	(#)	**23**	-	16	16	17	19	21	21	23	19	20
Ore Truck	(#)	**8**	-	4	5	4	6	5	5	6	7	6
Large Drill	(#)	**4**	-	4	4	4	4	4	4	4	3	3
Medium Drill	(#)	**3**	-	3	3	3	3	2	2	2	2	2
Small Drill	(#)	**1**	-	1	1	1	1	1	1	1	1	1
Track Dozer	(#)	**10**	-	7	7	7	8	9	9	10	9	9
Backhoe	(#)	**2**	-	2	2	2	2	2	2	2	2	2
Compactor	(#)	**1**	-	1	1	1	1	1	1	1	1	1
Motor Grader	(#)	**4**	-	3	3	3	3	4	4	4	4	4
Water Truck	(#)	**4**	-	3	3	3	3	4	4	4	4	4
Tire Handler	(#)	**1**	-	1	1	1	1	1	1	1	1	1
Fuel/Lube Truck	(#)	**2**	-	2	2	2	2	2	2	2	2	2
Service Truck	(#)	**1**	-	1	1	1	1	1	1	1	1	1
Lighting Plant	(#)	**21**	-	21	20	20	20	20	20	19	16	17
Light Vehicle	(#)	**10**	-	10	10	10	10	10	10	10	10	10

1.15.6 Mine Schedule

1.15.6.1 Overview

The mine schedule was developed using Geovia's Minesched™ scheduling software as the primary scheduling tool. Schedules were analysed on the cost of mining capital (equipment and pre-strip), whilst also considering unit mining operating costs, vertical pit advance rates to sustain plant throughput requirements and the requirement to optimise Project value.

The production schedule was developed monthly for 2024 and 2025, quarterly for 2026, 2027 and 2028, and annually thereafter.

1.15.6.2 Schedule Assumptions and Parameters

Adopting a rules-based iterative approach through Minesched allows one to determine an appropriate production schedule which provides the optimal economic value, whilst balancing practical mining constraints, particularly bench turnover rates and capital expenditure during ramp-up periods.

MineSched develops schedules based on the following:

- A set of user definitions and objectives such as material types, movement and priorities, fleet capacity, etc.
- On a block-by-block as opposed to a bench-by-bench scheduling approach. Consequently, the software does not need to utilise bench averaging. Instead, the actual grades and actual strip ratios are reported in any reporting period.
- Any schedule generated by Minesched adheres to set Parameters, Precedences and special relationship rules to improve the practicality of the schedule.
- Capacity constraints and targets can be used to control the tonnes, volumes of content being mined for Flagged material types.

1.15.6.3 Defined Grade Envelopes

Various material types were developed to improve material blending to the process plant during scheduling. Materials were based on weathering type; Laterite (LT), Saprolite (SP), Transitional (SP) and Fresh Rock (FR). Ore material was based on classification, and CoG was split into various grade envelopes or bins. These material types are shown in Table 1-19.

Table 1-19: Defined Grade Envelopes

Material type	Description	CoG (g/t Au)
Laterite		
• LT_HG	High Grade	>= 1.5
• LT_MG	Medium Grade	>0.8 & <=1.5
• LT_LG	Low Grade	>0.4 & <=0.8
• LT_MO	Marginal Ore	na
Saprolite		
• SP_HG	High Grade	>= 1.5
• SP_MG	Medium Grade	>0.8 & <=1.5
• SP_LG	Low Grade	>0.4 & <=0.8
• SP_MO	Marginal Ore	na
Transitional		
• TR_HG	High Grade	>= 1.5
• TR_MG	Medium Grade	>0.8 & <=1.5
• TR_LG	Low Grade	>0.4 & <=0.8
• TR_MO	Marginal Ore	na
Fresh		
• FR_HG	High Grade	>= 1.5
• FR_MG	Medium Grade	>0.8 & <=1.5
• FR_LG	Low Grade	>0.5 & <=0.8
• FR_MO	Marginal Ore	>0.4 & <=0.5

Table 1-19: Defined Grade Envelopes

Material type	Description	CoG (g/t Au)
Waste		
• LT_WST		Any
• SP_WST		Any
• TR_WST		Any
• FR_WST		Any

1.15.6.4 Max Capacity and Mining Ramp-up

To sustain plant throughput requirements, the maximum required capacity was tested between 50 and 55 Mt/a (db). As a result, the current ramp-up rate equates to 74% of total capacity based on 55 Mt/a in year 1. The monthly percentage of total capacity is illustrated in Table 1-20.

For the FS, the focus was to determine the required pre-strip to sustain the process plant throughput and blend requirements in the first year. The required ramp-up rate and duration were determined based on the pre-strip needs. The current projected timeline for mobilisation will form part of the Project's critical path and is a risk to the first-year production.

Table 1-20: Mining Ramp-up

Month	Percentage of Full Capacity (%)
1	20
2	58
3	76
4	87
5	90
6	100

1.15.6.5 Process Blend and Throughput Rate

Process throughput rate was increased from the original PFS of 3.3 Mt/a (db) to 4 Mt/a (db) to maintain relatively the same ounce profile due to the drop in average grade, within the revised Resource model. Plant ramp-up is planned as illustrated in Table 1-21, with the first month at 80% throughput and maintaining a full capacity of 4 Mt/a (db) thereafter. The minimum turn-down capacity was capped at 3.8 Mt/a (db) if insufficient fresh material was available to maintain the feed blend.

The Oxide process blend requirements must be restricted to less than 20 to 25% of total feed to ensure optimal performance of the High-Pressure Grinding Rolls ('HPGR'). The HPGR needs resistance to work against, and with no rocks in the feed, cannot be pushed apart. To overcome the above-mentioned constraint, a mobile crusher will be acquired (same one that will be used for the blast hole stemming material), oxide and transitional material will be sent through the mobile crusher directly into the ball mill thus by-passing the HPGR. Bypassing the HPGR allows the total oxide material in the feed blend to be increased to 200 t/h (db) or 2 Mt/a (db).

Table 1-21: Processing Ramp-up

Year	Percentage of Full Capacity (%)	Activity
1	0	Pre-production
2	98	Commissioning one month at 80% before total production
Remaining	100	Production

1.15.6.6 Gold Production

Various schedule scenarios were developed to derive the optimal schedule selected and discussed in more detail later within this section. All the strategic Whittle schedule scenarios and the initial tactical schedule following the guidance from the strategic schedules, resulted in a significantly fluctuating ounce profile ranging between 130 and 350 koz/a. The fluctuating ounce profile is primarily due to the nature of the deposit, which comprises of disseminated HG lenses, with LG gaps in between. This occurrence of HG and LG lenses results in large quantities of HG material being mined in one period followed by LG material, which results in LG stockpile material being fed until the next HG lens is intersected.

As a result, gold production was limited to between 200 and 250 koz/a, so as to present a more conservative and sustainable mining schedule. For this reason, a direct comparison just on NPV between the various schedule scenarios will not favour the sustainable scenario.

1.15.6.7 Schedule Results

Figure 1-7 shows the ex-pit movement by location. It peaks at around 55 Mt/a (db) with the variation in part due to different productivities by weathering type in Figure 1-8. Mining typically occurs in multiple stages and is primarily driven by high tonnage movement and slow sinking rate (maintaining enough working space). Figure 1-9 following shows ore tonnes mined by location, and in Figure 1-10, total ex-pit inferred tonnes.

Figure 1-11 shows the average sink rate per period. Depending on material weathering, sink rates were restricted to between 80 and 100 m. The initial sink rate is high during pre-strip, due to the initial topography and high percentage of oxides being mined. Other spikes in 2027 to 2030 are also due to oxides being mined during the start of pushback 3 and 4.

Figure 1-12 illustrate the stockpile balance and movement by grade envelope and weathering type. The grade is shown as Fully Graded Ore ('FGO') above economic cut-off and FGO with marginal ore ('MO'). The MO ore stockpiles are only processed at the end of the mining life, when mining stops and the overall cost structure decreases. The peak of 4.9 Mt at 0.48 g/t Au, occurs in 2031. Therefore, the only material in the stockpile is LG and MO material.

Table 1-22 summarises the annual mining production schedule, with Figure 1-13 illustrating the annual mine progression.

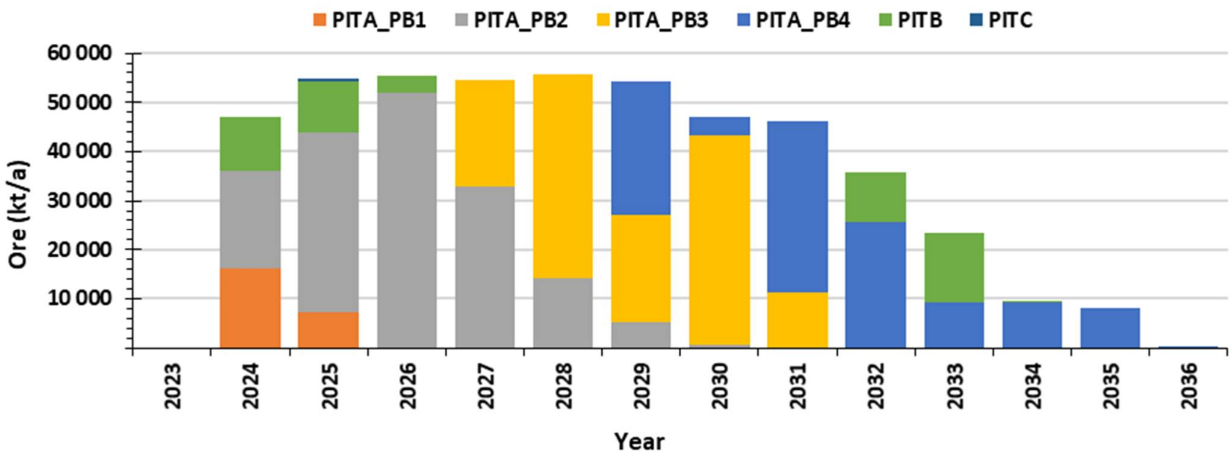

Figure 1-7: Total Ex-Pit Movement by Stage

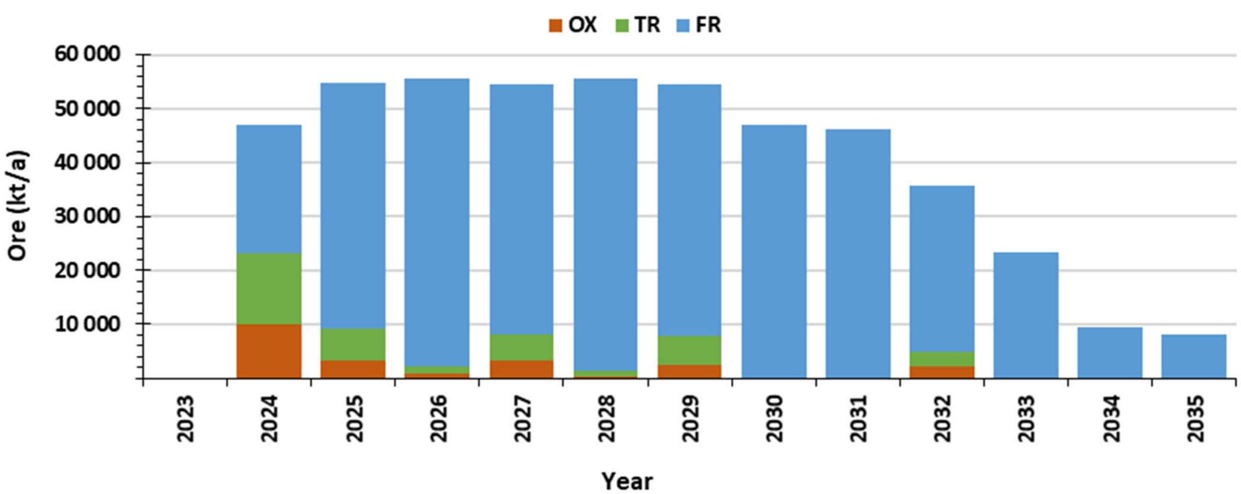

Figure 1-8: Total Ex-Pit Movement by Weathering

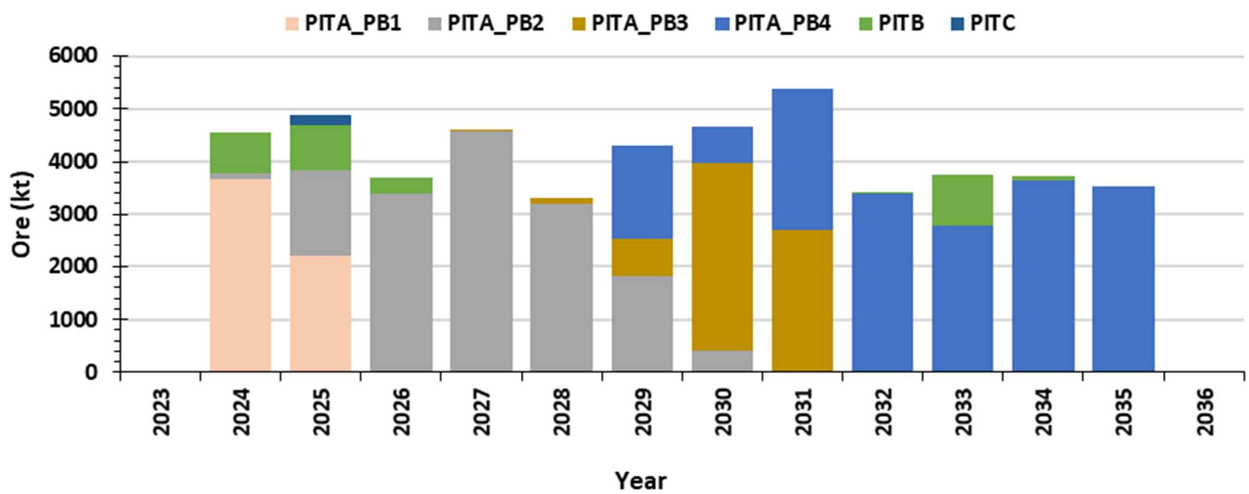

Figure 1-9: Total Ex-Pit Ore Movement by Location/Pit

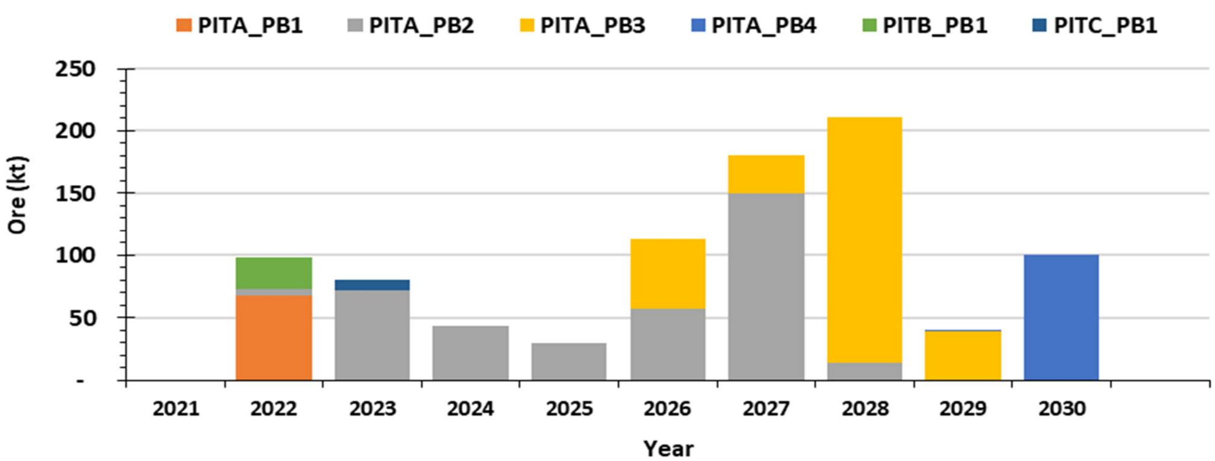

Figure 1-10: Total Ex-Pit Inferred Ore Mined by Location/Pit

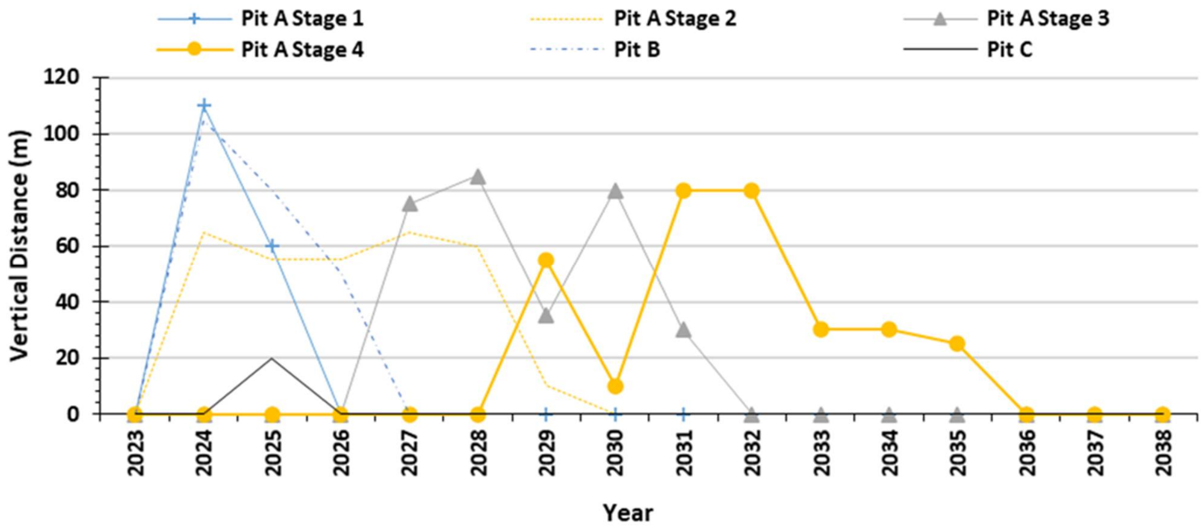

Figure 1-11: Bench Sink Rates

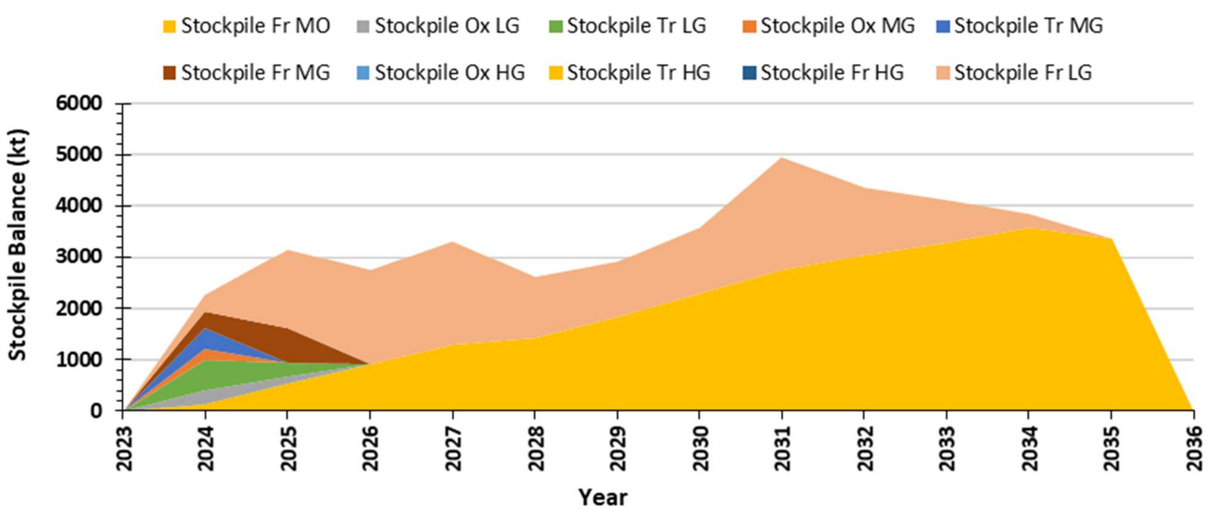

Figure 1-12: Long-Term Stockpile Balance by Grade Envelope

Table 1-22: Annual Mining Schedule

Mining Schedule	Units	Totals	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Total volume	(kbcm)	180 584	19 957	20 385	19 779	20 305	19 633	19 982	16 538	16 185	13 293	8250	3340	2924	14
Total Tonnes	(kt)	491 615	47 072	54 750	55 480	54 518	55 632	54 385	46 926	46 070	35 816	23 430	9401	8097	37
• Oxide	(kt)	22 390	10 085	3381	835	3151	258	2351	0	0	2330	0	0	0	0
• Transition	(kt)	34 248	13 092	5813	1276	4887	1167	5369	10	41	2496	96	0	0	0
• Fresh	(kt)	434 977	23 895	45 557	53 369	46 480	54 207	46 665	46 916	46 029	30 990	23 333	9401	8097	37
Waste mined	(kt)	441 802	42 529	49 878	51 779	49 941	52 322	50 079	42 263	40 683	32 409	19 677	5671	4562	10
Operating strip ratio	(t:t)	8.87	9.36	10.24	13.99	10.91	15.81	11.63	9.06	7.55	9.51	5.24	1.52	1.29	0.35
Feedable ore mined	(kt)	49 813	4543	4872	3701	4577	3310	4306	4664	5386	3407	3753	3731	3535	28
Feedable ore grade	(g/t)	1.69	1.37	1.57	1.40	1.70	2.16	1.78	1.57	1.44	2.18	1.89	1.90	1.71	1.96
Total ore mined	(kt)	49 813	4543	4872	3701	4577	3310	4306	4664	5386	3407	3753	3731	3535	28
Total ore grade	(g/t)	1.69	1.37	1.56	1.40	1.70	2.16	1.78	1.57	1.44	2.18	1.89	1.90	1.71	1.96
Ounces mined	(koz)	2714	200.52	245.10	166.59	249.88	230.34	245.85	235.75	249.36	238.28	228.24	228.07	194.60	1.76
• MI	(koz)	2715	200.66	245.16	166.59	249.88	230.34	245.85	235.75	249.36	238.28	228.24	228.07	194.60	1.76
• MIF	(koz)	38	3.07	2.23	0.96	0.64	4.09	12.78	6.57	1.99	4.36	0.83	0.41	0.09	0.00
Gold recoverable	(koz)	2584	190.48	233.19	158.39	237.84	219.71	234.27	224.30	236.94	227.52	217.44	217.41	185.31	1.68
HG - Ore tonne	(kt)	18 328	1522	1641	1012	1849	1656	1575	1627	1732	1381	1622	1450	1252	10
HG - Ore grade	(g/t)	3.23	2.50	3.09	3.05	3.02	3.46	3.52	3.03	2.75	4.22	3.30	3.63	3.41	3.87
MG - Ore tonne	(kt)	15 413	1559	1540	1223	1283	980	1193	1376	1941	1006	1115	1087	1100	12
MG - Ore grade	(g/t)	1.04	1.03	1.04	1.05	1.06	1.07	1.03	1.07	1.04	1.01	1.05	1.06	1.02	1.16
LG - Ore tonne	(kt)	12 179	1335	1276	1094	1077	533	1141	1193	1261	713	771	911	869	3
LG - Ore grade	(g/t)	0.61	0.58	0.61	0.61	0.63	0.62	0.61	0.61	0.62	0.61	0.61	0.61	0.61	0.55

Table 1-22: Annual Mining Schedule

Mining Schedule	Units	Totals	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Marginal-Ore - Ore tonne	(kt)	3892	127	415	373	368	141	398	468	452	307	245	282	314	3
Marginal-Ore - Ore grade	(g/t)	0.43	0.43	0.43	0.43	0.43	0.43	0.42	0.43	0.43	0.43	0.42	0.43	0.43	0.42
NF-Inferred tonne	(kt)	953	98	81	43	30	113	180	211	40	100	34	20	2	0
NF-Inferred grade	(g/t)	1.24	0.97	0.86	0.68	0.66	1.12	2.20	0.97	1.56	1.36	0.77	0.65	1.18	0.00
NF-Inferred ounce	(koz)	38	3.07	2.23	0.96	0.64	4.09	12.78	6.57	1.99	4.36	0.83	0.41	0.09	0.00
Mining+GC for forecast	(USD M)	1236	108	121	126	131	137	137	131	123	98	65	32	28	0
Mining+GC Unit Cost	(USD/t)	2.51	2.31	2.22	2.27	2.40	2.47	2.51	2.77	2.67	2.74	2.80	3.30	3.33	2.71

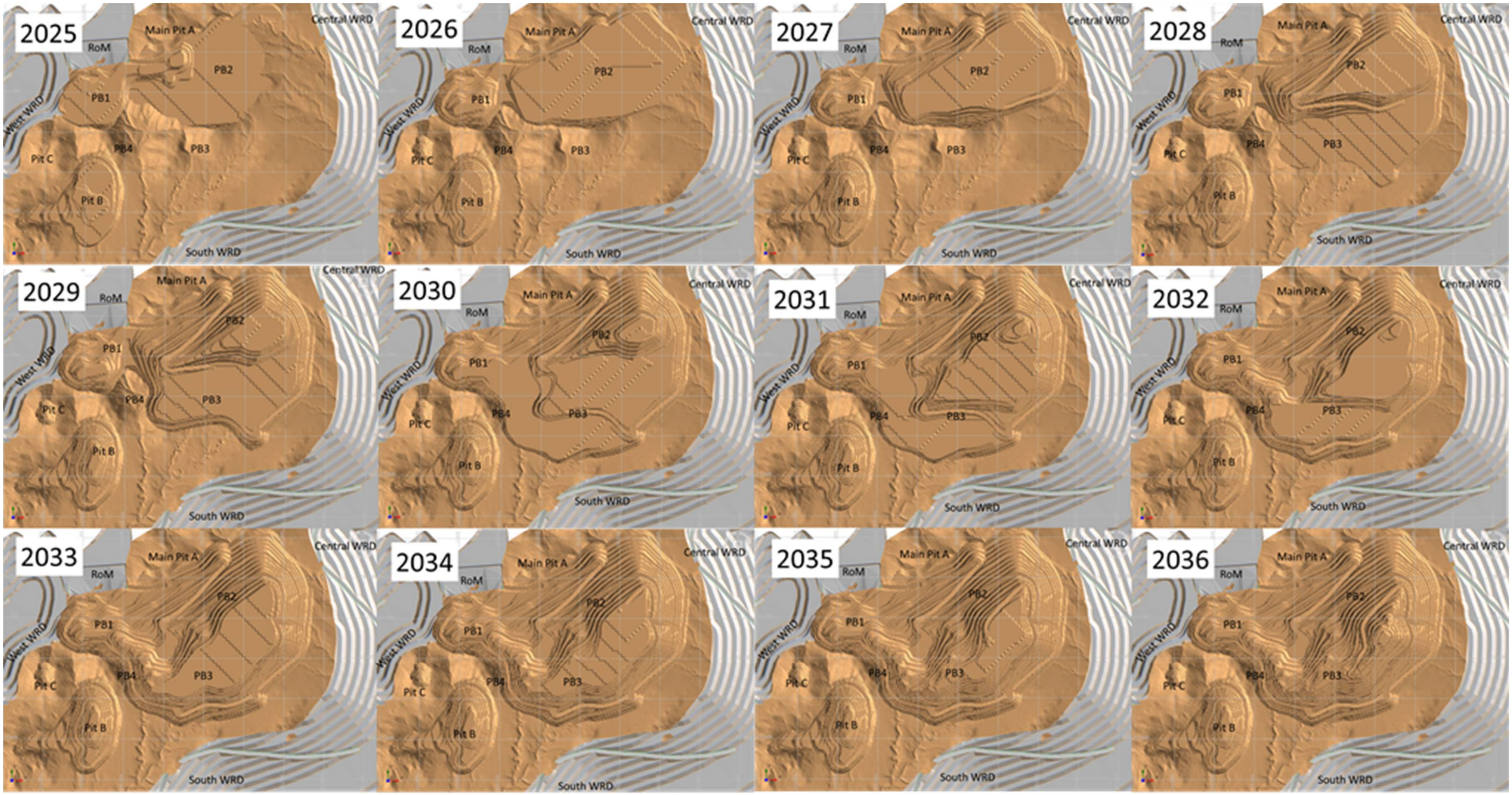

Figure 1-13: Annual Mine Progression

1.15.7 Processing Schedule

Figure 1-14 shows the ore mined and mined grade, process feed and feed grade. The feed grade is predominately between 1.4 and 2.0 g/t Au and decreases at the backend of the LoM, as feed over this period, is sourced from the MO stockpile. RoM feed grade is predominantly above the mined grade, apart from when insufficient ore is mined to sustain plant capacity, and feed is sourced from stockpile material.

Figure 1-15 shows the plant throughput by year, and ore feed to the processing plant by weathering type. Table 1-23 summarises the annual Processing schedule.

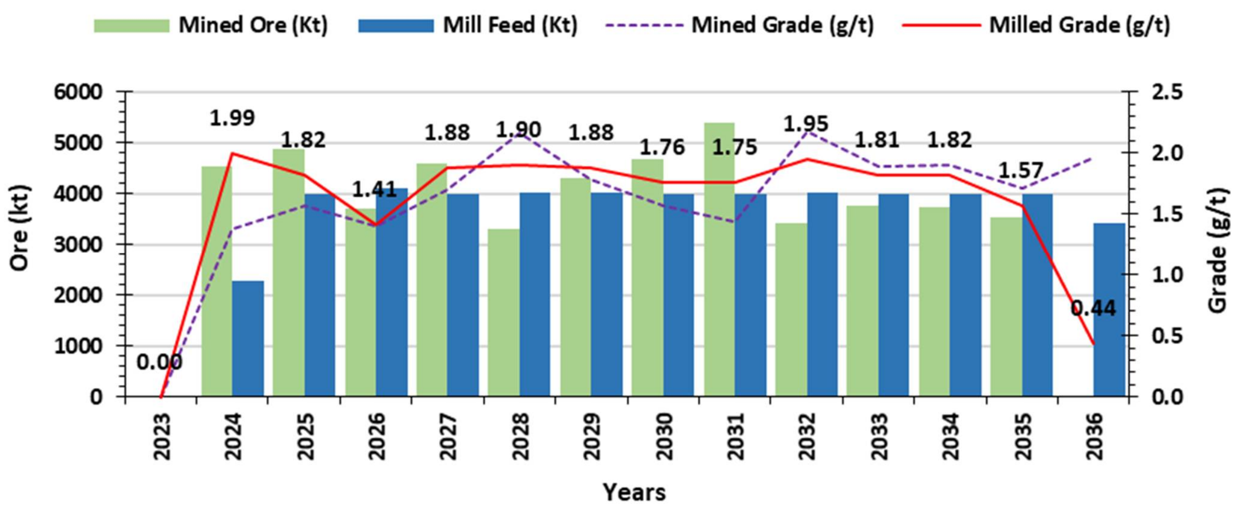

Figure 1-14: Ore Feed by Material Type

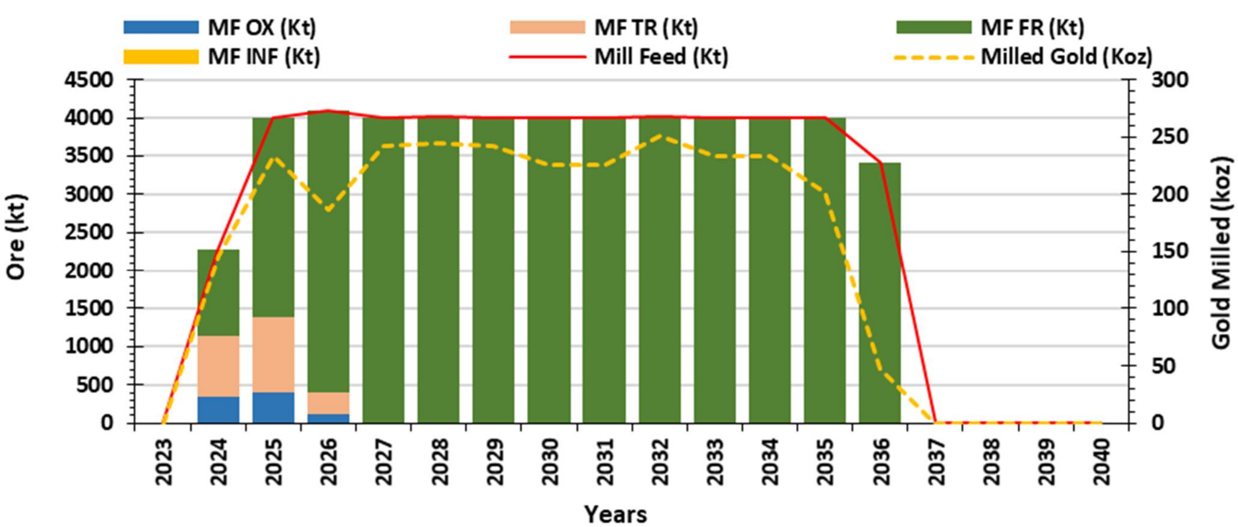

Figure 1-15: Ore Feed by Weathering Type and Gold Milled

1.15.8 Gold Product Schedule

Figure 1-16 illustrate the weighted average metallurgical recovery and gold produced by year. Production during the first three years is the most constrained, with 2026 not exceeding 200 koz. After 2026, production improves with higher grades becoming available as Pit A pushback 2, reaches a thicker portion of a HG lens. After that, gold production fluctuates between 210 and 240 koz/a as targeted, with the tail-end decreasing as MO becomes the primary feed material.

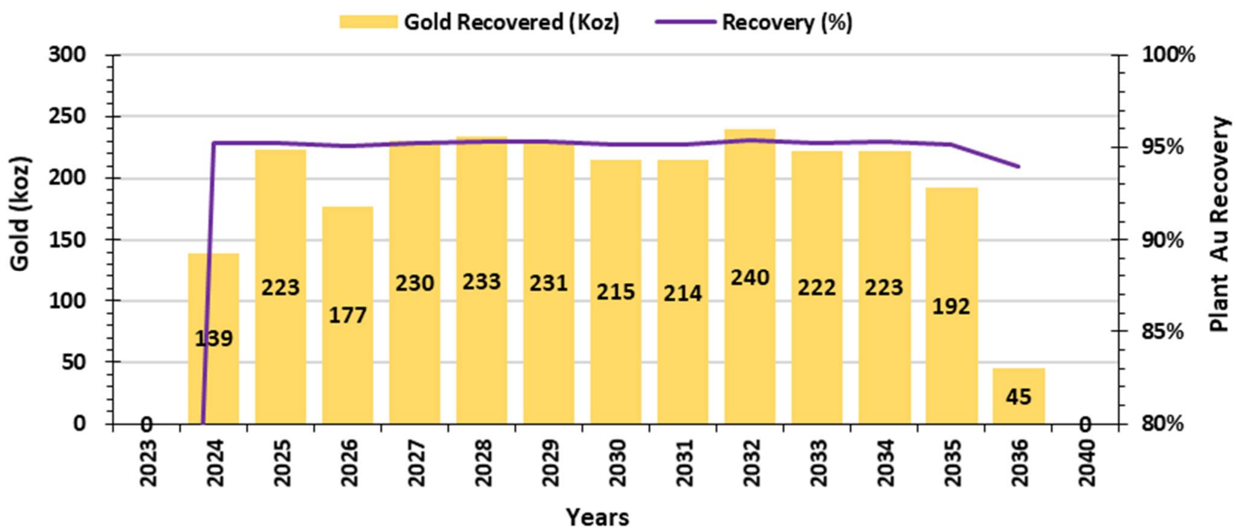

Figure 1-16: LoM Gold Production and Gold Metallurgical Recovery by Year

Figure 1-16 note: Gold production is reported by calendar year, with first gold being poured in Q2 2024. The reporting periods within the financial model was adjusted to first gold pour date, with year one running from Q2 2024 to Q2 2025.

Table 1-23: Annual Processing Schedule

Description	Units	Totals	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Ore Processed	(kt)	49 813	2279	4000	4098	4000	4011	4008	4000	4000	4011	4000	4000	4000	3405
Grade	(g/t)	1.69	1.99	1.82	1.41	1.88	1.90	1.88	1.76	1.75	1.95	1.81	1.82	1.57	0.44
	(kt)	867	342	400	125	0	0	0	0	0	0	0	0	0	0
Oxide	(%)	1.7	15	10	3	0	0	0	0	0	0	0	0	0	0
	(kt)	2079	798	995	279	0	0	8	0	0	0	0	0	0	0
Transitional	(%)	4.2	35	25	7	0	0	0	0	0	0	0	0	0	0
	(kt)	46 866	1140	2605	3694	4000	4011	4000	4000	4000	4011	4000	4000	4000	3405
Fresh	(%)	94.1	50	65	90	100	100	100	100	100	100	100	100	100	100
	(kt)	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Inferred	(%)	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Contained Gold	(koz)	2714	146	234	186	242	245	242	226	225	251	233	234	202	48
Gold Recovery	(%)	95.2	95	95	95	95	95	95	95	95	95	95	95	95	94
Gold Recovered	(koz)	2584	139	223	177	230	233	231	215	214	240	222	223	192	45

1.16 Recovery Methods

Based on Mine Schedule 13k (SRK, 2022) and previous mine schedule iterations, the Lafigué Process Plant (the 'Plant') has been designed to process 4.0 Mt/a (dry basis (db)) of fresh ore, over a 13-year life of mine (LoM) The weighted average LoM gold feed grade to the plant is 1.69 g/t, with a mean monthly range of (0.42 to 6.75) g/t. The mine plan was developed to suit the processing plant capacity/configuration and will produce a LoM weighted average of 208 koz/a of gold, with a LoM production range of (177 to 240) koz/a[1] over the years of full production from year 2 to year 11. Given low silver grades in the RoM ore, the gold doré produced is likely to contain in excess of 92% w/w gold. Metallurgical recoveries of gold are generally expected to exceed 96%.

At the plant front-end, a two-stage crushing/HPGR circuit was selected on the basis that the ore is essentially 95% unweathered rock, with only minor transitional and oxide components; and ore characteristics which indicate that the fresh ore will have high comminution energy requirements. The downstream circuit comprises a conventional ball milling and gravity/hybrid CIL treatment plant.

The key project and ore specific design criteria that the plant design must meet are as follows:

- 4.0 Mt/a (db) of fresh ore.
- For purposes of process design, the following mechanical availabilities have been assigned:
  - Closed circuit secondary crushing plant (70%).
  - Closed circuit high pressure grinding rolls (HPGR) circuit (86.7%).
  - Remainder of the downstream plant (91.3%).
- Intermediate crushed ore storage and provision of standby equipment for critical duties have been included to ensure the overall availability and nameplate throughput can be met on a sustainable basis.

The proposed process flow sheet is illustrated in Figure 1-17 following.

[1] Based on a calendar production schedule, not on a yearly production schedule from first gold pour, in Q2 2024.

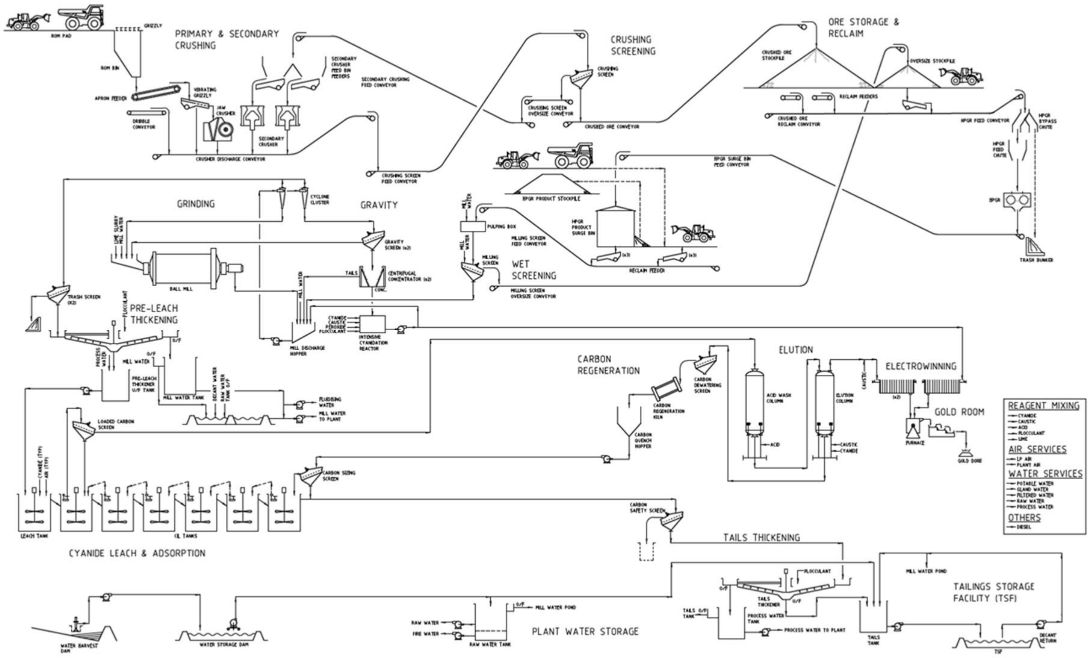

Figure 1-17: Overall Process Flow Diagram (Lycopodium, 2022)

As illustrated in Figure 1-17, the selected Plant flowsheet, incorporates the following unit process operations:

- Ore receipt at a RoM bin loaded by direct tip from haul trucks, or front-end loader (FEL). Mine operations will stockpile oxide/transition and low-grade ores on the RoM pad to allow controlled blending of the plant feed.
- Fixed grizzly protection to prevent oversize blockages and vibrating grizzly screening to bypass fines ahead of jaw crushing.
- Primary jaw crushing to produce a coarse crushed product.
- Secondary cone crushing in closed circuit with a dry sizing screen to produce an intermediate crushed product.
- A live secondary crushed ore stockpile providing 20 hours of buffer storage of crushed ore, with continuous reclaim to feed the HPGR circuit.
- HPGR operation in closed circuit with a wet sizing screen, with undersize slurry reporting to the ball milling circuit via the mill discharge hopper and classification hydrocyclones.
- A ball mill in closed circuit with cyclones to produce a grind size of 80% passing (P_{80}) 106 micron (µm).
- Gravity concentration and recovery of coarse gold from the milling circuit, with treatment of the gravity concentrate by intensive cyanidation and electrowinning of the pregnant solution to recover gold doré.
- Trash screening to remove any wood trash or grit/oversize material from the cyclone overflow ahead of the carbon-in-leach (CIL) circuit.

- Pre-leach thickening of the trash screen underflow to dewater the leach feed to reduce reagent consumption and the leach and adsorption tankage volume required. Pre-leach thickening also recovers much of the essentially cyanide free water for recycle to the mill circuit to allow reuse of the cyanide recovered in the process water at a moderate concentration. This lowers total cyanide usage in line with the International Cyanide Management Code (ICMC)[1] requirements.
- A leach tank ahead of the CIL tanks to maximise the gold solution grade feeding the adsorption tanks and to cater for pre-aeration of the slurry, should some ores consume the available dissolved oxygen provided by standard air sparging. The CIL circuit will continue leaching the gold in parallel with adsorption of the gold in solution, onto the activated carbon.
- A split AARL elution circuit, electrowinning, and gold smelting operations to recover gold from the loaded carbon to produce doré.
- Thickening of the CIL tails slurry to maximise the tails solids concentration, minimise gold solution losses, and to recover process water and cyanide.
- Dilution of the tails thickener underflow with decant return/raw water in order to meet the target cyanide discharge level to the tailings storage facility (TSF).
- Tailings pumping to the TSF.
- Reagent mixing, storage, and distribution facilities.
- Provision of water treatment as required, with storage and distribution of the various water services throughout the process plant.
- Generation of the compressed air required, and distribution through the circuit.

With the exception of diesel for elution heating, the average drawn power (14.2 MWe) for heating and power comes from the grid, which is a largely a mixture of hydro and gas fired generation capacity (Section 5) and is thus, less susceptible to changing oil prices.

Given that the plant is utilising water harvest dams for the supply of water, water conservation and re-use is a priority. Nominally, 0.42 t of raw water make-up/t of RoM ore (db) is required.

Whilst the ore is hard and abrasive, requiring nominally 2500 t/a of grinding media, cyanide and lime consumption are low at 852 t/a, and 1028 t/a respectively.

1.17 Project Infrastructure

1.17.1 Earthworks and Site Preparation

1.17.1.1 Overview

Preliminary bulk earthworks designs have been completed for the DFS across the various Project infrastructure/facilities. Allowances have been made for the site to be cleared and grubbed, cut back, filled and prepared to the necessary design levels, and to accommodate drainage.

[1] https://cyanidecode.org.

No specialist earthworks, foundations or ground improvement works (such as piling, ground anchors, grout injection, etc.) are proposed for the construction of the Lafigué Project infrastructure.

1.17.1.2 Geotechnical

Geotechnical investigations were undertaken for the: Tailings storage facility (TSF); Water storage dam (WSD); Water harvesting dam (WHD); Plant; Waste dumps and Airstrip and an assessment was done for construction materials

General

Typically, the ground conditions encountered at the site comprise:

- Fine and coarse grained alluvial and colluvial material to approximately 1 m depth, which is low strength in places and the fines often high plasticity.
- High plasticity residual clay which is typically stiff to very stiff but the upper layer can be lower strength and firm. This transitions to extremely weathered material.
- Igneous bedrock from approximately 10 m depth. Rock outcrops are present in places.
- The upper approximately 3 m of ground tends to be more variable and can be low strength and more compressible.
- Groundwater was typically observed in boreholes at greater than 15 m depths on higher ground, and at close to surface in the base of the valleys.

Artisanal Mining

Artisanal miners have historically been active on site, with the resultant deforestation, excavations, sediment generation (through washing of ore) and use of chemicals such as hydrochloric acid, mercury and cyanide in processing the ores. Artisanal mining has been undertaken over much of the valley floor, through the Water Storage Dam (WSD) area and parts of the Tailings Storage Facility (TSF) basin.

No mining was observed at the Water Harvest Dam (WHD). The mining activities have created disturbed ground including open pits, mine shafts, and tailings and ground and/or groundwater pollution may have occurred. The full extent of current and potentially future ground disturbance caused by the artisanal mining is still to be confirmed.

TSF, WSD, WHD and Waste Dumps

At the WSD, both the transported and residual soils tended to be more sandy and have a lower fines content and plasticity than at the TSF and WHD.

At the TSF majority of the soils are suitable to provide a smooth subgrade to the HDPE liner.

There are a number of rock outcrops associated with rounded hills across the valley floor of the TSF and at the WSD, which will need to be removed or capped.

Stability assessment have considered undrained stability of the embankments and particularly the lower strength near surface material and the extent to which this needs to be removed.

The laboratory test results indicate that the site materials are potentially dispersive.

The materials are considered suitable for embankment material for the construction of embankments providing the design incorporates measures to mitigate against the dispersive nature of the soils. The design incorporates an HDPE liner which will reduce the risks associated with dispersion.

Process Plant

The ground conditions comprise an upper layer of transported and residual soil that investigation indicates extends to a maximum depth of 3 m and can comprise more compressible material. Underlying this is more competent residual soil that becomes more competent and less weathered with depth. Rockhead is variable and modelled at 11 m depth. The process plant platform is will predominantly be located in cut, which averages approximately 2.5 m and with much of the more compressible surface soil being removed.

The ground conditions are considered suitable to support ground bearing foundations and piling, or similar approaches, are not expected to be required. However, some settlement reduction measures are likely to be required.

The calculated values of settlement indicated are generally lower than the allowable settlement values with the exception of total settlement for the main stockpile and differential settlement for the HGPR. The settlement of a number of other structures is borderline. These structures require more detailed consideration.

During detailed design, consideration will need to be given to measures to reduce foundation settlement to meet structural and operational requirements. These include:

- Undertaking settlement calculations using more detailed loading of the different stages of construction.
- The opportunity to preload some structures, where practicable. For example, the CIL tanks can be preloaded with water (hydraulic testing undertaken for a longer period).
- Replacing some of the nearer surface ground below specific foundations with compacted rockfill to increase the stress-strain modulus and reduce settlement.
- For some structures, the allowable settlement values should be reviewed.
- High plasticity clay soils are present at the site which have the potential to shrink and swell with seasonal changes in moisture content. It is recommended that foundations are founded at a minimum of 1 m depth.

Airstrip

The ground conditions at the airstrip were notably different to other parts of the site as ferricrete and lateritic gravel was commonly encountered.

Construction Materials

It is considered that low permeability fill (Zone A) and structural fill (Zone C) will be available on site from local borrow or mine waste.

It is expected that select rockfill (Zone E and Zone G) will be sourced from mine waste or from an onsite quarry. Limited amount may be obtained from rock outcrops within the dam basins.

Laboratory test results indicate that two local sand sourcescould be considered for use as drainage sand (Zone F) on a case by case basis.

Selected ferricrete, laterite and gravel colluvium are expected to be suitable for sub-base and basecourse for unsealed roads, and structural fill.

1.17.1.3 Seismicity

A probabilistic seismic hazard analysis was carried out to determine appropriate seismic design parameters for Lafigué. It is recommended that the 1000-year ARI earthquake be adopted as the Operating Basis Earthquake (OBE). The estimated peak ground acceleration (PGA) for the 1000-year earthquake was calculated as 0.026 g or 0.26 m/s^2. It is recommended that the 5000-year ARI earthquake be adopted as the Safety Evaluation Earthquake (SEE) for Lafigué. The estimated peak ground acceleration (PGA) for the 5000-year earthquake was calculated as 0.034 g or 0.33 m/s^2. A design magnitude of M6.9 earthquake at a distance of 190 km from site was nominated as the Maximum Credible Earthquake (MCE) for Lafigué. The estimated PGA for this earthquake was calculated as 0.053 g or 0.52 m/s^2.

1.17.2 Transport and Logistics

The Transport/Logistics basis for construction and operations is summarised in Sections 1.17.2.1 to 1.17.2.5, following.

1.17.2.1 Port

For goods and materials sourced from abroad, the Project and operation will be serviced by the Autonomous Port of Abidjan (APA).

1.17.2.2 Rail

Whilst the Abidjan to Ouagadougou railway lies approximately 55 km west of PE 58 in Katiola, it is not currently envisaged that this rail line will be used for construction or for the mine's operational logistics requirements.

1.17.2.3 Roads/Access

An upgrade of the existing public road/track from Koundoudougou off the B412 is currently being executed as early works during the DFS phase. This upgraded all-weather unsealed road will extend southeast for approximately 11 km before turning due south for a further 4 km towards the village of Lafigué. The Lafigué village will then be bypassed with the construction of a new 2 km all-weather unsealed access road to the main access gate at the Lafigué site.

The following internal site access roads (laterite construction) will be provided on PE 58:

- Main site entrance gatehouse to the process plant (2.4 km).
- Turn-off from process plant access road to Permanent Accommodation Camp (4.3 km).
- Turn-off from Permanent Accommodation Camp access road to Airstrip (2.4 km).
- Process plant to Tailings Storage Facility following the decant pipeline to the decant towers, and tails pipeline along the eastern boundary of the TSF to facilitate maintenance and monitoring (4.3 km length TSF Stage 1). From here access will be via the embankments.

Internal site access roads (13.4 km in total length), will be of laterite construction and consist of two 3.5 m width running lanes with a 1 m shoulder each side of the road, for a total formation width of 9 m. Associated drains will be unlined.

Allowances have also been made for minor roads and tracks (approx. 60 km) around the process plant and to other infrastructure facilities, including the Water Harvesting Dam, for operations and maintenance access.

A security access track will be constructed either side of the perimeter fence (total 24.3 km).

Mine haul roads connect the open pits, waste dumps, TSF embankment (for construction) and mine services area. The total length of the haul roads is 6.6 km. The haul roads will consist of two 12 m width running lanes, with two 1.5 m high safety bunds, for a total formation width of 30 m.

Culverts will be required in a number of locations to provide adequate drainage for road crossings of waterways.

1.17.2.4 Airstrip

An airstrip will be constructed on Site and is located 3.5 km north of the Lafigué permanent accommodation camp. Flight services will be drop-off and pick-up flights only. No refuelling facilities will be provided at the Lafigué airstrip given the distance from Abidjan[1].

The Lafigué airstrip design is summarised as follows:

- The prevailing wind direction of the Lafigué project site is SSW to NNE. The airstrip will be orientated similarly to ensure optimal operability.
- The design aircraft for Lafigué is a Pilatus PC-12.
- The airstrip will be unsealed.
- The runway running surface is 1 060 m long and 23 m wide. The surrounding runway strip is 80 m wide.
- Cut and fill operations will be required to achieve a design compliant with the International Civil Aviation Organisation (ICAO) guidelines, in particular to achieve required longitudinal profiles.
- The airstrip is designed with a gravel pavement as the nominated design aircraft (Pilatus PC-12) is gravel compatible. The pavement design is also suitable for limited passes (nominally two. flights per annum and less than 100 flights over the life of the airstrip) by the proposed medical evacuation aircraft (Lockheed C-130 Hercules aircraft).
- Formal routine maintenance of the should be performed at least quarterly and more frequently if the pavement shows signs of degradation, deformation, rutting or erosion.

1.17.2.5 Construction and Operational Logistics

Construction and operations materials and equipment will be typically transported to site by truck, using CI's public road network. Inbound construction materials and equipment sourced from overseas will utilise the APA, before being transported to Site by road. For both construction and operations, the use of the rail station at Katiola, does not form part of the DFS.

For both construction and operations, personnel residing in nearby villages will be transported to and from site using the public road network. Expatriate and non-local personnel will be flown to and from Site via Abidjan for regional and international airport connections.

In general, operational transport volumes are not high, with less than 128 trucks per month, with the greatest contribution being from fuel (57 trucks/month) and explosives/emulsions (52 trucks/month).

Gold product from Site will be transported off-Site by plane (250 to 300 kg consignments).

[1] Endeavour's Ity site has hangers and refuelling facilities catering for its Côte d'Ivoire aircraft.

1.17.3 Power

1.17.3.1 Project Grid Power Supply

ECG Engineering Pty Ltd (ECG) have indicated that power quality on the CI 225 kV transmission network is good, and power availability should be in excess of 98%. Whilst low rainfall/dam levels and other factors led to 'load shedding/'power rationing' in-country in 2021, ECG state that heavy industry are likely the last to be load shed (ECG Engineering, 2020).

The Project involves the upgrade of the Dabakala Substation, involving extending the existing 225 kV bus, adding a 225 kV transmission line feeder bay, construction of 33 km of 225 kV single circuit lattice tower transmission line, and constructing a substation on PE 58. The Lafigué Substation will be owned and operated by Compagnie Ivoirienne d'Électricité (CIE) and the Project will take a 225 kV tariff metered feeder, install a 225/11 kV transformer in their substation and take an 11 kV feeder to the plant main 11 kV switchboard.

Currently the full project cost for the transmission line is borne by the mine developer. However, the commercial terms for the supply of power are still to be negotiated.

1.17.3.2 Power Demand, Generation and Power Management

Generation capacity on Site is limited to electrical loads not connected to the Site power distribution network and Emergency Power Generation. Further, whilst solar generated power has been considered, it has not been incorporated in the Project/Operational energy mix.

The Site has a connected grid-based load of 25.5 MWe and consumes 148 GWh/a of power.

For techno-economic reasons, the following electrical loads will be met with local diesel power generation capacity (0.825 MWe), rather than from Site's power distribution network:

- Water Harvesting Dam pumping station.
- Remote Borefields.
- Mine pit dewatering.
- Explosives Storage and Emulsion Plant.
- Airstrip.
- Remote security control guardhouses.

An emergency backup power station using diesel generators will be provided for the Lafigué Project to supply critical loads during planned and unplanned outages on the grid.

1.17.3.3 Power Distribution

An 11 kV overhead power line fed from the plant main 11 kV switchboard will distribute power to the following remote infrastructure loads:

- Gendarmes Barracks and Main Gatehouse area.
- Dog Kennels and Canine Caretaker's Accommodation.
- Permanent Accommodation Camp.
- Exploration Camp.
- Sewage Treatment Plant.

- MSA.
- Tailings Storage Facility.

1.17.4 Water Management

1.17.4.1 Overview

Knight Piésold Pty Ltd (KP) completed the DFS design of the following Site water management infrastructure, for the Project:

- Tailings Storage Facility (TSF)
- Surface water management system including:
  - Water Harvesting Dam (WHD)
  - Water Storage Dam (WSD)
  - Sediment management systems (SCS)

The closest perennial water source is 23 km from Site (the N'zi river) and thus, water will primarily be sourced from Site wet season surface water run-off, and to a lesser degree from ground and pit water. This makes the sizing and the balancing of water between the WHD and WSD critical for the sustainable operation of the mine. Both ground and surface water are expected to be of a quality suitable for the intended use, with only minor treatment required.

The basis for the design, operation and closure of these facilities is discussed in Sections 1.17.4.2 to 1.17.4.8.

1.17.4.2 Climate

Daily precipitation records (1922 to 2000) from the Dabakala meteorological station (25 km north east of the Site) were used for the short-duration climatic analysis and summed to produce monthly and annual totals for long-duration climatic analysis. The wet season (>100 mm/month) typically starts from the beginning of May running through to the end of September. The mean average rainfall (MAP) and mean average evaporation (MAE) rate for the project area is 1119 mm/a (Table 18-19) and 1373 mm/a (Table 18-21), respectively.

1.17.4.3 Water Balance Modelling

To understand (and control) the flow of water around the site, a water balance model was developed by KP. Key findings from the water balance modelling are as follows:

- The TSF is designed to hold the tailings plus the design rainfall conditions, and thus has sufficient storm water storage capacity for all design storm events and rainfall sequences.
- The TSF supernatant pond volume peaks in September each year (at the end of the wet season), before returning to the minimum operating pond volume during the subsequent dry season.
- Decant return/process water shortfall is expected to occur under average and design dry climatic conditions.
- All make-up water requirements can be provided by the WSD reservoir, supplemented by the WHD for design dry conditions. It is necessary that the WSD is completed early to allow a full wet season for filling prior to commissioning.
- A WHD capacity of 0.54 Mm^3 is required to reduce the risk of shortfalls under design dry conditions.

- A WSD storage capacity of 1.2 Mm^3 is required to provide sufficient make-up water, supplemented by an abstraction rate of 536 m^3/h from the WHD.[1]

1.17.4.4 Water Harvesting Dam (WHD)

The WHD will be the primary water collection structure and will be able to store up to 0.54 Mm^3 of water at the maximum operating level. The WHD has a catchment area of 40 593 ha and when the reservoir is at its maximum level, the reservoir surface area will be 51 ha. The water collected in the WHD will be pumped to the WSD, with a view to filling the WSD reservoir to its maximum storage level prior to each dry season.

An ANCOLD dam failure consequence category (ANCOLD, 2019) of 'Significant' was determined for the WHD on the basis of a potential PAR in the range of '≥1 to <10' and a Severity Level of 'MEDIUM', primarily due to the anticipated business impacts if the WHD were to fail (primarily temporary loss of water supply for the project).

1.17.4.5 Water Storage Dam (WSD)

The Water Storage Dam (WSD) will be the primary storage pond for clean process water on site and will be able to store up to 1.2 Mm^3 of water at its maximum operating level. The WSD has a catchment area of 219 ha and when the reservoir is at its maximum level, the reservoir surface area will be 22 ha. The water collected in the WSD will be pumped to the plant to supply plant raw water requirements and process make-up water requirements.

An ANCOLD dam failure consequence category (ANCOLD, 2019) of 'Significant' was determined for the WSD on the basis of a potential PAR in the range of '≥1 to <10' and a Severity Level of 'MEDIUM', primarily due to the anticipated business impacts if the WHD were to fail (primarily temporary loss of water supply for the project).

1.17.4.6 Sediment Control Structures (SCSs)

Sediment control structures (SCSs) are sediment dams that will be constructed in the downstream reaches of catchments impacted by site infrastructure. The SCSs were designed to limit the maximum water depth to 2 m for safety reasons (drowning risk). Source control will be used to reduce the amount of sediment generated. In some instances, the site access roads will also operate as SCSs, with a culvert structure operating to convey flow beneath the site access road (installed within an abutment of the SCS embankment) and off-site downstream of the SCS.

1.17.4.7 Monitoring

A total of two groundwater monitoring stations will be installed downstream of the TSF to facilitate early detection of changes in groundwater level and/or quality, both during the operating life and following decommissioning.

Standpipe piezometers will be installed in the TSF and WSD embankments and vibrating wire piezometers will be installed in the WHD embankment to monitor pore water pressures at several locations within the embankments to ensure that stability is not compromised.

Survey pins will be installed at regular intervals along the TSF, WSD and WHD embankments crest to monitor embankment movements and assess effects of any such movement on the embankment.

[1] Late stage DFS updated, amended to 1.8 Mm3 and 605 m^3/h, surface areas quoted will subsequently change.

1.17.4.8 Closure

At the end of the TSF operation, the downstream faces of the embankments will have an overall slope profile of approximately 3.5H:1V. The profile will be inherently stable under both normal and seismic loading conditions. The embankment downstream face will be re-vegetated once the final downstream profile is achieved.

Upon closure the final tailings surface will be capped with a soil cover. The following cover system was assumed:

- Mine waste capillary break (500 mm).
- Low permeability fill layer (300 mm).
- Topsoil growth medium layer (200 mm).
- The finished surface will be shallow ripped and seeded with shrubs and grasses.

If required, the WSD and/or WHD may be decommissioned by breaching the embankments to achieve full drainage of the reservoir. However, there is an option to work with the local communities to use the dams after mine closure. SCSs will be decommissioned by breaching the embankments to achieve full drainage of the reservoir.

1.17.5 Site Services

Site services will be provided for the Lafigué Project as outlined in Section 18.6.

1.17.5.1 Security and Fencing

Security infrastructure on Site, includes but is not limited to:

- Security fencing of facilities.
- Perimeter monitoring of process plant.
- Targeted monitoring of high-risk areas.
- Access control to high security areas for personnel and vehicles.
- Remote monitoring of operations (via CCTV).

1.17.5.2 Water Systems

The water supply basis for the Project and operations are summarised below:

- Raw water will be primarily sourced from a water harvesting dam (WHD) located approximately 9 km southwest of the Plant. The WHD collects surface water runoff during the wet season. Water will be pumped via a pipeline to the WSD located near the Plant.
- Potable water will be generated by treatment of ground water sourced from dedicated borefields proximate to the points of use. Potable water treatment plants will be provided at the Plant, gendarmes barracks and permanent accommodation camp.
- Process plant raw water will be reticulated from the WSD to the Plant raw water tank and onto the MSA facilities.
- Dust suppression water systems will include water sprays in process plant and water spraying of roads and earthworks features using water trucks.

1.17.5.3 Bulk Fuel Storage, Distribution and Dispensing

The fuel supply basis for the Project and operations is summarised below:

- Diesel fuel will be transported by road to site using bulk fuel road tankers that will be unloaded at a bulk fuel storage facility at Site (circa two to three trucks per day). Diesel consumption will peak in 2029 at 29 100 m^3/a.
- Fuel will be distributed on Site using mobile refuelling trucks to supply diesel for remote fuel requirements including diesel power generators, plant diesel tanks, waste incinerator, explosives emulsion plant and in-pit mine fleet refuelling.
- A vendor supplied bulk fuel storage (1300 m^3 total storage capacity) and pumping system will be supplied as part of the diesel fuel supply contract. This facility will be located between the MSA and Emergency Backup Power Station, and will include bulk storage tanks (2 x 600 m^3 tanks) to provide approximate 18 days onsite storage capacity.

1.17.5.4 Waste Management

Facilities have been provided for the management of non-mining and process waste at Site. Where applicable certain wastes will be moved off-Site by licensed third party service providers.

1.17.5.5 Fire Detection and Protection

Fire detection and protection systems employed on Site are summarised below:

- Firewater system – Firewater storage will be provided in the form of a protected firewater reserve built into the raw water supply tank. Firewater at the required flow and pressure will be provided by one electric motor driven pump and one diesel motor driven pump. Firewater protection distribution piping will form a closed loop around the main Plant area. Single branch lines from the ring main will provide firewater to nearby facilities on Site.
- Process Plant Facilities – Portable fire extinguishers and hose reels will be provided at appropriate intervals throughout the plant for the primary intervention of small fires. Fire hydrants will be provided to cover conveyor belts in open, accessible areas.
- Electrical Equipment and Switchrooms - The plant electrical switchrooms will incorporate Very Early Smoke Detection Apparatus (VESDA) fire detection systems. Smoke sensing input points will be located in each individual high voltage panel or MCC tier. Signals from the VESDA fire detection systems will be wired to the PLC located in each switchroom, which will annunciate on the control room operators screen. CO_2 fire extinguishers will also be provided inside all switchrooms.
- Buildings – Portable fire extinguishers and smoke detectors wired to local alarms/beacons in buildings with regular human occupation.
- Fuel Storage – Diesel storage tanks with capacity greater than 38 m^3 will be provided with at least two fire hydrants, located to spray directly onto all exposed tank wall surfaces. The fire hydrants will be provided with a permanently mounted foam eductor monitor nozzle. The bulk fuel storage facility will have a fire suppression system as nominated by the fuel supply vendor.

1.17.5.6 Non-Production Waste Management

Systems in place for the management/disposal of non-production wastes are summarised below:

- Separate packaged sewage treatment plants (STP) will be provided to process daily sewage waste from the Process Plant and MSA facilities; Gendarmes Barracks and Main Gatehouse area; and the Permanent Accommodation Camp.
- A diesel-fired waste incinerator facility will be provided within the process plant high security fenced area.
- A waste management facility and salvage/recyclable yard will be established on site for storage and managing various waste materials.
- A waste land fill will be established close/adjacent to the waste rock dump.

1.17.5.7 Communication Systems

CI has a well-developed fibre and cellular network in-country and close to Site. A microwave tower installed At Site will provide external connectivity with one or more third-party service providers.

Communication systems on site are summarised below:

- For the plant and general offices, internal communications and IT services will be distributed via a site-wide high-capacity fibre optic network. The backbone of the system will be single mode fibre optic distributed throughout the site via a fibre optic cable (OPGW) run along the overhead power lines.
- A backup radio link will be deployed from the Main Administration Offices to the Camp communications tower in case of a failure on the optic fibre link.
- The IT server infrastructure will consist of a main datacentre and storage at the Administration Building, plus a replication link to the MSA Offices IT Room for storage redundancy.
- On-site general radio communication will be undertaken via hand-held radios, with a central radio control located within a communications control centre. Communication towers will be provided to support the site-wide radio communications network.

1.17.5.8 Control System

The approach to process control on site is summarised below:

- The general control philosophy for Plant and Site process infrastructure, will be one with a high level of automation and remote control. Instrumentation will be provided where required to measure and control key parameters.
- The main plant control room will house two PC based operator interface terminals (OIT). The control room is intended to provide a central area from where the process infrastructure is operated and monitored, and from where the regulatory control loops can be monitored and adjusted.
- All key process and maintenance parameters will be available for trending and alarming on the process control system (PCS). The PCS that will be used for the process infrastructure will be a programmable logic controller (PLC) and SCADA based system.

1.17.6 Buildings, Stores, Workshop and Ancillary Facilities

1.17.6.1 General Mine Buildings, Stores, Workshops and Ancillary Facilities

The buildings, stores, workshops and ancillary facilities to be provided for the Project are outlined below:

- General infrastructure buildings including: the main administration offices; clinic/first aid and emergency response buildings; main warehouse; light vehicle workshop; airstrip arrival/departure building; social performance offices; main entrance security gatehouse; security command posts/guardhouses; and security control centre.
- Plant infrastructure buildings including the plant security gatehouse and change room; plant offices and control room; plant diner; plant ablutions; plant laboratory; plant workshop; and reagent stores.
- Mining support facilities will be developed by the Mining Contractor in consultation with Endeavour at the Mine Services Area (MSA). These MSA facilities will include mining offices; mining training building and simulator; canteen; change rooms, showers and ablutions; heavy vehicle mine workshop; tyre change area; mine warehouse; mine laboratory for grade control; truck washdown area; waste area; container and laydown area; heavy vehicle/equipment parking; light vehicle parking; heavy vehicle refuelling bays with fuel pumped from the nearby bulk fuel storage facility; and supporting utilities and services reticulated from the nearby process plant facilities.
- An emulsion plant and explosives storage facility will be located within a locally fenced and secured compound at the southeast end of the site, providing direct road access to the mine and MSA, whilst ensuring suitable blast separation distances to protected works facilities.

1.17.6.2 Site Accommodation

Construction Camp

A construction camp catering for 324 personnel will be installed using prefabricated flatpack buildings, which will be repurposed post construction to form part of the permanent accommodation camp. The construction camp will be located within the permanent accommodation camp compound at the eastern end of the mine site, within the main perimeter fence line. The camp compound will be fenced with a security gatehouse, controlling access into this area.

Permanent Accommodation Camp

The Permanent Accommodation Camp will have capacity to accommodate up to 340 operations staff utilising the repurposed construction camp facilities. Accommodation at the Permanent Camp will be prioritised to management and more senior operations personnel.

The selected location for the Permanent Accommodation Camp is approximately 4 km east of the Plant site (approximately 8 km by road).

Gendarmes Barracks

A Gendarmes Barracks will be fenced separately and located just outside the main gatehouse entrance to site. It will provide housing for up to 48 gendarmes as required and will include basic messing and recreational facilities. The barracks will be constructed using prefabricated flatpack buildings with the majority of the facilities relocated from the starter construction camp. Supporting services such as power, water and sewage handling will be provided.

Exploration Camp

An exploration camp was established early on site to support exploration and early works activities for the Project. This camp is proposed to remain in use throughout the construction and operations phases. Permanent power and communications services will be provided to the exploration camp via overhead power lines installed in the Project execution phase.

1.17.7 Mine and Production Wastes

1.17.7.1 Tailings Storage Facility (TSF)

The TSF will comprise a cross valley storage facility formed by multi-zoned earth fill embankments, comprising a total footprint area (including the basin area) of approximately 120 ha for the Stage 1 TSF increasing to 200 ha for the final TSF. The TSF is designed to accommodate a total of 41 Mt (db) of tailings. The Stage 1 TSF will be designed for 36 months storage capacity. Subsequently, the TSF will be constructed in annual raises to suit storage requirements. Downstream raise construction methods will be utilised for all TSF embankment raises based on the S12i mining schedule. It is noted that the current mining schedule (Sc13k) requires an additional 9.6 Mt of tailings; however, the additional tonnage occurs during the final years of operation. It is estimated that the TSF can be expanded to approximately 80 Mt before impacting other site infrastructure, subject to embankment stability checks. This affords the mine the opportunity to process new ore deposits.

A downstream seepage collection system will be installed within and downstream of the TSF embankment, to capture seepage from the TSF and pump back to the embankment crest (if required), where it will be deposited back into the supernatant pond.

The TSF basin area will be cleared, grubbed and topsoil stripped and a 200 mm thick compacted soil liner will be constructed in the TSF basin area and overlain with 1.5 mm smooth HDPE geomembrane over the entire basin area. The embankment upstream face and decant tower areas will be lined with 1.5 mm textured HDPE geomembrane liner.

The TSF design incorporates an underdrainage system to reduce pressure head acting on the compacted soil and HDPE geomembrane liners, reduce seepage, increase tailings densities, and improve the geotechnical stability of the embankments. The underdrainage system comprises a network of collector and finger drains. The underdrainage system drains by gravity to a collection sump located at the lowest point in the TSF basin. A leakage collection and recovery system (LCRS) will be installed beneath the basin composite liner. Solution recovered from the underdrainage system and LCRS will be released to the top of the tailings mass via submersible pump, reporting to the supernatant pond.

Supernatant water will be removed from the TSF via submersible pumps (designed by others) located within a series of decant towers, constructed at start-up and raised during operation. The supernatant pond will be located in the eastern valley within the TSF basin. Solution recovered from the decant system will be pumped back to the plant for re-use in the process circuit.

An emergency spillway will be available at all times during TSF operation to protect the integrity of the constructed embankments in the unlikely event of emergency overflow. The closure spillway will be located through the eastern saddle, at the low point of the final tailings beach. The closure spillway will discharge into the existing drainage course downstream of the TSF. Upon closure, the TSF will be a fully water-shedding structure.

Tailings will be discharged into the TSF by sub-aerial deposition methods, using a combination of spigots at regularly spaced intervals from the TSF embankment.

An ANCOLD dam failure consequence category (ANCOLD, 2019) of 'High B' was determined on the basis of a potential PAR in the range of '≥10 to <100' and a Severity Level of 'MAJOR', primarily due to the anticipated business and public health impacts if tailings were to impact local communities downstream.

An ANCOLD environmental consequence category (ANCOLD, 2019) of 'Significant' was determined on the basis of a potential PAR in the range of '>1' and a Severity Level of 'MAJOR'.

Physical and geochemical testing of combined tailings samples derived from the different ore bodies was conducted during the study. The testing completed is typical for a DFS level design.

The rate of supernatant release for all samples sample was quick and reached typical dry densities, with a good increase due to drying and consolidation. Assuming that the facility is efficiently operated, it is estimated that the average settled density for the sample will be approximately 1.35 t/m^3.

The TSF incorporates sufficient measures for containment of tailings from the facility based on the expected tailings geochemistry.

1.17.7.2 Waste Rock

Background

The basis of the Waste Rock Dump (WRD) design and positioning is based on: the Mine Plan (SRK, 2022); the waste rock geochemical and geotechnical parameters (Section 18.8.2.2 and 18.8.2.3 respectively), and

- material handling, haulage and rehabilitation costs;
- environmental, community and visual impacts;
- topographical features;
- proposed mine infrastructure;
- resource sterilization; and
- rehabilitation requirements.

The fundamental principles of construction will be as per the Environmental Management Plan (EMP). The dump benches will be covered with a soil layer pad, docked and vegetated as soon as possible after dumping. The top surface of the dumps will be soil clad and graded back at approximately 1:200 to prevent ponding on the top cover.

Geotechnical Design Parameters

The WRD are anticipated to remain relatively stable over the long-term, with minor slope creep over an extended period. Potential environmental impacts resulting from individual batter failure of the dumps will be minimal due to wide catch berms. Geotechnical design parameters are presented in Table 18-38, Section 16.

Waste Rock Geochemistry

Based on test work undertaken (KP, 2021d), no issues are foreseen with respect to WRD contact water quality, and its planned to discharge directly to the receiving environment, after the contact water has passed through the sediment control systems (Section 18.8.2.3). Waste rock geochemistry results are discussed in Section 20.

Waste Rock Dump Water Runoff

Sediment control systems (SCS) or dams will be constructed to lessen the sediment-laden runoff to the receiving environment (Section 18.5.6).

Waste Rock Dump Design

Table 18-39 illustrates the storage capacity requirements of the various waste dumps and the ROM pad. Over the LoM, waste comprises: 88% Fresh, 7% transitional, 4% saprolite and 1% laterite material.

Table 1-24: Waste Dump Capacities

Location	Waste Tonnes Mined (Mt)	In situ Waste Volume (Mm³)	In situ Density (t/bcm)	Waste Volume Mined (40% Swell) (Mlcm)	Loose Density (t/lcm)	Waste Dump Capacity (Mm³)	Contingency (%)
Waste South and Central	**408.9**	**148.5**	**2.75**	**207.9**	**1.97**	**216.0**	**3.8**
Waste	**408.0**	**148.1**	**2.75**	**207.4**	**1.97**		
Inferred	**0.9**	**0.3**	**2.81**	**0.5**	**2.01**		
Western and RoM	**32.9**	**13.9**	**2.37**	**19.4**	**1.69**	**19.42**	**0.1**
Waste	**32.8**	**13.8**	**2.37**	**19.4**	**1.69**		
Inferred	**0.03**	**0.01**	**2.79**	**0.0**	**1.99**		
Total	**441.8**	**162.3**	**2.72**	**227.3**	**1.94**	**235.4**	**3.9**

Table 18-39 note: all data is reported on a dry basis

The current design capacity is sufficient for the 223 Mlcm of waste (assuming 30% swell with re-compaction) and allows for variations in waste tonnes and swell, with additional capacity available on the south and central dump above 420 mamsl. In addition, more dump space is open in the north, but this area was not utilised due to the higher elevation and haulage distances.

There are currently no site-based layout constraints, for the placement of waste rock.

1.18 Market Studies and Contracts

1.18.1 Markets

The forecast commodity prices and macro-economic assumptions within this Report were compiled from Endeavour's determinations, with reference to Consensus Market Forecasts (CMF). The forecasts are not directly supported by detailed analysis undertaken by recognised commodity market specialists, who typically use short/medium/long-term demand-supply-price analysis to support their determinations.

As such, all forecasts should be considered on a relative basis and compared to that reflected by the CMF. Where possible, historical data has been aggregated and reported through to March 31, 2022, and CMF were sourced from consensus data obtained in June 2022. All historical real terms data has been dated to March 31, 2022.

Gold pricing used for resource and reserve modelling (reserves: USD 1300/ozt and resources: USD1500/ozt) are reasonable and in alignment with industry norms stated in Section 19.1.2. The long-term consensus price for gold is USD 1746/ozt (nominal) and USD 1668/ozt (real) are both higher than the USD 1500/ozt priced used in the financial model.

Table 1-25 following outlines gold and silver commodity prices over the LoM, constrained to the depletion of Mineral Reserves. Silver is not reported in Reserves and is thus, not reported in the financial model.

Table 1-25: Summary Assumptions - Commodity Prices (Endeavour, (S&P Capital IQ, 2022a))

Commodity	Source	Units	2022	2023	2024	2025	2026	2027	2028	2029	2030	LTP
Real[1]												
Gold	Endeavour	USD/ozt	1785	1715	1655	1640	1610	1610	1610	1610	1610	1610
	CMF	USD/ozt	1838	1765	1732	1716	1716	1712	1712	1712	1712	1668
Silver	Endeavour	USD/ozt	15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.00	15.00
	CMF	USD/ozt	24.02	23.53	22.96	22.06	22.06	22.06	22.06	22.06	22.06	21.50
Nominal												
Gold	Endeavour	USD/ozt	1821	1749	1688	1673	1642	1642	1642	1642	1642	1642
	CMF	USD/ozt	1875	1800	1767	1750	1750	1746	1746	1746	1746	1746
Silver	Endeavour	USD/ozt	15.30	15.30	15.30	15.30	15.30	15.30	15.30	15.30	15.30	15.30
	CMF	USD/ozt	24.50	24.00	23.42	22.50	22.50	22.50	22.50	22.50	22.50	22.50

1.18.2 Fuel and Energy Pricing

Since 2018, CI has managed to artificially control the diesel price at around 0.89 to 0.92 USD/L to the mine, whilst the Brent crude price over this period varied between USD 29 and 114/bbl. It is unclear whether this level of price control can be sustained over the longer-term if oil prices stay high for a protracted period. Whilst there is currently no correlation between oil and diesel price in CI, a long-term Brent crude price of USD 73/bbl. has been used, with a corresponding diesel price of USD 0.91/L[2]

As noted in Section 5 of this Report, power generation in CI (2018) is largely a mix of natural gas-fired generation (60%) and hydropower (40%)[3]. Whilst there is a drive for more renewables and alternate energy sources, the implementation of some of these projects has not met the proposed implementation schedule. CI has approximately 10 years of natural gas reserves remaining; however, recent finds are likely to extend this. The electricity price at USD 0.112/kWh is controlled by government and given that natural gas productions is not exported presently, there is nothing to suggest that the mine will see power price rises that will be detrimental to project economics.

1.18.3 Steel Pricing

While Endeavour does not budget for the price of steel in its financial models, steel is considered a key commodity given its use in plant construction (stainless/structural steel and platework) and equipment supply (yellow kit and plant machinery).

As outlined within Section 19.1.4.4, forecasted steel/stainless steel prices currently indicate that any escalation in prices will not continue over an extended period (less than one year) and in all likelihood, should start to decrease over the longer-term. However, the impact of the ongoing Russia-Ukraine war may disrupt this forecast and long-term, the potential impact of CO2 taxes on steel prices is not clear (OECD, 2022).

[1] Real term prices as of 1 January 2022, LTP discounted at an inflation rate of 4.6%.

[2] No correlation

[3] 27% capacity factor

1.18.4 Contracts

It is Endeavour's, and by association SML's, strategy to outsource key mine primary and secondary value chain functions (Table 19-11), where it makes techno-economic sense to do so. Prior to start-up, contractors are requested to tender, with the most appropriate tender on a; technical, legal, social, and commercial basis accepted. Care is taken at the time of finalising contracts to ensure that the rise and fall formula is totally representative of the build-up of the quoted price per unit; and the prices quoted are comparable to benchmark prices from other Endeavour operations.

The hybrid business model to be employed at the SML mine needs to be further developed/refined before contracts are placed, specifically with respect to how services and facilities are to be utilised and shared between mine stakeholders and the associated charging basis. In setting up said business model, consideration will need to be given to:

- The tax provisions agreed in the Lafigué mining convention when signed;
- Human resources requirements, local labour sourcing and development and the employment of woman and/or other targeted groups;
- Social development requirements, specifically local sourcing, and the development of local businesses;
- Environmental requirements and standards;
- The size, local capacity and strengths and weaknesses of each contractor/service provider;
- Minimising the duplication of roles across the mine, where there is no good rationale for doing so; and
- Leveraging group buying power to negotiated better terms, based on economies of scale in-country.

As of the 'Effective Date' of this Report, no contracts have been entered into for operations and only one formal tender has been issued for negotiations.

With respect to gold sales, Gold Dore (approximately 92% m/m Au) produced at the mine site, will be transported by air (250 to 300 kg consignments by Brink's Inc.)[1] to Switzerland (Zurich) for refining by Metalor Technologies SA (Metalor). Gold sales are contracted though one of the three following entities:

- METALOR Technologies SA.
- StoneX Group Inc.
- Endeavour's Syndicate Banks.

Gold production will be sold on the spot market, with no plan currently to hedge any sales.

1.19 Environmental Studies, Permitting and Social or Community Impact

1.19.1 Introduction

An Environmental and Social Impact Assessment (ESIA) study, dated February 2021, was carried out for the Lafigué Project by a Cote d'Ivoire (CI) based environmental and social consultancy, Cabinet ENVAL (Enval). The findings of the ESIA are based on extensive environmental and social specialist investigations carried out from 2019 to early 2021 on the prefeasibility mine plan, and layouts prepared by Endeavour.

[1] Two shipments per month.

This environmental and social (E&S) chapter has been compiled based on the investigations and outcomes of the ESIA (2021).

1.19.2 Mining and Environmental Permitting

An Environmental Authorisation was issued for PE 58, with the approval of the ESIA on 18 February 2021.This ESIA formed the basis of the issuance of PE 58 (the Lafigué mining License) to LMCI and latterly to SML.

The ESIA characterised the biophysical and socio-economic baseline conditions of the Project's Area of Influence (AoI) or Study Area, and subsequently identified and quantified potential negative and positive impacts for the Project. An environmental and social management plan (ESMP) and Mine Rehabilitation and Closure Plan (MRCP) were developed with commitments of implementable actions to avoid/minimise adverse environmental and social impacts during the execution and closure of the project.

1.19.3 Environmental Baseline Setting

CI is situated in the Sudanese climate region, which is characterised by warm and humid conditions, with an annual average temperature of 28°C and annual average rainfall of 800 mm. The wet season largely occurs from April to October with average annual temperatures ranging from 24 to 28°C while the dry season occurs from November to February and is dominated by the harmattan, a dry, cool wind that blows from the Sahelian zones.

The project occurs in the Sudanian terrestrial ecoregion which is typically characterised by wooded savannas, shrubby savannas. The savannas generally have a woody component, with trees growing among the tall grasses. Gallery and riparian forests are typical, and run along permanent or temporary stream networks.

Five distinct vegetation communities were found in the Study Area, namely: wooded savannahs, grassy savannahs, gallery forests, fallow land and cultivated lands. Infield surveys confirmed that large portions of the Study Area's terrestrial vegetation have been severely degraded by subsistence agriculture and artisanal and small-scale mining (ASM) activities. Despite the degradation, some sensitive species still exist in the area. Six threatened/protected and one endemic floral species were found. Faunal diversity included recordings of 23 large mammal, 129 avifaunal, seven amphibian, and eight reptile species. Of note, this includes the Common Patas Monkey (listed as Near Threatened by the International Union for Conservation of Nature (IUCN)) and Black-bellied Pangolin (listed as Vulnerable by the IUCN).

Although wetland/riparian habitat was not specifically assessed, it is likely to be present in the AoI and offers considerable ecosystem services that support floral and faunal species. A level of degradation is expected as wetlands are commonly used for cultivation and ASM activities, which impacts the overall biodiversity value of the area.

The Study Area is associated with the N'zi River which is one of three major perennial systems in the region. The N'zi drains the Study Area through its tributaries. The Nz'i River at its closest point is 8 km from the western edge of PR 329 and 15 km from the southwest edge of PE 58. Sampling for aquatic biodiversity in the associated tributaries up- and downstream of the Study Area, identified 17 fish species during the dry season and 30 species in the rainy season. Aquatic macroinvertebrate sampling in the dry season yielded 11 taxa belonging to 11 families, seven orders and three classes. In terms of water quality, the samples taken generally comply with water quality standards for domestic uses. Groundwater is the main source of drinking water for people in the area, and the water quality results revealed that groundwater is generally clean. Metals such as manganese, iron, and zinc are below WHO drinking limits, while arsenic, nickel, lead and chromium are below detection limits.

1.19.4 Social Baseline Setting

The Project is located in the administrative region of Hambol and the Dabakala Department/District. The determined AoI comprises five villages/localities, namely: Village Lafigué, Village Toledougou, Village Fenessedougou, Village Lognene and Village Oualeguera.

The project will result mostly in economic displacement within the immediate fence line of farmlands associated with inhabitants of these villages. The main economic activities in the AoI comprises; ASM, agriculture (subsistence), livestock breeding (subsistence), small-scale trade and handicraft. Where agriculture is commercial, this is typically associated with cashew which is exported. ASM is practised by a diverse group of people, including migrants from other regions. Provision for basic socio-economic infrastructure such as schools, health care facilities, electricity and wells/ boreholes is available in the AoI, although not evenly distributed.

No tangible archaeological and cultural heritage sites were recorded in the direct Study Area.

1.19.5 Key Environmental and Social Impacts

Site clearance for the establishment of Project infrastructure is the source of several impacts, resulting in the direct loss of undisturbed areas with consequences to; terrestrial biodiversity, soil resources and associated land capability, surface water resources and dust emissions.

The remaining habitat and any supporting biodiversity within the Study Area is confirmed to be significantly affected by anthropogenic activities, thus the Project will contribute to a cumulative impact. The confirmed presence of several threatened/ protected floral and faunal species suggest that significant biodiversity value may still be present within the area despite the level of degradation.

Based on the established geochemical characterisation of waste rock and tailings material, these waste streams are not expected to result in significant pollution impacts to surface- and groundwater, however some contamination is possible, as well as potential sedimentation emanating from these facilities and therefore, the facilities should be well managed and monitored.

Open pit mining in the operational phase of the mine's life cycle will have significant impact on the landscape. In terms of nuisance impact (air quality, noise and visual) are expected issues, but may be reduced with appropriate mitigation measures. Significant impacts are related to water and include deterioration of surface and groundwater quality and quantity. Economic displacement is a significant adverse social impact identified for the Project, which even with the implementation of mitigation measures including a Relocation Action Plan (RAP), will remain a significant impact with long-term effects.

The Project is connected to the national grid, which relies on both renewable and fossil fuels for power generation. Current and future generation sources are discussed in Section 5 of this Report.

Key energy and CO_2 metrics/Key Performance Indicators (KPI's) for the Project and for operations are summarised in Table 1-26.

Table 1-26: Key Energy and Emissions Metrics for the Project and Mine (Issiyakou, 2022), (Thomson, 2022)

Area	Units	PFS	DFS Total/Avg
Production			
• Tonnes mined (Open Pit)	Mt (db)		491.615
• Tonnes Processed (Total)	Mt (db)		49.813
• Gold Produced	koz		2584
Emissions (CO_2)			
• Total (S1 & S2)	kt CO_2-e		1538
• Total (S1 to S3, excluding C2 and C4)	kt CO_2-e		1921
Emissions Intensity (per 'tonne' mined)			
• Total (S1 & S2)	t CO_2-e/t		0.0031
• Total (S1 to S3, excluding C2 and C4)	t CO_2-e/t		0.0039
Emissions Intensity (per 'oz' of gold produced)			
• Total (S1 & S2)	t CO_2-e/oz	0.36	0.595
• Total (S1 to S3, excluding C2 and C4)	t CO_2-e/oz		0.743
Energy used (S1 and S2)	GJ		18 993 112
• Energy Intensity (per 'tonne' processed)' (S1 and S2)	GJ/t		0.381
• Energy Intensity (per 'oz' of gold produced) (S1 and S2)	GJ/oz	6.99	7.35

Table 1-26 notes:

- S1 – Scope 1 emissions
- S2 – Scope 2 emissions
- S3 – Scope 3 emissions.
- C2 – Category 2 emissions (Capital Goods) for Scope 3 (not included and/or not applicable)
- C4 – Category 4 emissions (Upstream Transportation & Distribution) for Scope 3 (not included and/or not applicable)
- Scope 3 emissions are forecasts/general estimates only and subject to refinement.

There are number of reasons for the change in metrics between the PFS and DFS, including but not limited to:

- a DFS has a greater level of technical definition than a PFS, particularly around power and energy/fuel usage;
- errors and omissions; and
- the drop in the overall weighted average gold grade between the PFS (2.0 g/t) and DFS (1.69 g/t).

The cost estimate for the Closure Bond was developed at a conceptual level, using the Endeavour standard closure cost model and the historic PFS mine plan and layouts. The MRCP considers a two-year 'Pre-closure Stage' (2034 to 2035), followed by a 'Closure Stage' (2036 to 2037), and a final 'Post Closure Stage' that starts in 2038 and runs for a period of five years (2038 to 2042).

The holder of the exploitation permit retains civil liability for damages and accidents that could be caused by the mine over the five (5) year 'Post Closure Stage'.

The closure plan needs to be updated to a definitive feasibility study level and additional studies and characterization of the impacted area, will provide additional information that will allow the evaluation and detail of proposed closure actions, that should be incorporated in future MRCPs.[1]

The conceptual closure cost for the PFS, excluding labour retrenchment costs is presented in Table 1-27. This closure cost is also the basis for the DFS.

Table 1-27: PFS Mine Closure Cost (USD (M))

Direct Costs	**13.82**
• Pits	0.45
• Waste Dumps, Stockpiles	1.69
• Storage Facilities-Tailings, Rejects, Slimes	2.28
• Water Management	0.40
• Buildings and Infrastructure	4.75
• Processing Plant	1.81
• Roads and other disturbed areas	0.07
• Site Wide	0.06
• Social Cost	0
• Owners Cost	0
• Contingency	2.30
Indirect Costs	**10.60**
• Studies and Research	2.40
• Post Closure/Maintenance & Monitoring	2.21
• Social Costs	0
• Owner Costs	5.98
• Retrenchment costs	excluded[2]
Total	**24.39**

Positive impacts associated with the implementation of this Project include direct and indirect employment, contributions to community development projects, training and skills development, and payments of taxes and royalties which all contribute to the improvement of the local economy, both directly and indirectly.

1.19.6 Environmental and Social Management Plan

An ESMP was developed by Enval which provides mitigation and management measures which follow the mitigation hierarchy that aims to anticipate and avoid, and where avoidance is not possible, minimise, and, where residual impacts remain, compensate/offset for risks, and impacts to workers, Affected Communities, and the environment. The mitigation measures pertain to:

- Soil erosion and sediment management;
- General domestic, hazardous and mineralised waste streams management;

[1] To be updated every three years after the start of production.

[2] Estimated to be USD 2.1 M for the DFS

- Water management (including water quality, clean and dirty water separation, mine-water balance and water infrastructure maintenance);
- Nuisance impact management (including dust, noise and visual amenity);
- Reduction of Greenhouse Gas (GHG) emissions and climate risks management;
- Biodiversity conservation and management (including alien/exotic species management);
- Land acquisition and livelihood restoration management;
- Community and occupational health, safety and security management;
- Community development and procurement management (including local skills development);
- Community engagement procedures (including grievance mechanisms);
- Archaeological and cultural heritage preservation and management (including chance finds procedures);
- Emergency preparedness and response management;
- Spill response management; and
- Rehabilitation and closure management.

1.19.7 Conclusion and Recommendations

The Project will generate adverse environmental and socio-economic impacts, including economic displacement of surrounding communities. The local area is characterised by extensive ecological degradation, some inadequacies in the provision of socio-economic infrastructure and services. Should this Project proceed and the ESMP effectively implemented, opportunities for sustainable investment and social benefits exist.

Further detail on 'Interpretations, conclusions and risks', and 'recommendations' are presented in Sections 25.1 and 26.1, respectively.

1.20 Capital and Operating Costs

1.20.1 Operating Cost Estimate

Operating costs for the Lafigue DFS have been built up from individual cost elements within each business cost centre; and reported by year. The basis for the operating cost estimate is:

- 'Schedule 13' mining schedule presented in Section 16.
- 4 Mt/a (db) Plant, tailings facility and other supporting infrastructure on Site.
- Operating mine life of 13 years as per the current mine plan (Schedule 13).

Operating costs are presented in US dollars (USD) based on input pricing from the second quarter of 2022 (2Q22) and have an accuracy provision of ±15%. No contingency has been allowed for operating costs.

The operating costs presented reflects the direct production costs for doré bars in the goldroom safe and apply from ore through the mill for the first gold pour. Operating costs prior to this date are capitalised and reported separately as pre-production costs. Additionally, the following cost elements are reported in the financial/economic analysis section and not in the OPEX estimate.

- All operating costs/government payments associated with gold sales/revenue, including gold transporting, vaulting, refining and sale; royalties and community levies.

- Ongoing sustaining capital and closure costs.
- Financing, Joint Venture charges/payments and taxes.

All reagent and consumable costing is on a DDP basis (Incoterms® 2010) and includes the statutory 2.5% regional/ECOWAS levy. As per the 2014 Cote 'd'Ivoire Mining Code, reagents, and fuel, are duty exempt from the first commercial production (not exempt from the 2.5% levy) and subject to full duties for consumables.

Corporate costs, including costs associated with regional and head offices and exploration, are not assigned/apportioned to the mine or Project.

The following major cost areas have contributed to the overall operating costs summarised in Table 21-6:

- Mining contractor costs built up from equipment fleet operating hours and fuel usage rates.
- Labour pay rates and manning, as advised by Endeavour.
- Diesel cost, as advised by Endeavour.
- Grid power cost, as advised by ECG based on CI Energies supply.
- Processing consumable prices, as advised by Endeavour (Incoterms® 2010 DDP basis).
- Plant maintenance costs factored from the capital equipment supply cost, using factors from the Lycopodium database.
- Quoted site laboratory operating costs.
- Processing consumable usage and gold recoveries based on metallurgical testwork results.
- General and Administration (G&A) costs as advised by Endeavour, based on costs for a similar in-country mine site.
- Constant average gold recoveries over the life of mine, based on testwork and the narrow average head grade range.
- Silver production is assumed to be 5.4% of the recovered gold oz. No silver resource is quoted, so any silver revenue received is an unaccounted project upside.

Table 1-28: Operating Cost Estimate and Production Summary by Year (USD, 2Q22, ±15%)

Ore Weathering/Grade	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**	**Year 12**	**Year 13**	**LoM**
Fresh (kt)	1633	2901	3891	4011	4014	4000	4000	4003	4008	4000	4000	3851	2554	**46 866**
Transition (kt)	1192	699	180	0	2	6	0	0	0	0	0	0	0	**2079**
Oxide (kt)	441	400	27	0	0	0	0	0	0	0	0	0	0	**867**
Total kt ore feed	**3266**	**4000**	**4098**	**4011**	**4016**	**4006**	**4000**	**4003**	**4008**	**4000**	**4000**	**3851**	**2554**	**49 813**
Avg. Grade (Au g/t)	2.01	1.64	1.52	2.05	1.75	1.85	1.76	1.80	1.91	1.81	1.74	1.25	0.33	**1.68**
Mining Cost (USD M)	106.4	129.1	131.8	141.1	143.6	142.0	135.4	123.0	94.4	60.2	33.3	22.5	0.1	**1262.9**
Process Cost (Incl. Rehandle) (USD M)	36.5	45.2	47.2	46.8	46.1	46.7	46.6	46.7	46.7	46.7	46.6	45.2	31.0	**577.8**
G&A Cost (USD M)	15.6	18.7	18.7	18.7	18.7	18.7	18.7	18.7	18.7	18.7	18.7	18.7	14.0	**235.4**
Total Cost USD (M)	**158.5**	**192.9**	**197.8**	**206.5**	**208.4**	**207.4**	**200.7**	**188.4**	**159.8**	**125.6**	**98.6**	**86.4**	**45.1**	**2076.1**
Gold Produced (koz)	201	201	190	251	215	226	215	220	234	222	213	147	26	**2560**
Silver Produced (koz)*	11	11	10	13	12	12	12	12	13	12	11	8	1	**138**

Table 1-28 *notes*

- *Assumed
- Project financial year for this presentation is from start Q2 to end Q1 the following year.
- Per study schedule, Year 1 is 2024.
- Year 1 tonnes reflect a typically short ramp up to nameplate production, but also a reduced number of operating months.
- Based on reduced tonnes in the final year of operations, year 13 labour and G&A costs were calculated as ¾ of a full year as advised by Endeavour.

1.20.2 Capital Cost Estimate

The capital cost estimate for the Project has been compiled by Lycopodium with input from KP on the tailings storage facility, water and drainage infrastructure, site access roads and airstrip. Endeavour, supported by SRK and ECG, has also provided project specific portions for mine establishment and facilities, infrastructure facilities, high voltage power supply and Owner's costs.

The capital cost estimate reflects the Project scope described in this study report and has been peer reviewed for acceptance by the study team. All costs are expressed in United States Dollars (USD) unless otherwise stated and are based on 2Q22 pricing. The DFS has been developed in accordance with Lycopodium's capital cost estimating procedures and has an associated accuracy provision of (-5 to +15)%.

The capital cost estimate is summarised in Table 1-29. The capital estimate presented is based on a 4 Mt/a (db) production throughput, the mine development schedule presented in Section 24, and the capitalisation of mine development costs incurred from 1 January 2022

Table 1-29: Capital Estimate Summary

Main Area	USD (M)
000 Construction Distributables	37.38
100 Treatment Plant Costs	96.61
200 Reagents and Plant Services	23.79
300 Infrastructure	84.52
400 Mining	60.36
500 Management Costs	33.97
600 Owner's Project Costs	79.93
700 Owner's Operation Costs (Working Capital)	Excl.
Subtotal	**416.56**
Contingency	43.03
Taxes & Duties	5.64
Escalation	Excl.
Estimated Total	**465.23**

The CAPEX summary presented in Table 1-29 was subsequently revised by Endeavour, considering transport savings that are being realised (commercial contracts), a change to how operational spares are incorporated in the financial model, and the removal of Project/Site early works costs expended between 31 December 2021 and 1 June 2022 (the 'Effective Date' of the DFS/Report). The revised estimate as applied in the financial model is presented in Table 1-30 following.

Table 1-30: Revised Capital Estimate Summary (Endeavour, 2022b)

Main Area (WBS Level 2)	USD (M)	Comment
Estimate Total	465.23	From Table 1-29
Transport Savings	-8.70	33% saving banked, based on updated transport costs
Spares	-2.20	Moved into working capital (Section 21.2.9.11)
Sunk costs	-6.19	Project/Site development costs incurred from 31 December 2021 and 1 June 2022 (Section 21.2.6.1)
Revised CAPEX Total	**448.14**	Applied in Section 22, Financial Analysis

A Monte Carlo analysis was conducted on the capital cost estimate and the results provided confidence that the contingency included in the estimate, previously calculated by deterministic assessment, is sufficient for a P_{80} (or better) confidence level with event modelling turned off and a P_{50} (or better) confidence level with the event modelling turned on.

1.21 Economic Analysis

The economic model show robust financial results. Applying a long-term gold price of USD 1500/oz on a flat line basis from the Base Date (Q2 2022), delivers a Project after-tax NPV5% of USD 477 M on a 100% basis; an IRR of 21%; and a 4.2-year project pay-back period.

From first gold pour (Q2 2024), gold production varies between (155 to 251) koz per 12-month period[21], over the <13-year life of mine, with a LoM AISC of USD 871/oz. The Project has a relatively low sensitivity to capital and operating costs but is sensitive to both gold price and grade.

The Issuer uses relatively conservative values for gold pricing, and as can be seen in Section 19 and 22.4, there is NPV/IRR upside if gold prices remain high.

1.22 Adjacent Properties

Properties adjacent to the Lafigué Project (defined as within a 50 km radius of the centre of PR 329), comprise a series of artisanal mining or semi-industrial claims and eleven active Exploration Licences (PR, Permis de Recherche). Geological and exploration data in the public domain is limited to the PR 575 and PR 544 permits, which are being developed by the ASX-listed Turaco Gold group and Resolute (Eburnea Project, comprising of the Bouaké North and Satama sub-projects). Work being completed on the other permits adjacent to PR329 is not in the public domain.

According to publicly available information, exploration works on the Eburnea project have highlighted the presence of gold mineralisation that occurs either in Birimian volcano-sediments at the margin of dykes within the Oumé Fetekro greenstone belt (Bouaké North sub-project) or hosted within carbonate-silica altered fine-grained sandstones of the Birimian Comoé basin (Satama sub-project). As of the 'Effective Date' of this Report, none of these occurrences have been sufficiently drilled to define Mineral Resources reported in the public domain.

[21] Excludes year 13 which is not a full year. Average gold production over year one to twelve is 212 koz/a

1.23 Other Relevant Data

1.23.1 Human Resources

During the construction phase, approximately 900 persons will be engaged onsite directly, with a further 200+ persons providing construction support services (camp, security and other). During operations, the mine will employ some 1551 persons, of which 285 will be SML employees, with the remainder being employed by contractors providing services to the mine. Of the 1551 persons, approximately 3% of the total work force are expatriates. The number of non-local Nationals employed relative to locals has not been defined.

Whilst day workers (428 persons) will work eight hours per day, five days per week; shift workers (288 per shift panel (four panels)) will generally work either eight or twelve hours per shift and not more than 75 hours of overtime per year. The actual shift system to be employed for the mine and/or by business area is still to be defined.

Whilst SML is not bound by legislative targets in CI with respect to the employment of local tribal/religious/ethnic groups; Nationals; expatriates; woman and disabled persons, SML is committed to supporting and developing local communities and CI as a whole. Thus, there will be over the coming years, a drive to reduce the number of expatriates employed, empower women (25% employment target), upskill and employ local persons, and grow local/regional procurement and by association, businesses.

In a 2012 report by PricewaterhouseCoopers (PWC) for Mines in British Columbia, it was noted that for every person employed at a mine (owner's team and contractors) a further 0.8 indirect and 0.4 induced jobs were created. Thus, in a western country, a multiplier of 2.1 could be used to determine the total number of jobs created per mine (PWC, 2012). Cordes (Cordes, 2016) noted that Rio Tinto's Simandou iron ore project (Guinea), assumed a multiplier of 6.3 to calculate the total number of jobs created (direct, indirect and induced). Other studies have noted a much higher level of induced employment in developing countries (Cordes, 2016).

1.23.2 Project Implementation

The implementation approach proposed for the Project is for Endeavour/SML (the 'Owner') to engage a principal Engineering, Procurement and Construction Management (EPCM) contactor to provide: design, procurement and construction management services for the execution of the process plant and selected infrastructure facilities, which will be handed over to the Owner's team on completion. The construction of the mine, tailings dam, water storage and harvest dams, incoming high voltage transmission line, 225 kV switchyard, camps and non-process infrastructure buildings will be either self-performed by the Owner's team or by specialist consultants/contractors engaged directly by the Owner. This project execution approach was used as the basis for the preliminary implementation schedule and the capital cost estimate developed for the DFS.

Key milestone dates for the Project are listed in Table 1-31. A high-level summary of the Project Implementation Schedule (PIS) and critical activities are shown in Figure 1-18.

Table 1-31: Key Milestone Dates

Activity	Date
Approval to Proceed with Full Project Design	04-Apr-22
Commence Procurement of Long Lead Equipment	4-July-22
Commence Process Plant Earthworks	19-Sep-22
Commence Process Plant Concrete Works	04-Jan-22
Design & Engineering Complete	14-Jul-23
Commence Commissioning	14-Dec-23
Ore to Mill	12-May-24
First Gold Product	17-Jun-24

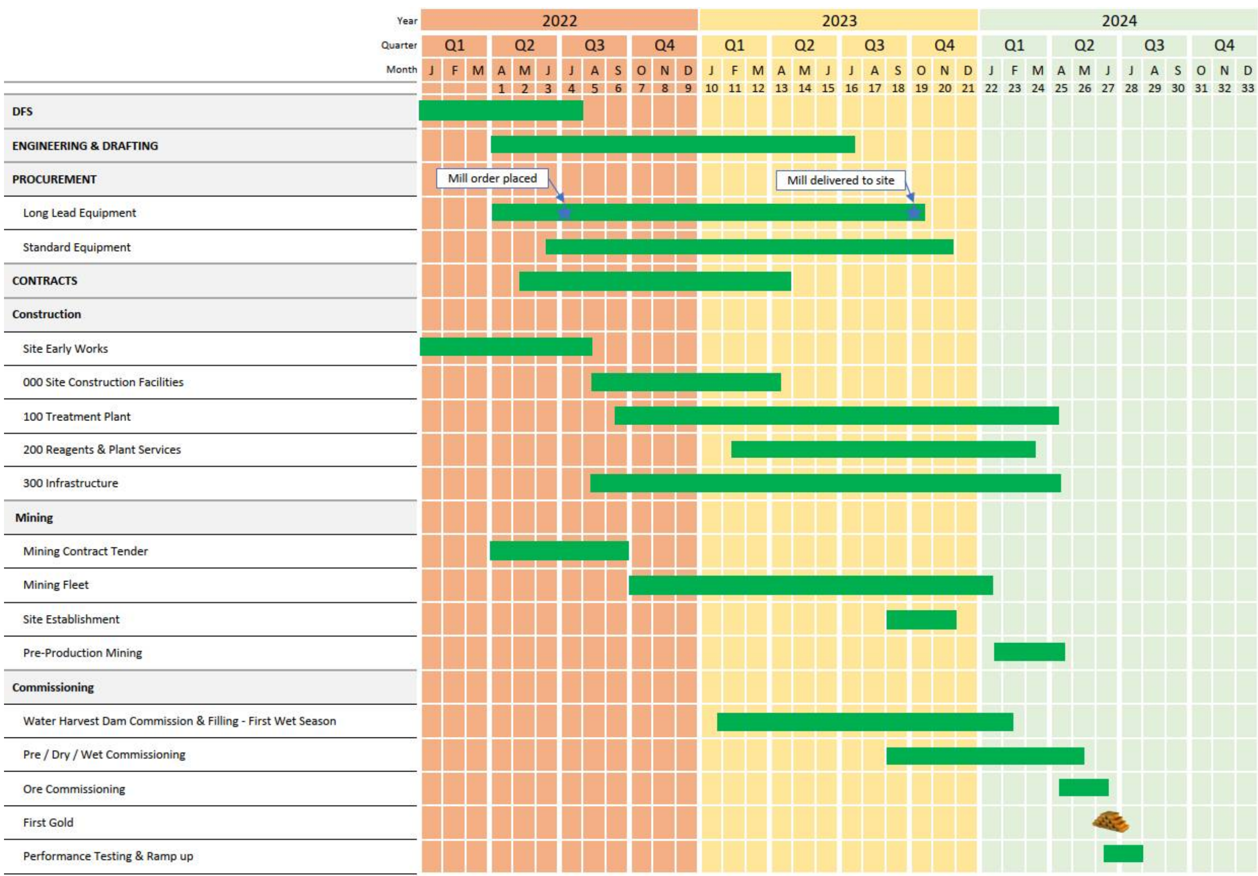

Figure 1-18: Project Implementation Schedule Summary

1.24 Interpretations and Conclusions

Interpretations, conclusions and risks are discussed in part in this section and described more fully in Section 25.

1.25 Recommendations

Recommendations/forward work programme (FWP) activities covering the period from the end of the DFS to the Project execution phase, and over the Mine's operational life cycle are discussed fully in Section 26.

2. INTRODUCTION

2.1 Issuer

The Issuer, Endeavour Mining plc or 'Endeavour', is an established gold producer and the largest in West Africa, with operating assets across Senegal (SN), Cote d'Ivoire (CI) and Burkina Faso (BF) and a strong portfolio of advanced development projects and exploration assets in the highly prospective Birimian Greenstone Belt across West Africa.

As a member of the World Gold Council, Endeavour is committed to the principles of responsible mining and delivering sustainable value to its employees, stakeholders, and the communities where it operates. Endeavour is listed on the London Stock Exchange and the Toronto Stock Exchange, under the symbol EDV (www.endeavourmining.com).

This NI 43-101 Technical Report (the 'Report') pertains to:

- The Issuer's 80% ownership interest in Exploitation Permit (PE 58), hereafter also referred to as the 'Lafigué Mining Licence' or 'Lafigué ML'. Ownership of PE 58 is as noted below:
  - Société des Mines de Lafigué SA (SML) the permit holder;
  - Lafigué Holdings Pty Ltd (80%) and ultimately Endeavour;
  - Société pour le Développement Minier de la Côte d'Ivoire (10%) or SODEMI; and
  - the Government of Côte d'Ivoire (10%) or 'GoCI'.
- The issuer's 100% interest in Exploration Permit (PR 329), hereafter also referred to as the Fétékro Exploration Licence or 'Fétékro EL'. The permit holder for PR 329 is La Mancha Côte d'Ivoire SARL (100%) or 'LMCI' and ultimately, Endeavour.

As per the 'Effective Date' of this Report, 1 June 2022, the renewal of PR 329 is pending with the authorising authority.

2.2 Terms of Reference

2.2.1 Overview

This Report has been prepared as a Technical Report Update for the Issuer's interests in the Lafigué ML and the Fétékro EL, as per the 'Effective Date' (Section 2.8) and incorporates; the 'Lafigué Project' (the 'Project'), as defined in Section 2.2.2 following. As such, it supersedes all historical NI 43-101 Reports prepared for the 'Fétékro EL'.

This Report is not independent of the Issuer and has been prepared in accordance with; Canadian National Instrument (NI) 43-101 Standards of Disclosure for Mineral Projects and the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards, and Best Practice Guidelines (BPG).

2.2.2 Lafigué Project

In March 2021 Endeavour engaged various consultants to undertake a Definitive Feasibility Study (DFS) for the Project. The Project is on Endeavour's Lafigué Mining Licence (PE 58) or the 'Site', and is located approximately 470 km by road, northeast of Abidjan in CI.

The Project comprises an open pit mine, processing plant, and supporting infrastructure, with a processing capacity of 4.0 Mt/a (db), to produce some 155 to 251 koz/a[22] of gold (average for years 1 to 12, 212 koz/a)[23], over a <13-year life of mine (LoM).

Key mine features are as note below:

- Enabling offsite infrastructure.
  - 33 km, 225 kV Power Transmission line from Dabakala to Site.
  - Upgrade of approximately 17 km laterite access road to Site (2.5 km new).
- Open pit mine with attendant waste rock dumps and water management infrastructure;
- 4 Mt/a (db) three stage crushing, milling and Gravity/CIL Process Plant (the 'Plant').
- Downstream construct Tailings Storage Facility (TSF).
- General mine infrastructure:
  - Mine Services Area (MSA).
  - Emulsion and explosives facility.
  - General administration and plant buildings/facilities.
  - Accommodation facilities.
  - Water harvest and water storage dams.
  - Airstrip.
  - Contact water management systems.

The mine is to be developed/operated under a hybrid business model, with a number of outsourced operational contracts (Section 19).

2.3 Historical Background

The historical development of the Project with respect to key study milestones, is as noted below.

- Issuance of the Lafigué Pre-feasibility Study (PFS) completed by Endeavour/Lycopodium in February 2021.
- Issuance of the Lafigué NI 43-101 report by Endeavour/Lycopodium in March 2021, and subsequently amended and re-issued in December 2021.

Key differences between the historical PFS and DFS are as noted in Table 2-1.

[22] Excludes year 13, which is not a full year of operation. Doré grade (93±2% gold)

[23] Years are not calendar years, rather year one starts from first gold pour in Q2 2024 and runs to Q2 2025,

Table 2-1: PFS and DFS Key Parameters

Parameter	Unit	PFS	DFS
Gold Reserves (total)	Mt (db)	31.9	49.81
Gold grade	g/t	2.0	1.69
Contained gold (total)	Moz	2.05	2.714
Name plate capacity (oxide)	Mt/a (db)	3.5	
Name plate capacity (fresh)	Mt/a (db)	3.0	
Name plate capacity (combined)	Mt/a (db)	3.04	4.0
Plant LoM	years	~ 10	~ 13

2.4 Contributing Consultants

In addition to Endeavour staff, consultants who contributed to this Report are as noted below:

- Bastion Geotechnical Pty Ltd (Bastion) (www.bastiongeotech.com.au)

  Bastion specialises in operational geotechnical management and geotechnical data and systems. Fields of expertise include: Geotechnical design and reporting for open pit and underground excavations; Facilitation of geotechnical hazard and risk management assessments; Expert review and technical audits; Peer and technical reviews; Geotechnical database setup, maintenance and review; Photogrammetric model construction; Large structure modelling; Forensic back-analysis of falls-of-ground; Geotechnical laboratory sample selection, preparation, results interpretation and reporting; Structural and rock mass domaining; Rock mass and structural (Large scale and fabric) characterisation; and, Preparation of histograms, empirical charts, and borehole logs.

- Cabinet Enval (Enval) (www.enval-group.com)

  Enval was established in 1999, with Cabinet Enval, a consulting firm specializing in the environment and agribusiness. In 2002, Enval Laboratory (a testing laboratory) was established to support the business. Today, Enval operates across West Africa, providing consulting and test work services (physico-chemical and microbiological analysis, soil, foliar and oil analysis, as well as noise and air quality measurements). Enval Laboratory has been ISO 17025 accredited since 2012.

- Digby Wells Environmental (DWE) (www.digbywells.com)

  Digby Wells is an international company providing environmental and social expertise focused on the power generation and natural resources sectors. Digby Wells was established in Johannesburg, South Africa, in 1995 and has expanded to establish six offices (South Africa, Mali, Botswana, Tanzania, London and Jersey (Channel Islands)). Furthermore, Digby Wells has numerous in-country partners to ensure compliance to local standards and requirements.

  Digby Wells employs a large team of professional, committed, and specialised environmental and social consultants covering 15 specialist fields and have completed projects in 52 countries across Asia, Africa, Europe and North and South America. In house specialist divisions and services include Environmental, Compliance, Social and Heritage, Water Geosciences, Rehabilitation, Closure, Soils, GIS and Remote Sensing, Ecology, Atmospheric Sciences and ESG Reporting and Strategy.

- ECG Engineering Pty Ltd (ECG) (www.ecg-engineering.com)

  ECG provides specialised electrical engineering services to the Mining, Utilities, Materials Handling and Industrial industries. ECG has extensive and proven capabilities across all aspects of project management, power generation, power systems, control systems, automation, plant integrity and operations support. Expertise, experience, innovation and integrity – ECG clients are supported by a team of highly qualified and dedicated professionals with worldwide experience in the design, construction and commissioning of mineral processing facilities. We deliver efficient, innovative, reliable and cost-effective solutions for our clients' project needs.

- Lycopodium Ltd (Lycopodium) (www.lycopodium.com)

  Lycopodium has provided engineering and project management services to the international mining industry for 30 years, has extensive experience in West Africa, and was the lead consultant for the Report. Lycopodium has been operating since 1992 and has offices in Australia, Africa, North America and Southeast Asia. Over this time Lycopodium has assembled a group of engineering and management professionals with expertise spanning all aspects of our delivery service, providing integrated design and construction solutions across the globe. The sectors across which Lycopodium operate is diverse, including; resources, infrastructure, and industrial processes.

- SRK Consulting (UK) Ltd (SRK), (www.srk.com)

  SRK Consulting is an independent, international practice providing focused advice and solutions to the earth and water resource industries. We offer specialist services for the entire life cycle of a mining project, from exploration to closure. Formed in 1974, SRK employs more than 1500 professionals in over 45 offices on six continents. Our specialists are leaders in fields such as; due diligence, feasibility studies, mine waste and water management, permitting, and mine closure. Among our clients are many of the world's major, medium-sized and junior metal and industrial mineral mining houses, exploration companies, financial institutions, construction firms, and government departments.

- Knight Piésold (Australia), (www.knightpiesold.com)

  Knight Piésold is an employee-owned global consulting firm that provides specialised services to the mining, power, water resources, infrastructure, and oil and gas industries. The Knight Piésold team comprises; engineers, environmental scientists, geoscientists, and technologists; who focus on creating value at every stage of a project through quality driven, sustainable solutions. Knight Piésold specialises in creating customised solutions at every stage of a project life cycle, while delivering sustainable, bottom-line results. We have led numerous award-winning projects to completion and have fostered many long-term client relationships that hold strong today.

2.5 Consultant Scope Responsibility

The key consultants engaged by Endeavour and their respective scope of work for the DFS, are listed in Table 2-2 following. Importantly, contributors, are not necessarily acting as Qualified Persons.

Table 2-2: DFS Scope Responsibility

Consultant	Scope Responsibility
Lycopodium	Metallurgical testwork supervision and results interpretation Process plant and related process infrastructure design General infrastructure buildings and services Process capital and operating cost estimation Compilation of overall capital and operating cost estimates Execution planning Overall DFS report compilation
SRK	Mineral Resource estimate Mine design and scheduling Pit optimisation Waste rock dump design Mine fleet selection Mine supporting infrastructure Mine dewatering Assessment of contractor mining rates and input to mining cost estimating
Bastion	Geotechnical Studies for mine area
KP	Geotechnical investigations and waste rock geochemistry Tailings storage facility and TSF geochemistry Water harvesting and water storage dams Surface water management and sediment management Site access roads Haul roads Airstrip Overall site layout development
Enval and DWE	Environmental and social impact studies
ECG	Power supply
Endeavour	Acquiring permits, agreements, and land tenure Geology Hydrogeology Mine dewatering cost estimates Contract mining labour numbers IT Infrastructure Mining operating cost estimate General & Administration (G&A) operating cost input Financial analysis Execution and operations planning

2.6 Qualified Persons

Table 2-3 following, provides a list of Qualified Persons (QPs), and the respective sections/subsections for which they take responsibility. If a QP takes responsibly for an entire section, it is given that they also take responsibility for the associated subsections. Further definition is provided in the 'Certificates of Qualified Persons' (Section 28). All consultants contributed to Section 27, and each consultant takes responsibility for the references they have cited in their respective sections.

Table 2-3: Qualified Persons and QP Section/Subsection Responsibilities

Name	Position	Company	Sections/Subsections
Abraham Buys (NHD, FAusIMM)	Group Manager Process	Lycopodium	1.12, 1.16, 12.5, 12.8, 12.12.2, 13, 17, 18.3.3.3, 21.2.7, 21.2.8.1, 21.2.8.3, 21.3.2, 21.3.2.2, 21.3.2.3, 21.3.2.4, 21.3.3.2, 21.3.3.3, 21.3.3.4, 21.3.3.5, 21.3.3.6, 21.3.3.7, 21.3.3.8, 21.6.3, 25.10, 25.14, 25.18.3, 26.10, 26.14, 26.18.3.
Alex Veresezan (MSc, P.Eng)	Group Manager - Mining Contracts	Endeavour	1.20.1, 12.9.5, 12.9.6, 18.7.2, 18.7.3, 18.10.3, 18.10.4, 21.2.8.2, 21.3.1, 21.3.2.1, 21.3.3.1, 21.4.3.1, 21.6.1, 25.15.3, 25.15.4, 25.18.1, 26.15.4, 26.15.5, 26.18.1.
David Morgan (CPEng, MAusIMM)	Managing Director	KP	1.17.1.2, 1.17.1.3, 1.17.2.4, 1.17.4, 1.17.7.1, 12.9.1, 12.9.2, 12.9.3, 12.9.4, 12.9.8, 12.9.9, 18.2.2, 18.2.3, 18.2.9, 18.3.2.2, 18.3.2.4, 18.3.2.5, 18.5, 18.8.1, 18.10.1, 18.10.2, 18.10.6, 25.15.1, 25.15.2, 25.15.6, 26.15.1, 26.15.3, 26.15.7.
David Taylor (BEng, CP Eng, FIE(Aust)	Senior Consultant	Lycopodium	1.17.1.1, 1.17.2.1, 1.17.2.2, 1.17.2.3, 1.17.2.5, 1.17.3.2, 1.17.3.3, 1.17.5, 1.17.6, 1.20.2, 1.23.2, 12.9.10, 12.12.1, 12.15.2, 18.1, 18.2.1, 18.2.4, 18.2.5, 18.2.6, 18.2.7, 18.2.8, 18.2.10, 18.2.11, 18.3.1, 18.3.2.1, 18.3.2.3, 18.3.3.1, 18.4.3, 18.4.4.1, 18.4.4.2, 18.4.4.4, 18.4.5, 18.6.1, 18.6.2, 18.6.3, 18.6.4, 18.6.5, 18.6.6, 18.6.7, 18.7.1, 18.7.4, 18.7.5, 18.7.6, 18.7.8, 18.10.8, 21.1, 21.2.1, 21.2.2, 21.2.3, 21.2.4, 21.2.5, 21.2.6, 21.2.9, 21.4.1, 21.4.2, 21.4.3.2, 21.4.3.3, 21.4.4, 21.6.2, 24.2, 25.15.7, 25.18.2, 25.21.2, 26.15.2, 26.15.9, 26.18.2, 26.21.2.
Dr Lucy Roberts (BSc, MSc, PhD, MAusIMM(CP))	Principal Consultant (Resource Geology)	SRK	1.13, 14, 25.11, 26.11.
Francois Taljaard (BEng, Pr.Eng)	Principal Consultant (Mining Engineering)	SRK	1.14, 1.15, 1.17.7.2, 12.6, 12.7, 15, 16, 18.8.2, 18.10.7, 25.12, 25.13, 26.12, 26.13, 26.15.8.
Geoff Bailey (BEng, FIEAust, CPEng, NPER-3, REPQ)	Principal Engineer/Director	ECG	1.17.3.1, 12.9.7, 18.4.2, 18.4.4.3, 18.4.4.4, 18.10.5, 25.15.5, 26.15.6.
Graham Trusler (MSc, Pr Eng, MIChE, MSAIChE)	CEO	DWE	1.19, 12.11, 20, 25.17, 26.17.
Silvia Bottero (MSc, Pr. Sci. Nat.)	VP Exploration, Côte d'Ivoire	Endeavour	1.5, 1.6, 1.7, 1.8, 1.9, 1.10, 1.11.1, 1.22, 6, 7, 8, 9, 10, 11, 12.2, 12.14, 23, 25.3, 25.4, 25.5, 25.6, 25.7, 25.8, 25.9, 25.20, 26.3, 26.4, 26.5, 26.6, 26.7, 26.8, 26.9, 26.20.
Stuart Thomson (MEng, FSAIMM)	Group Studies Manager	Endeavour	1.1, 1.2, 1.3, 1.4, 1.11.2, 1.18, 1.21, 1.23.1, 1.24, 1.25, 2, 3, 4, 5, 12.1, 12.3, 12.4, 12.10, 12.13, 12.15.1, 18.3.3.2, 18.4.1, 18.7.7, 18.7.9, 19, 22, 24.1, 25.1, 25.2, 25.16, 25.19, 25.21.1, 26.1, 26.2, 26.16, 26.19, 26.21.1.

2.7 Site Visits and Scope of Personal Inspection

The date of QP visits to the Lafigué ML/Fétékro EL (the 'Site'), and the associated purpose, is presented in Table 2-4 following. Further information is provided in the QP, Section 29

Table 2-4: QP Site Visit Summary

Qualified Person	Date of Visit(s)	Purpose of Visit
Abraham Buys (NHD, FAusIMM)	No site visit completed	Not applicable
Alex Veresezan (MSc, P.Eng)	16 to 19 May 2022 (3 days)	Review of the Site and facilities proposed for mining services infrastructure, as well as a Site orientation for the various mining contractors tendering the works.
David Morgan (CPEng, MAusIMM)	02 to 03 July 2021	KP QP site visit, site inspection of all KP infrastructure locations.
David Taylor (BEng, CPEng, FIE(Aust)	No site visit completed	Not applicable
Dr Lucy Roberts (BSc, MSc, PhD, MAusIMM(CP))	14 to 16 May 2021	MRE QP Site visit, validation of data collection procedures, collar positions and deposit geology
Francois Taljaard (BEng, Pr.Eng)	No site visit completed	Not applicable
Geoff Bailey (BEng, FIEAust, CPEng, NPER-3, REPQ)	13 May 2022, for two days; and 8 October 2022, for two days.	Visited site on two occasions in 2022, to review the alignment options for the transmission line.
Graham Trusler (MSc, Pr.Eng, MIChE, MSAIChE)	No site visit completed	Not applicable
Silvia Bottero (Msc, Pr. Sci. Nat.)	From 2014 to 10 February 2022	Ongoing work on the Site, as the Lafigué exploration manager
Stuart Thomson (MEng, FSAIMM)	24 to 26 October 2021 (1 full day on Site)	General orientation of the Lafigué Site and visit to the Dabakala CI Energie's Switch Yard.

2.8 Effective Dates

Key 'Effective Dates' in this NI 43-101 Report are:

- Mineral Resources: 15 May 2022.
- Mineral Reserves: 1 June 2022.
- CAPEX and OPEX Estimate and Financial Model: 1 June 2022.

2.9 Information Sources and References

This Report relies on historical and recent data generated by the Issuer, including public filings. Endeavour has engaged several specialist consultants and information from reports prepared by previous independent consultants have been utilised in the compilation of this Report. Information sources and references relied upon are discussed in the relevant sections and defined in Section 27 (References).

2.10 Units and Currency

Unless stated otherwise, Le Système International d'Unités (SI) units have been used throughout the reports.

Currencies are reported in accordance with ISO 4217, with the most commonly used currencies being; the United States Dollar (USD), the West African Franc (XOF), the Australian dollar (AUD), the South African Rand (ZAR), the Euro (EUR) and the Chinese Yuan (CNY) and the Canadian Dollar (CAD). Conversion rates from other currencies to the reporting currency (USD) is detailed in Section 21.

2.11 Abbreviations and Acronyms

Abbreviations and Acronyms used in this study are presented in Table 2-5

Table 2-5: Abbreviations and Acronyms

Abbreviation	Definition/Meaning
% w/v	per cent Weight by Volume
% w/w	per cent Weight by Weight
°	degrees
°C	Degree Celsius
µm	Micrometre (Micron)
3YDMAV	Three-year daily moving average
A	Annum
AARL	Anglo American Research Laboratories
AAS	Atomic Absorption Spectrometry
ABA	Acid base accounting
AC	Air Core
AC	Acid Consuming
ACA	Average crustal abundance
ACE	Africa Coast to Europe
adb	Air dry basis
AFD	French Development Agency
AfDB	African Development Bank
Ag	Silver
A_i	Abrasion Index
AIC	Al in Costs
AISC	All in Sustaining Capital Costs
ALS	ALS Metallurgy
AMD	Acid Mine Drainage
ANAC	Autorité Nationale de l'Aviation Civile (ANAC)
ANAGED	National Agency for Waste Management
ANARE-CI	Côte d'Ivoire Electricity Regulation Authority
ANC	Acid Neutralising Capacity
ANCOLD	Australian National Committee on Large Dams
ANDE	National Environment Agency (Agence Nationale de l'Environnement)
AoI	Area of Interest
APA	Autonomous Port of Abidjan
ARI	Average Recurrence Interval
ARSN	Radiation Safety, and Nuclear Security Authority
ARTCI	Telecommunication Regulatory Authority of Côte d'Ivoire (ARTCI)

Table 2-5: Abbreviations and Acronyms

Abbreviation	Definition/Meaning
As	Tropical savanna climate with dry-summer characteristics (Köppen climate classification)
As	Arsenic
ASM	Artisanal and Small-Scale Mining
Au	Gold
AU	Australia or African Union
AUD	Australian Dollar
Avg.	Average
Aw	Tropical savanna climate with dry-winter characteristics (Köppen climate classification)
Axb	JKTech Determined Ore Impact Parameter
BAD	Business as
bbl.	Barrel
bcm	Bank cubic metre(s)
BEAC	Banque des États de l'Afrique Centrale ('BEAC')
BF	Burkina Faso
BFD	Block Flow Diagram
BG	Bastion Geotechnical
bgl	Below ground level
BLEG	Bulk Leach Extractable Gold
Blk	Blank reference sample
BMG	aFrench Guiana Mining Bureau
BOAD	West African Development Bank
BPT	Business Patente Tax
BRGGM	Bureau of Geological, Geophysical and Mining Research
BRGM	Bureau de Recherches Géologiques et Minières
BRMA	Bureau of Mining Research in Algeria
Bt	Biotite
BUMIFORM	Bureau Minier of the France d'Outre-mer'
BV	Bureau Veritas
BW_i	Bond Ball Work Index
C	Carbon
CAA	Civil Aviation Authority (CAA)
CAPEX	Capital expenditure
Cb	Carbonate
CBD	UN Convention on Biological Diversity
CBR	California Bearing Ratio
CCTV	Closed-circuit Television
CDA	Canadian Dam Association

Table 2-5: Abbreviations and Acronyms

Abbreviation	Definition/Meaning
CDLM	Social Development Fund
CDP	Carbon Disclosure Project (formerly)
CEO	Chief Executive Officer
CFA	Coopération financière en Afrique centrale
CFMM	Compagnie Française de Mines et Métaux
CGECI	General Confederation of Businesses of Côte d'Ivoire
Chl	Chlorite
CHM	Cultural Heritage Management
CI	Côte d'Ivoire
CI Energies	Cote d'Ivoire Energies
CIAPOL	Cote d'Ivoire Agency Against Pollution (established by Decree No. 91-662 of 9 October 1991)
CIE	Compagnie Ivoirienne d'électricité
CIF	Cost Insurance and Freight
CIL	Carbon in Leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	carbon-in-pulp
CMD	Consensus Market Data
CMF	Consensus Market Forecast
CN	Cyanide in Solution
CN_{free}	Free Cyanide
CN_T	Total Cyanide
CN_{WAD}	Weak Acid Dissociable Cyanide
CO_2	Carbon dioxide
COG	Cut-off Grade
COGEMA	Compagnie Générale des Matières Atomiques
COMINOR	Compagnie Minière Or
Cont.	Continuous
COO	Chief Operation Officer
C_{org}	Organic Carbon
CoS	Change of Support
CoV	Coefficient of Variation
cP	Centipoise
CPI	Consumer Price Inflation
CR	Critically Endangered
CRM	Certified Reference Material
C-S	Shear plane (C) and Foliation Surface (S)
CSP	Corrugated Steel Pipe

Table 2-5: Abbreviations and Acronyms

Abbreviation	Definition/Meaning
CSR	Corporate Social Responsibility
CV	conveyor
CWi	Bond Crushing Work Index
d	Day(s)
db	dry basis
DBMS	Database Management System
DCF	Discounted Cashflow
DD	Diamond Core drilling
DDF	Depth/Duration/Frequency
DDP	Delivery Duty Paid (Incoterms® 2010)
DFS	Definitive Feasibility Study
DIF	Diffuse horizontal irradiation
dmt	Dry metric tonne(s)
DNI	Direct normal irradiation
DO	Dissolved Oxygen
doh	drill hole
DWE	distilled water extract
DW_i	Drop Weight Index
D*x*	Deformation event (numbered chronologically)
E&I	Electrical and Instrumentation
E&S	Environmental and Social
ECG	ECG Engineering Pty Ltd
ECOG	Economic Cut-off Grade
ECOWAS	Economic Community of West African States
EEL	Endeavour Exploration
EGC	Endeavour Gold Corporation
EITI	Extractive Industries Transparency Initiative
EL	Exploration License
EMC	Endeavour Mining Corporation
EN	Endangered
Endeavour	Endeavour Mining Plc, the issuer
ENVAL	Cabinet Enval
EPCM	Engineering, Procurement and Construction Management
ERT	Electrical Resistivity Tomography
ESG	Environment, Social and Governance
ESIA	Environmental and Social Impact Assessment
ESMP	Environmental and Social Management Plan

Table 2-5: Abbreviations and Acronyms

Abbreviation	Definition/Meaning
ESTMA	Extractive Sector Transparency Measures Act
EU	European Union
EUR	Euro
EZ	Eurozone
F	Forecast
F_{80}	80% Passing Size in the Feed
FEED	Front end engineering design
FEL	front end loader
FGO	Full Grade Ore – Excludes Marginal Ore
FIP	Fire Indication Panel
FoB	Free on Board
FOREX	Foreign Exchange
FP	Feed Phase
FR	Fresh (sulphide) material
FRP	fibre reinforced polymer
F*x*	Fold (numbered chronologically)
g	Gram
G&A	General and Administrative
G&A	General and Administration
g/L	Grams per Litre
g/t	grams per tonne
GAI	geochemical abundance indices
GATRO-CI	GATRO-Côte d'Ivoire
GB	Great Britain
GBL	GBL Process Pty Ltd
GBP	British Pound Sterling
GC	Grade Control
GDP	Gross Domestic Product
GENCOR	GENCOR Limited
GHG	Greenhouse Gas
GHI	Global horizontal irradiation
GIC	gold in circuit
GISTM	Global Industry Standard on Tailings Management
GoCI	Government of Côte d'Ivoire
GPS	Global Positioning System
GRG	Gravity Recoverable Gold
GRI	Global Reporting Initiative

Table 2-5: Abbreviations and Acronyms

Abbreviation	Definition/Meaning
GSA	general and administration
GSI	Geological Strength Index
GTI	Global tilted irradiation at optimum angle
h	hour(s)
HDPE	high-density polyethylene
HFO	Heavy Fuel Oil
HG	High Grade
Hg	Mercury
HG	High grade
HPGR	High Pressure Grinding Rolls
HR	Human Resources
HSE	Health, Safety and Environmental
HV	high voltage
I/O	input/output
IBC	intermediate bulk container
Icm	loose cubic metre(s)
ICMC	International Cyanide Management Code
ICP	Inductively Coupled Plasma
ICR	intensive cyanidation reactor
IDW	Inverse Distance Weighted
IFC	International Finance Corporation
IFEL	Intrusive Felsic Country Rocks
IGO	Intergovernmental organisation
IGO	Intergovernmental organisation
IMAF	Intrusive Mafic Country Rock
IMF	International Monetary Fund
IP	Induced polarisation
IPCC	Intergovernmental Panel on Climate Change
IPP	Independent Power Producer
IR	Industrial Relations
IRA	Inter-ramp Angle
IRR	Internal rate of return
IRS	Intact Rock Strengths
IS	In Situ
ISOCOG	In situ Operating Cut-Off Grade
IT	Information Technology
ITCZ	Inter-Tropical Convergence Zone

Table 2-5: Abbreviations and Acronyms

Abbreviation	Definition/Meaning
ITU	International Telecommunication Union
ITYH	Ity Holdings Ltd.
IUCN	International Union for Conservation of Nature
JKTech	JK Tech Pty Ltd
k	Kilo
kdmt	thousand dry metric tonnes
KE	Kriging Efficiency
kg	Kilogram
km	Kilometres
KNA	Kriging Neighbourhood Analysis
Köppern	Köppern Aufbereitungstechnik GmbH & Co.
koz	thousand ounces
koz/a	thousand ounces per annum
KP	Knight Piésold
KPI	Key Performance Indicator
kt	Kilo tonne
kV	Kilo Volt
kW	Kilo Watt
kW	Kilo Watt
kWe	Kilo Watt electrical
kWh	Kilo Watt hour
kWh/m³	Kilo Watt hour per cubic metre
kWh/t	Kilo Watt hour per tonne
kWp	Kilo Watt Peak
LAFH	Lafigué Holding
LAN	local area network
LAT	Laterite
LCRS	Leakage Collection and Recovery System
LED	light-emitting diode
LF	Leach Feed
LFO	Light Fuel Oil (Diesel)
LG	Low grade
LGA	Lerchs Grossman
LIMS	Laboratory Information Management System
LM2	Model of pulverising mill used for sample preparation
LMCI	La Mancha Côte d'Ivoire SARL
LME	London Metal Exchange

Table 2-5: Abbreviations and Acronyms

Abbreviation	Definition/Meaning
LNG	Liquified Natural Gas
LOD	Limit of Detection
LoM	Life of Mine
LoMp	Life of Mine plan
LOS	latch-off-stop
LP	low pressure
LPRM	Local Procurement Reporting Mechanism
LR	Leach Recovery
LRP	Livelihood Restoration Programme
LT	Laterite
LTP	Long-term Price
LV	low voltage
Lycopodium	Lycopodium Minerals Pty Ltd
m	Metre
M	Million
m/doh	Meters per drill hole
m^2	Square metre
m^3	Cubic metre
m^3/h	Cubic Metres per Hour
MAE	Mean Annual Evaporation
mamsl	Metres above mean sea level
MAP	Mean Annual Precipitation
Mbcm	million bank cubic metres
mbgl	meters below ground level or mBGL
MCC	motor control centre
MCRP	Mine Closure and Rehabilitation Plan
ME	multi-element
MESD	Ministry of Environment and Sustainable development (Constituted 1- July 2018)
MG	Medium Grade
MI	Measured + Indicated
MIF	Measured + Indicated + Inferred
MINEDD	Ministère de l'Environnement et du Développement Durable (www.environnement.gouv.ci)
ML	Mining License
mL	Millilitre
MI	Measured and Indicated only
Mlcm	million loose cubic metres
mm	Millimetre

Table 2-5: Abbreviations and Acronyms

Abbreviation	Definition/Meaning
MMPE	Ministry of Mines, Petroleum, and Energy
MMZ	Main Mineralization Zone
MO	Marginal Ore
MO Group	Metso Outotec Australia Ltd
Mo.	Month
MOD-SCN	Moderate scenario
Moz	million ounces
MPA	maximum potential acidity
MRC	maximum rated capacity
MRE	Mineral Resource Estimate
MSA	Mine Services Area
Mt	million dry metric tonnes
Mt/a	million dry metric tonnes per annum
Mt/a (db)	Million tonnes per annum (dry basis)
MV	medium voltage
MW	Mega Watt
MWe	Mega Watt electrical
N	Newton
N/mm^2	Newton per Millimetre Squared
na	Not applicable
NaCN	Sodium Cyanide
NAF	Non-acid Forming
NAG	Net Acid Generation
NAPP	Net Acid Production Potential
NDP or PND	National Development Programmes
NE	Nugget Effect
NER	neutral earthing resistors
NF	Inferred
NI 43-101	National Instrument (for the Standards of Disclosure for Mineral Projects within Canada)
NMC	New mining code
NO_2	Nitrogen oxide
NPV	Net Present value
NPV5%	Net present value at a discount rate of 5%
O/F	overflow
O_2	Oxygen
OBE	operating basis earthquake
OCN	Cyanate

Table 2-5: Abbreviations and Acronyms

Abbreviation	Definition/Meaning
OCOG	Operating Cut-off Grade
OEC	Observatory of Economic Complexity
OIT	operator interface terminals
OK	Ordinary Kriging
OLS	Obstacle Limitation Surface
OMC	Old mining code
OMC	Orway Mineral Consultants Pty Ltd
OP	Operational Phase
OPEX	Operating expenditure
OPGW	Optical Ground Wire
OPT-SCN	Optimistic scenario
OSA	Overall slope Angle
OX	Oxide
oz or ozt	Troy ounce
P&Gs	Preliminaries and General Costs – Contractor Distributables
P&IDs	Piping and Instrumentation Diagrams
P_{100}	100% Passing Size
P_{80}	80% Passing Size
Pa	Pascal
PAF	potentially acid forming
PAF-LC	potentially acid forming – low capacity
PAR	Population at Risk
PCC	Prelevement communautaire cedeao (Community Levy)
PCS	Prelevement Communaute Solidarte (Community ECOWAS Levy)
PCS	Process Control System
PDC	Process design criteria
PE	Exploitation Permit
PEP	Project Execution Plan
PFP	Prior to Feed Phase
PFS	Pre-feasibility Study
PGA	Peak Ground Acceleration
pH	Hydrogen Ion Exponent (Measure of Acidity of Alkalinity)
PLC	programmable logic controller
PM	post merīdiem (After midday)
PMP	Probable Maximum Precipitation
PMR	private mobile radio
Po	Pyrrhotite (FeS)

Table 2-5: Abbreviations and Acronyms

Abbreviation	Definition/Meaning
ppb	Parts per billion
PPE	Personal Protective Equipment
PPI	Producer Price Inflation
ppm	Parts per million
PPP	Purchasing Power Parity
PR	Exploration permit
PS	Performance Standard
PSD	Particle Size Distribution
PUA	Prelevement union africaine (African Union Levy)
PV	Photovoltaic
PVC	polyvinyl chloride
PVOut (Specific)	Specific photovoltaic power output
Py	Pyrite
Py	Pyrite (FeS_2)
QAQC	Quality Assurance, Quality Control
QEMSCAN	Quantitative Evaluation of Minerals by Scanning Electron Microscopy
QTZ	Quartz
Qz	Quartz
R&R	rest and relaxation
RAB	Rotary Air Blast drilling
RC	Reverse Circulation drilling
RC-DD	Reverse circulation drillhole with a diamond core tail
RD	Relative Difference
Rec	Recovery (%)
Report	NI 43-101 Technical Report
RESA	Runway End Safety Area
RGMPs	World Gold Council's Responsible Gold Mining Principles
RGPH	Recensement Général de la Population et de l'Habitat
RH	Relative Humidity
RL	Reduced Level, mamsl for Project
RNHD	'Le Réseau National Haut Débit
ROM	Run of Mine
RoR	Run of River
RoRo	Roll-on-Roll-off
RPEEE	Reasonable Prospects for Eventual Economic Extraction
RPM	Revolutions Per Minute
RW_i	Bond Rod Work Index

Table 2-5: Abbreviations and Acronyms

Abbreviation	Definition/Meaning
S	Sulphur
S^{2-}	Sulphide sulphur
SASB	Sustainability Accounting Standards Board
SAT-3/WASC	South Atlantic 3/West
Sb	Antimony
SCADA	supervisory control and data acquisition
SCC	Supplier Code Conduct
SCS	Sediment Control Structure or System?
SD	Standard Deviation
SDIIC	Regulation Of Discharges and Emissions from Installations Classified for the Protection of the Environment
SEE	Safety Evaluation Earthquake
Ser	Sericite
SG	Specific Gravity
SHE	Safety, Health and Environment
SHEC	Safety, Health, Environment and Community
SIB	Sustaining Capital
SIMP	Social Impact Management Plan
SLA	Service Level Agreements
SML	Société des Mines de Lafigué SA
SMP	Structural, Mechanical and Piping
SMPP	Structural, Mechanical, Platework and Piping
SMU	Selective Mining Unit
SNT	Société Nationale de Topographie
SO	Site Offices/Facilities
SO_2	Sulphur dioxide
SODEMI	Société pour le Développement Minier de la Côte d'Ivoire
SODEXAM	Société D'Exploitation Et De Development Aeroporto Aeronaut Et Metrologies
SOP	Standard Operating Procedure/Practice
SoR	Slope of Regression
SP	Saprolite
SPD	Social Performance Department
SPT	standard penetration test
SQL	Structured Query Language
SR	Stripping Ratio (t:t)
STP	Sewage Treatment Plant
S*x*	Foliation (numbered chronologically)
t	Tonnes

Table 2-5: Abbreviations and Acronyms

Abbreviation	Definition/Meaning
t/doh	Tonnes per drill hole
t/m³	Tonne per cubic metre
TAPP	Theoretical Acid Production Potential
TBC	To be confirmed
TCFD	Task Force on Climate-Related Financial Disclosures
TCTC	Total Cost To Company
TDRT	Tailings and Decant Return Trench
TDS	Total Dissolved Solids
Te	Tellurium
TEU	Twenty foot equivalent unit (i.e. equivalent 20' container units)
Project	Lafigué Project
TM	Trade mark
To	Tourmaline
t_{ore}	Total tonnes (ore)
TR	Technical Report or Report
TR	Transitional or Transitional material
t_{rock}	Total rock tonnes
TSF	Tailings Storage Facility
TSF	tailings storage facility
TSP	Total Suspended Particles
TSS	Total Suspended Solids
U/F	underflow
UCS	Uniaxial Compressive Strength
UNESCO	United Nations Educational, Scientific and Cultural Organization
UNFCCC	United Nations Framework Convention on Climate Change
UPS	uninterruptible power supplies
USD	United States Dollar
UV	ultraviolet
V1-V32	Vein Domains (numbered 1-32)
VAT	Value Added Tax
VEA	value engineering assessment
VESDA	very early smoke detection apparatus
VFEL	Volcanic Felsic rocks
VFR	Visual Flight Rules
VMAF	Mafic Volcanic Country Rock
VMAF	Volcanic Mafic rocks
VoIP	voice over internet protocol

Table 2-5: Abbreviations and Acronyms

Abbreviation	Definition/Meaning
VPN	Virtual Private Network
VSD	variable speed drive
VTEM	Vertical tilt-angle derivative
W	Watt
WAC	West African Craton (WAC)
Wavg	Weighted average
WBS	Work Breakdown Structure
We	Watt electrical
WFP	World Food Programme
WHD	Water Harvesting Dam
Whittle	Whittle Four-X™
WHO	World Health Organisation
WMO	World Meteorological Organisation
WMZ1	Mineralization Zone 1 (West)
WORST-SCN	Worst case scenario
WRD	Waste Rock Dumps
WSD	Water Storage Dam
XOF	West African CFA franc
XRD	X-Ray Diffraction
XRF	X-Ray Fluorescence
YoY	Year-on-Year
ZA	South Africa
ZAR	South African Rand

3. RELIANCE ON OTHER EXPERTS

3.1 Introduction

Sections 3.2 to 3.8, following, outline the areas in the NI 43-101 Report, where the contributing Qualified Persons (QPs) have relied on information provided by other experts, either within or outside of Endeavour.

3.2 Project Ownership, Mineral Tenure, Permits and Agreements

QPs, namely; Silvia Bottero, Lucy Roberts, Stuart Thomson, Francois Taljaard, David Morgan, David Taylor, Abraham Buys and Graham Trusler, have relied on information provided by Ms. Julie Blot [Secretary General West Africa, Endeavour] and Ms. Natasha Baston [Senior Corporate Counsel, Endeavour] relating to: '*Property ownership', 'mineral tenure', 'permits (exploitation and exploration)'* and the Issuers interests in '*agreements'* between the Government of Côte d'Ivoire (GoCI) and other parties (Endeavour, 2022a).

By virtue of the positions/roles held by the 'Other Experts', it is considered that the information provided is appropriate for use.

The relevant information is presented in Section 4 and used in the respective sections that the aforementioned QPs are signing off on. The information has not been independently verified and no opinion is offered in this area.

3.3 Taxes, Royalties and other Statutory Payments

QPs, namely Francois Taljaard, Lucy Roberts and Stuart Thomson, have relied on tax and other payment information provided by Mr. Mathieu Calame [Group Tax Director, Endeavour], Mr. Anicet Assamoi Djeti [Country Tax Manager - CI, Endeavour], Mr. Charles Mendy [Treasury Director, Endeavour], Ms. Julie Blot [Secretary General West Africa, Endeavour] and Ms. Natasha Bason [Senior Corporate Counsel, Endeavour], relating to '*Taxes*, *Royalties and other 'Statutory 'Payments* (Endeavour, 2022a). The contribution of each expert is as noted below (Endeavour, 2022a).

- Taxes – Mr. Mathieu Calame and Mr. Anicet Assamoi Djeti
- Royalties and Statutory License and Agreement Payments - Ms. Julie Blot and Ms. Natasha Baston
- Statutory and other payments related to foreign exchange conversion – Mr. Charles Mendy.

By virtue of the positions/roles held by the 'Other Experts', it is considered that the information provided is appropriate for use.

The relevant information is presented in Section 4 and used in the respective sections that the aforementioned QPs are signing off on.

The information has not been independently verified, and to the extent permitted by the NI 43-101, no opinion is offered in this area.

3.4 Revenue and Cost of Sales

QPs, namely Stuart Thomson, Francois Taljaard and Lucy Roberts, have relied on information provided by Mr Michael Sumares [VP Finance, Endeavour] and Ms. Veronique Jallabert [Corporate Treasury Manager, Endeavour], relating to: *'Sales Agreements', 'Gold pricing' and 'Costs of Sales'* Information' (Endeavour, 2022a).

By virtue of the positions/roles held by the 'Other Experts', it is considered that the information provided is appropriate for use.

The relevant information is presented in Section 4 and used in the respective sections that the aforementioned QPs are signing off on.

The information has not been independently verified, and to the extent permitted by the NI 43-101, no opinion is offered in this area.

3.5 Owner's Team Labour Costs

QPs, namely Abraham Buys and Stuart Thomson, have relied on information authorised by Ms. Ludivine Guth [VP Human Resources, Endeavour] relating to: 'CI Owner's Team Labour Costs'. (Endeavour, 2022d). Given that Ms. Guth is a responsible for Human Resources across the Group, it seems reasonable that the labour cost data provided can be relied upon.

The relevant information is used either directly or indirectly where relevant in the respective sections that the aforementioned QPs are signing off on.

The information has not been independently verified, and to the extent permitted by the NI 43-101, no opinion is offered in this area.

3.6 Fuel and Reagent and Consumable Pricing

QPs, namely Abraham Buys, Stuart Thomson and Francois Taljaard, have relied on information provided by Ms. Djaria Traore [VP Supply Chain, Endeavour] with respect to the long-term fuel supply price, and/or reagent and consumable pricing (Q2 2022 budgets) (Endeavour, 2022e).

By virtue of the positions/roles held by Ms. Traore, it is considered that the information provided is appropriate for use.

The relevant information is presented in Sections 19 and 21 and used where relevant, in the respective sections that the aforementioned QPs are signing off on. The information has not been independently verified, and to the extent permitted by the NI 43-101, no opinion is offered in this area.

3.7 Financial

QPs namely Stuart Thomson and Francois Taljaard, have relied on financial modelling data, including 'All in Sustaining Capital Costs' (AISC) provided by Mr. Chris Dollman (ACA) [Business Development Manager, Endeavour]. Mr Dollman is responsible for developing and reporting the financials for the Project (Chris Dollman, 2022).

The relevant information is primarily presented in Section 21 of this Report and used where relevant in the respective sections that the aforementioned QPs are signing off on.

The information has not been independently verified, and to the extent permitted by the NI 43-101, no opinion is offered in this area.

3.8 Environmental and Social

QPs namely Graham Trusler, Francois Taljaard, Lucy Roberts and Stuart Thomson, have relied on information provided by; Mr. Beh Diarrassouba [Project Manager, Cabinet Enval (Enval)], Ms. Daphnée Turcotte [Mine Closure Manager, Endeavour], Mr. Mahamadou Issiyakou [Group Manager Hydrocarbon and Energy Transition, Endeavour], Mr. Kevin Landry N'Guessan [Environmental Superintendent, SML], Mr. Adam Kouyate [Social Performance Manager, SML], Mr. Tony Kneuker [Lafigué Project Director, Endeavour], Mr. Philippe Comte [Director of the Security Department, SML] and Mr. Salah Dallali [EDV-SML-Deputy Security Site Manager].

The contribution of each expert is as noted below.

- ESIA - Mr. Beh Diarrassouba (Cabinet Enval, 2021).
- Closure Costs - Ms. Daphnée Turcotte (Endeavour, 2022c).
- Carbon dioxide emissions (CO_2) - Mr. Mahamadou Issiyakou (Issiyakou, 2022).
- Post ESIA environmental and social data - Mr. Kevin Landry N'Guessan and Mr Adam Kouyate (SML, 2022c), (SML, 2022d).
- Artisanal Mining Activity - Mr. Philippe Comte, Mr. Salah Dallali, and Mr. Tony Kneuker. (Endeavour, 2022b) (SML, 2022a), (SML, 2022b).

By virtue of the positions/roles held by the 'Other Experts', it is considered that the information provided is appropriate for use.

The relevant information is presented in Sections 4 and 20 and used where relevant in the respective sections that the aforementioned QPs are signing off on.

The information has not been independently verified, and to the extent permitted by the NI 43-101, no opinion is offered in this area.

3.9 Market Information

QP, namely Stuart Thomson, has relied on information provided by Mr Brendan Sprague (CPA) [Projects Finance Manager, Endeavour] for the sign-off of Section 19. No independent report was produced by Mr Sprague in the compilation of Section 19. The information presented is considered appropriate for its intended purpose; on the basis of the information sources used, cited and reviewed by the QP for this section. To the extent permitted by the NI 43-101, no other opinion is offered in this area.

3.10 Recovery Methods

QP, namely Abraham Buys, has relied on the following experts for Lycopodium's process plant design:

- ALS for metallurgical testwork results (Refer to section 13 for ALS testwork report references).
- OMC for comminution modelling (Orway Mineral Consultants (OMC), 2021). Whilst Lycopodium believes the parameter provided are reasonable and correct, and to the extent permitted by the NI 43-101, no other opinion is offered in this area.

3.11 References

References cited in the preparation of Section 3, are presented in Section 27 of this Report.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

Endeavour's Exploitation Permit (PE 58) and the associated Lafigué Project (the 'Project'), the subject of this Definitive Feasibility Study (DFS)/NI 43-101 Report, and the associated Exploration Permit (PR 329), are located in the north-central region of Côte d'Ivoire (CI), approximately 330 km north-northwest of the port city of Abidjan (approximately 470 km by road). The southwest corner of PR 329 lies approximately 63 km north-northeast of Bouake, the second largest city in Côte d'Ivoire (CI), and 38 km east of Katiola. The northwest corner of PR 329 lies approximately 18 km west-southwest of Dabakala, and PE 58 lies within the boundary perimeter of PR 329 (Figure 4-1).

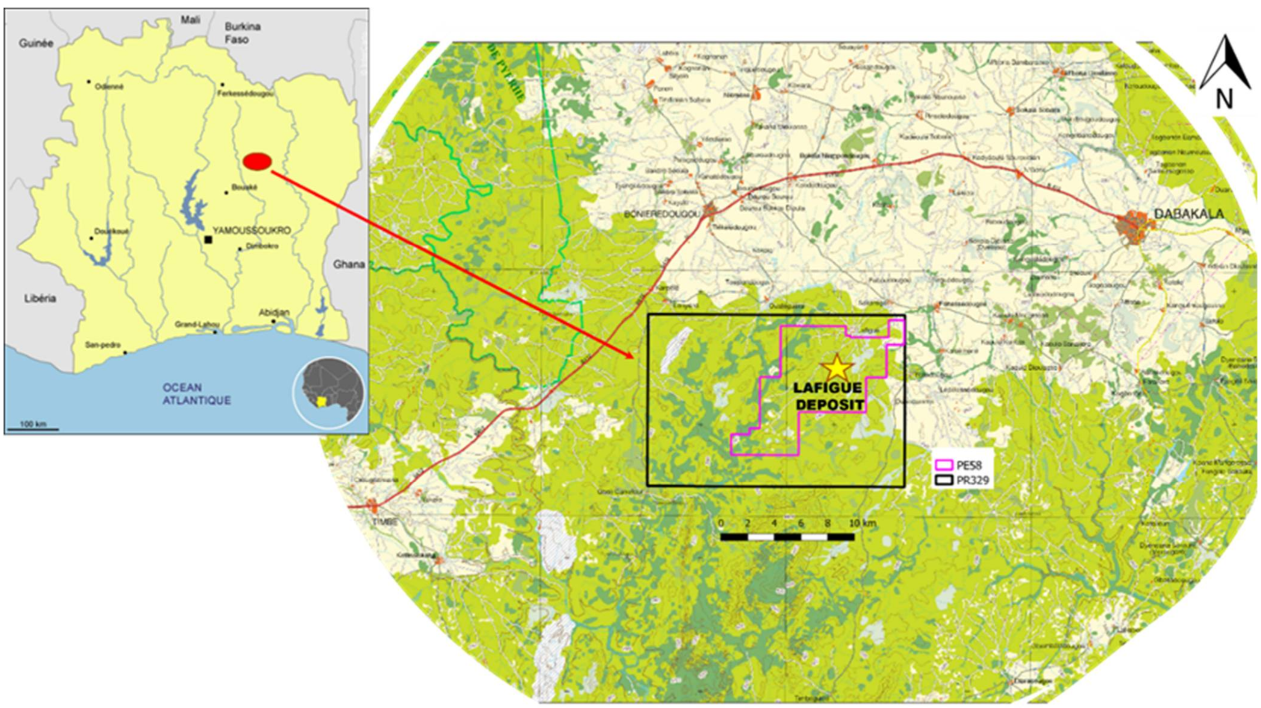

Figure 4-1: Location of PR 329 and PE 58 in CI (Endeavour, 2022)

4.2 Mineral Property and Title in Côte d'Ivoire

4.2.1 Introduction

The following section outlines the general regulatory principles regarding exploration and mining in CI and the key stakeholders. In addition, this section summarises key changes between the 1995 and 2014 Mining Codes.

4.2.2 Mining Legislation Overview

The following section outlines the applicable mining legislation and legal framework in CI and the key stakeholders. On 24 March 2014, CI's parliament approved Law No. 2014-138 adopting the new mining code (the '**New Mining Code or 'NMC**'). A Decree No. 2014-397 implementing the NMC was issued on 25 June 2014 (the '**Decree**'). The NMC replaced the former mining code (Law No. 95-553 dated 18 July 1995) (the '**Old Mining Code or OMC**').

Other laws applicable to mining activities are the: Environment Code; and the Labour Code. Additional regulation must also be scrutinised, such as the 'Decree on Surface and proportionate Fees' dated 26 March 2014. According to the Mining Code, the main regulatory bodies in the Ivory Coast are the President of the Republic and the Ministry of Mines and Geology (the '**Ministry'**).

4.2.3 2014 Mining Code

The NMC reflects the government of CI's desire to attract more investors, particularly in the gold sector, and to better regulate the mining sector as a whole. To this end, several measures have been taken, including:

- the additional profit tax under the OMC, paid by mining licensees, has been abolished;
- holders of mining permits sign a 'mining agreement/*Convention Minière*' within sixty working days of being granted a permit (Article 12);
- the State guarantees the stability of the tax and customs regime to the holder of the mining permit (Article 164);
- renewal of the research permit: under the OMC, the permit had an initial duration of three years, renewable twice, each time for two years. In addition, an exceptional three-year renewal could be granted. The maximum duration of the permit was therefore 10 years. In the NMC, the research permit has an initial duration of four years, renewable twice, each time for three years. In addition, an exceptional renewal of two years may be granted. The maximum duration of the permit is therefore 12 years;
- under the NMC the surface area of research permits was reduced from 1000 km² to a maximum of 400 km². The rationale being, to open up exploration and mining to a greater number of investors. The 1000 km² permits under the OMC will at the time of their renewal, be split into two permits of a respective size of 400 km², which will lead in principle to a loss of 200 km²;
- the State's participation in the capital of the operating company, which is not subject to financial contribution (free carry interest), remains limited to 10%. However, the NMC limits the additional participation of the State in these companies to a contributory participation that cannot exceed 15% of the share capital. Importantly, shares held by state-owned companies (Société pour le Développement Minier de la Côte d'Ivoire (SODEMI)[24]) and companies with a majority public shareholding, are not considered in determining this 15% limit; and
- under the OMC, disputes between a holder of a mining title or a beneficiary of a mining authorization and the State could only be settled, in the absence of an amicable settlement, by an Ivorian court or an arbitration tribunal 'under Ivorian law'. Under the NMC, disputes may also be settled by an 'international arbitral tribunal', provided that the parties have so provided in their mining agreement.

4.2.4 Exploration Permits (PRs)

In accordance with the NMC, an 'Exploration Permit' or 'Permit or PR' is granted by Presidential Decree. The Permit is valid for an initial period of four years and may be renewed for two consecutive periods of three years, with an exceptional renewal for a final two-year period, provided the Permit titleholder, complies with the rights and obligations set out under the NMC. At each renewal, at least 25 per cent of the original area must be relinquished, however the titleholder may elect to maintain the full area, by paying an 'Option Fee'.

[24] (www.sodemi.ci)

When applying for a Permit, the applicant must also file a costed programme of exploration work to be undertaken over the period for which the Permit is valid. Where the Permit is granted, the Permit titleholder must start exploration works within the allocated Permit area, no later than six months from the Permit's 'date of validity'; and must continue to work diligently for the prescribed term.

The Permit grants an exclusive right to the holder, to explore[25] within the Permit area (not exceeding 400 km^2), and to dispose of the products extracted during exploration activities. However, disposal is subject to a prior declaration to the Ministry; and the payment of the applicable mining duties. In addition, the Permit holder is automatically entitled to request and obtain an 'Exploitation Permit' at any time during the exploration period, provided that the Permit holder has carried out all its obligations and that a feasibility study has proven the existence of one or several economically viable deposits within the perimeter of the Permit.

4.2.5 Exploitation Permits (PEs)

An 'Exploitation Permit' or 'Permit or PE' is issued for an initial period based on the life of mine stated in the feasibility study submitted for permitting purpose, with a limit of 20 years. At its expiry, a PE can be renewed for successive periods of 10 years maximum. The holder of the Permit is required to sign a mining convention (*Convention Minière*) with the State, within sixty (60) working days from the award of the Permit. The mining convention is valid for an initial period of twelve (12) years (renewable for successive periods of ten (10) years maximum). The purpose of the mining convention, according to the Mining Code, is to stabilise the tax and customs regime.

The holder of the Permit has the exclusive right to:

- exploit the named deposits (i.e., gold) within the limits of the Permit's perimeter;
- transport or to arrange the transport of the extracted ore;
- establish the necessary facilities to condition, treat, refine and transform the ore; and
- trade the ore/product on the internal or external markets (export).

Chapter III, Article 127 of the 2014 Mining Code, prescribes conditions related to occupancy/use of the land. Said occupancy of the land gives the Permit holder the right to:

- use of the land, subject to the lawful occupant of the land receiving fair indemnity (supervision by 'Mines Administration);
- 'cut wood needed for said activity' (not sell); and
- use free waterfalls within the perimeter defined by the mining title.

Article 169, Chapter IV of the 2014 Mining Code, exempts the permit holder from:

- the exploitation levy for the withdrawal of water from the water table as part of mine drainage operations in the perimeter of the permit, during the period of validity of the exploitation permit; and
- the felling tax in the perimeter of the permit during the period of validity of the exploitation permit, provided that the woody essences are not sold.

[25] 'Research' is defined in the mining code as: all the work carried out on the surface, at depth, or airborne, to establish the continuity of mineral occurrences, to determine the existence or not of a deposit, to study the conditions of exploitation and industrial use, in order to submit a feasibility study to the Ministry.

The holder of the Permit does not have the have right to exploit other mineral/industrial commodities[26] on the permit, which are not specifically named in the exploitation permit.

The Permit is granted by right, by decree taken in Council of Ministers, to the holder of the exploration permit who has proved that there is a deposit within its exploration permit. Said proof is materialized by a feasibility study.[27]

The applicant must have complied with its obligations under the provisions of the law; and must present an application compliant with the provisions of the implementing decree of the Mining Code, prior to the expiry of the period of validity of the exploration permit under which the application for the exploitation permit is made.

Several exploitation permits may stem from the same exploration permit. The allocation of an exploitation permit gives rise to the cancellation of the exploration permit, within the perimeter of the exploitation permit. The exploration permit subsists for the remaining surface area outside the perimeter of the exploitation perimeter, up to the expiry of its period of validity.

4.2.6 Environmental and Social Requirements

Construction, mining and related activities must be carried out in such a way as to protect the environment including; rehabilitation of exploited sites; conservation of forest resources in accordance with the requirements of the Law; and conduct operations in a way that ensures the protection of the environment.

To obtain a Permit, the title holder must submit an Environmental and Social Impact Assessment (ESIA) to the Ministry of the Environment for approval. The ESIA must include an Environmental and Social Management Plan (ESMP), comprising a site rehabilitation plan as well as addressing provisional rehabilitation/closure costs. The rehabilitation plan must consider several aspects, including; cleaning of the site, dismantling and removal of mining installations, the post-rehabilitation surveillance of the site, and suggestions of how the site could be reconverted. These matters must be addressed during the exploitation period, and not just at the end of operations.

After the closure of the mine, any exploitation permit holder remains liable under civil law for damages and accidents on the site that could be caused by the former installations for five years following closure. Mining activities also fall within the scope of the Environment Code, which notably requires investors to provide an environmental report assessing the environmental impact of the project.

As previously stated, any applicant for an exploitation licence is required to provide, along with the ESIA, a 'Mine Closure and Rehabilitation Plan' ('**MCRP**'). The Closure Plan is submitted for approval to the Administration of Mines and Environment, respectively.

When changes in mining activities justify a modification of the Closure Plan, the holder of the exploitation licence is required to submit it for revision. The Closure Plan is established according to the site and the type of operation and must indicate the planned methods of dismantling and reclaiming all components of the mining facilities, including those facilities and equipment that are specified in the implementing decree.

[26] i.e., Aggregate, or other.

[27] Whilst a 'feasibility study is stated', the associated level of technical and cost development is not. Thus, in the Issuers case, the Exploitation Permit was granted on a pre-feasibility study.

The Closure Plan must also provide for:

- progressive reclamation work to be carried out over the course of operations, and not only at the end of operations; and
- post-closure environmental monitoring.

For Société des Mines de Lafigué SA (SML), the terms of the Closure/Rehabilitation Bond will be as defined in the Mining Convention (not in place as of the 'Effective Date'). The Bond will need to be in place from the 'Date of First Commercial Production' (Blot, 2022a). Additionally:

- The Closure/Rehabilitation Bond is paid annually in instalments (20% escrow and 80% bank guarantee), over a period of nine[28] years (not paid in last year, and not paid over the two-year construction Period). Thus, the effective payment is 11.11 per cent per annum.
- The escrow account is to be opened within 20 days following first commercial production, the amounts need to be updated on a yearly basis, within 20 days following the beginning of each year.
- The Bank guarantee is to be put in place within 120 days from date of first commercial production. Then the bank guarantee must be updated on a yearly basis, within 20 days following the beginning of the year.
- The Rehabilitation Bond amount is to be updated every three years.

See also Section 4.5.8.

Exploitation permit holders must draw up a Community Development Plan, jointly with local communities and administrative authorities and constitute a Social Development Fund (CDLM) for the benefit of villages identified as 'affected localities' by the ESIA. This fund is annually credited and will be used to realise socioeconomic development projects, the amount involved being deductible from the profit tax base (Section 4.5.7). The Permit holder must also develop/conduct training for Ivorian small and medium-sized enterprises, so as to increase their participation in the mining sector.

The terms of the CDLM are defined in the mining convention (not yet agreed as per the 'Effective Date' of this Report), Notwithstanding this, it may be assumed that the CDLM fund will be in place from the Date of First Commercial Production. SML will pay in year (Y+1) based on the turnover in the preceding year (Y). SML will obtain a dedicated Ministerial Order, confirming the terms of the CDLM (Blot, 2022a).

In the event of expiration, renunciation, withdrawal of an exploitation license, the perimeter it covers is released from all rights resulting therefrom, as of zero hour on the day following the expiration of its period of validity or the date of notification of the decision by the Administration of Mines. The buildings, outbuildings, shafts, galleries and, in general, all structures permanently installed for the operation, are left to the State as of right under the conditions provided for in the environmental management and rehabilitation plan for the operated sites.

[28] 12 year permit (first renewal)

4.3 Mineral Tenure, Ownership and Back-in Rights

4.3.1 Overview

Endeavour's indirect/direct interest with PE 58 dates back to 1993, with the issuance of PR 57 to SODEMI. SODEMI is a state company created in law (n° 62-82 of 22/03/1962), whose remit is to develop the mining industry in CI. Historical ownership and the associated chronological change in permits (PR 57, PR 328 and PE 58) and permit surficial area, is described more fully in Sections 4.3.2 and 4.3.3, and summarised in Figure 4-2.

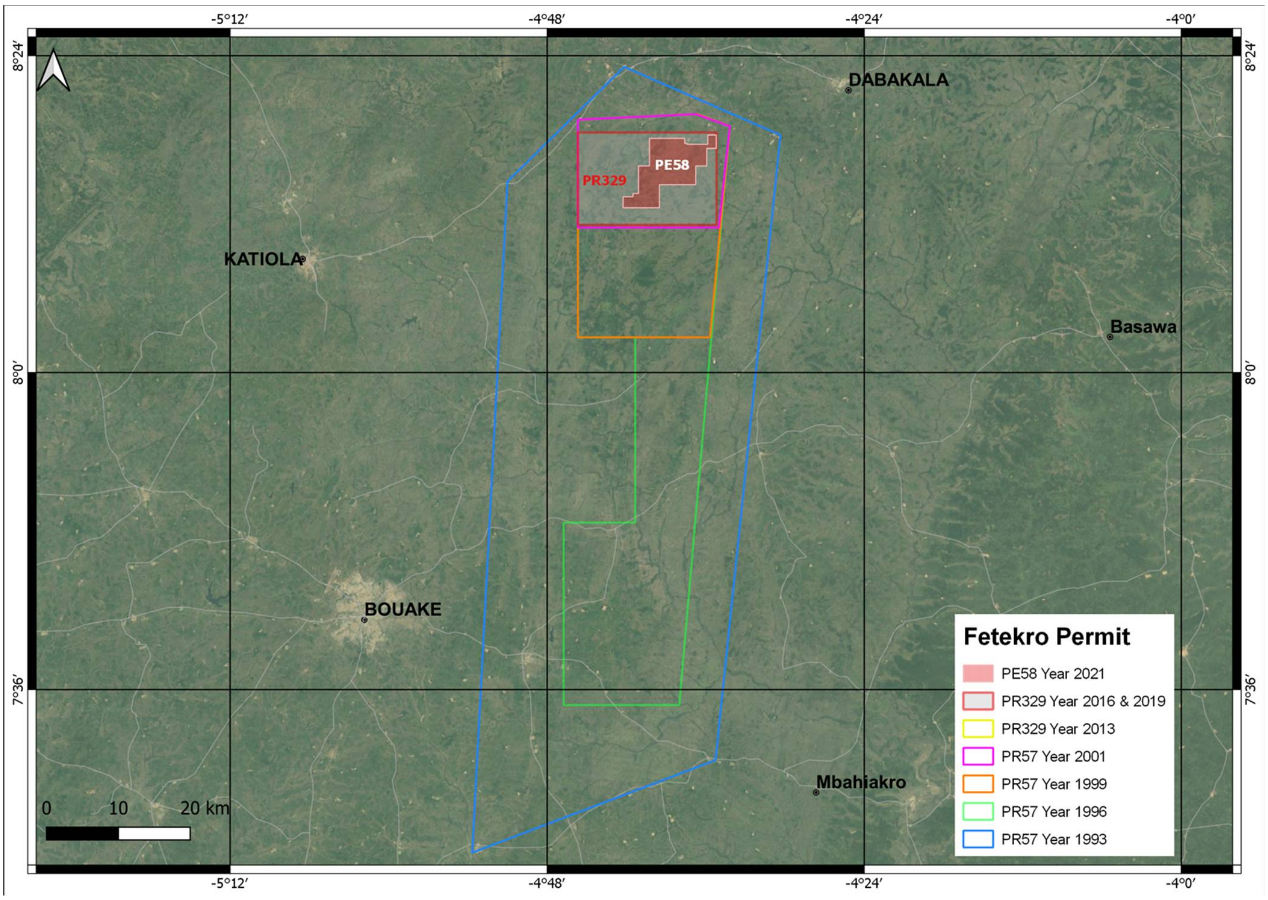

Figure 4-2: Chronological Permit Changes (PR 57, PR 329 and PE 58), Endeavour, 2022

4.3.2 Property and Cadastral Information

Property and cadastral information for PR 329 and PE 58, is summarised in Sections 4.3.2.1 and 4.3.2.2, following.

4.3.2.1 PR 329

Endeavour's Exploration Permit (PR 329), hereafter referred to as the '**Fetekro Exploration Licence**' or '**Fetekro EL**' is a rectangular block approximately 19.3 km (east-west) by 12.9 km (north-south), with a historical surficial area of 249.8 km². The Fetekro EL expires on 6 June 2022, and as of the 'Effective Date' of this Report, the third renewal is pending.

The renewal application, as submitted (Endeavour, 2022b), subtracts PE 58 from the historical licence area and aligns the eastern boundary of PR 329 with the eastern boundary of PE 58. This realignment of the eastern boundary reduces the surficial area of PR 329 by 1.8 km^2, which, when combined with the removal of the surface area associated with PE 58, reduces PR 329 to a new surficial area of 183.9 km^2.

The historical Fetekro EL boundary coordinates are set out in Table 4-1, following.

Table 4-1: PR 329 Boundary Coordinates (Ministere de Mines et de la Geologie, 2020)

Point	Latitude North	Longitude West
A	08°18'12"	04°45'40"
B	08°18'12"	04°35'12"
C	08°11'12"	04°35'12"
D	08°11'12"	04°45'40"

4.3.2.2 PE 58

As illustrated in Figure 4-1, the Exploitation Permit (PE 58), hereafter referred to as the '**Lafigué Mining Licence**' or '**Lafigué ML**', is elongated in a north-westerly/south-easterly orientation though Exploration Permit (PR 329). The permit covers an area of 64.08 km^2 (Ministre des Mines du Petrole Et de L'Energie, 2022b) and is approximately 13 km (east-west) by 10 km (north-south). The boundary coordinates for PE 58 are illustrated in Table 4-2.

Hereafter, the Lafigué ML may also be referred to as the 'Project Site' or 'Site', given that the mine and attendant infrastructure is all located within PE 58.

Table 4-2: PE 58 Boundary Coordinates (Ministre des Mines du Petrole Et de L'Energie, 2022a)

Point	Latitude North	Longitude West
1	08° 17' 45"	04° 40' 15"
2	08° 17' 45"	04° 37' 35"
3	08° 17' 22"	04° 37' 35"
4	08° 17' 22"	04° 37' 24"
5	08° 17' 18"	04° 37' 24"
6	08° 17' 18"	04° 35' 50"
7	08° 18' 00"	04° 35' 50"
8	08° 18' 00"	04° 35' 12"
9	08° 17' 00"	04° 35' 12"
10	08° 17' 00"	04° 35' 55"
11	08° 15' 40"	04° 35' 55"
12	08° 15' 40"	04° 36' 45"
13	08° 14' 15"	04° 36' 45"
14	08° 14' 15"	04° 39' 30"
15	08° 12' 30"	04° 39' 30"
16	08° 12' 30"	04° 42' 15"
17	08° 13' 20"	04° 42' 15"

Table 4-2: PE 58 Boundary Coordinates (Ministre des Mines du Petrole Et de L'Energie, 2022a)

Point	Latitude North	Longitude West
18	08° 13' 20"	04° 41' 30"
19	08° 13' 35"	04° 41' 30"
20	08° 13' 35"	04° 41' 05"
21	08° 15' 40"	04° 41' 05"
22	08° 15' 40"	04° 40' 15"

4.3.3 Historical and Current Property Ownership

PR N° 57 or 'PR 57', valid for gold and all substances except hydrocarbons was initially awarded to SODEMI according to the decree n° 93-215 of 03 February 1993 for a period of three years, with an initial allocated surficial area of 2600 km².

In 1996, an exploration, development and operating agreement was entered into by SODEMI, the Permit title holder and GENCOR Limited (GENCOR), through its Ivoirian company GATRO-Côte d'Ivoire (GATRO-CI) for PR 57 (the Exploration Agreement). According to this Exploration Agreement, the exploration campaigns were undertaken by GENCOR through its Ivoirian company (GATRO-CI), SODEMI, Bureau de Recherches Géologiques et Minières (BRGM)[29], and by the Australian mining group Normandy Mining, through its Ivorian subsidiary La Source.

In 1999, Compagnie Minière Or (COMINOR) took over La Source's participation and the GATRO-CI contractual commitments under the Exploration Agreement.

The civil war in Côte d'Ivoire from 2002 to 2010 did not allow the aforementioned SODEMI-COMINOR[30] partnership to continue exploration activity on PR N° 57, within the framework of an 'exceptional renewal'. Thus, PR 57 expired and a new permit PR 329 was awarded to SODEMI by decree N° 2013-410 of 6 June 2013 and covered by the Exploration Agreement.

In 2014, La Mancha Côte d'Ivoire SARL (LMCI) replaced COMINOR in the partnership with SODEMI, leading eventually to a transfer of PR 329 from SODEMI to LMCI in 2020, and the granting of PR 58 to LMCI, and then SML in 2021.

The chronological development and ownership of PR 329 and PE 58, from the date at which PR 329 was first issued is summarised in Table 4-3, following. Historical agreements pertaining to the development of PR 57, PR 329 and PE 58 are summarised more fully in Section 4.5.

[29] French state institution for the management of surface and subsurface resources and risks **Invalid source specified.**.

[30] 0n 31 July 2000, BRGM and Compagnie Générale des Matières Nucléaire (COGEMA)[30], signed an agreement by which COGEMA, through its subsidiary CFMM (Compagnie Française de Mines et Métaux), bought COMINOR (Compagnie Minière Or), a subsidiary of BRGM **Invalid source specified.**.

Table 4-3: Permits, Agreements and Ownership (Endeavour, 2022)

Description	Value	Comments
Exploration Permit:	**Fétékro Exploration Licence**	**'Fétékro EL' (Public/Internal name)**
• Number/Name:	PR 329	Official name on Permit
• Area:	249.8 km^2	Will reduce in size with the 3rd renewal (183.9 km^2)
• Date granted:	06/06/2013 (expired 06/06/2016), Arrêté No. 2013-410 of 6 June 2013.	1995 Mining Code (355 km^2).
• 1st renewal date:	6/6/2016 (expired 6/6/2019), Arrêté No. 090/MIM/DGMG of 11 July 2017.	2014 Mining Code (249.8 km^2).
• 2nd renewal date:	6/6/2019 (expired 06/06/2022) Arrêté No. 00008/MMG/DGMG of 13 January 2020.	2014 Mining Code (249.8 km^2).
• 3rd renewal date:	06/06/2022 (**when granted**, will expire 06/06/2024)	2014 Mining Code (183.9 km^2), exceptional renewal request received by government on 03/03/2022 (Letter reference No. VPE/SB/PK/129/03-2022)
• Applicable mining codes:	2014 Mining Code	
• Historical permit holder	Société pour le Développement Minier de la Cote d'Ivoire.	SODEMI
• Permit transfer date:	Arrête No. 00174/MMG/DGMD of 18 December 2020.	Transfer from SODEMI to LMCI
• Permit holder:	La Mancha Côte d'Ivoire SARL (100%)	'LMCI'
• Shareholder of LMCI:	Ity Holdings Ltd. (100%)	'ITYH'
• Ultimate shareholder of ITYH	Endeavour Gold Corporation (100%);	'EGC'
	Endeavour Mining Corporation (100%); and	'EMC'
	ultimately Endeavour Mining plc (100%)	'Endeavour'
Exploitation Permit:	**Lafigué Mining Licence**	**'Lafigué ML'(Public/Internal name)**
• Number/Name:	PE 58	'PE 58 (official name on Permit), valid for 'gold'
• Area:	64.08 km^2	
• Date granted:	22/09/2021. Decree No. 2021-538 of 22 September 2021	(12-year validity, based on a two-year construction period and a 10-year LoM).
• Expiry date:	21/09/2033	
• Applicable Mining Code:	2014 Mining Code	
• Mining Convention:	TBC	Still to be negotiated as of the 'Effective Date'. Valid for 12 years, but renewal.
• Historical permit holder:	La Mancha Côte d'Ivoire SARL	'LMCI'
• Permit transfer date:	12/01/2022	From LMCI to SML
• Current permit holder:	Société des Mines de Lafigué SA	'SML'
• Shareholders of SML:	Lafigué Holdings Ltd (80%);	'LAFH'
	Société pour le Développement Minier de la Côte d'Ivoire SARL (10%); and,	'SODEMI
	Government of Côte d'Ivoire (10%)	'GoCI'
Ultimate shareholders of LAFH	Endeavour Gold Corporation (100%); and	'EGC'
	ultimately Endeavour Mining plc (100%)	'Endeavour'

4.3.4 Back-in Rights

Property back-in rights for PR 329 and PE 58 are described in Section 4.5.

4.4 Surface Rights

As described in Section 4.2.5, SML has the requisite surface rights to develop a mine and the attendant infrastructure required on PE 58, as well as the rights to develop access to said Property. However, this is also contingent on having the requisite permits in place (Section 4.7).

Further, in July 2022, Endeavour approached the Director General of Mines and Geology for the right to blast/abstract non-mineral bearing materials for use in construction. Authorisation was subsequently received, with taxes payable on the material abstracted/used (Ouattara, 2022a).

4.5 Agreements and Encumbrances

4.5.1 Agreements

In 1996, an exploration, development and operating agreement was entered into by SODEMI, the title holder, and GENCOR (through its Ivoirian company GATRO-CI) in relation to two exploration permits, PR n°56 (Pranoi area) and PR n°57 (Fetekro area), the second one covering the current perimeter of the Lafigué Project (the Exploration Agreement). According to this Exploration Agreement, the exploration campaigns were done by GENCOR through its Ivoirian company (GATRO-CI) and SODEMI/BRGM/La Source.

In 1999, the Compagnie Minière Or (COMINOR), BRGM subsidiary, took over La Source participation and the GATRO-CI contractual commitments under the Exploration Agreement. At this occasion, the 1999 amendment also stated the expiry of PR n°56 (and, consequently, the removal of this agreement from the Exploration Agreement) and the renewal of PR n°57.

In 2000, COMINOR was transferred to Compagnie Générale des Matières Atomiques (COGEMA), which was subsumed into the La Mancha Group in 2006, via a reverse takeover of La Mancha by Compagnie Française de Mines et Métaux (CFMM), a fully owned subsidiary of the AREVA Group.

In 2013, AREVA sold its gold assets in Côte d'Ivoire to a private fund.

In 2014, PR n°57 expired and was replaced in the Exploration Agreement, by a new exploration permit n°329 covering a similar perimeter, which was attributed to SODEMI.

Moreover, in the same year La Mancha Côte d'Ivoire SARL (LMCI) was incorporated in Côte d'Ivoire, as a 100% subsidiary of COMINOR and subsequently took over the exploration activities of COMINOR, managed to date by its Ivoirian branch COMINOR CI. The takeover included all COMINOR contractual commitments under the Exploration Agreement, according to an amendment executed on 14 November 2014.

LMCI became a fully owned subsidiary of Endeavour Mining Corporation in November 2015. Since then, LMCI has been held 100% by Ity Holdings Ltd., a fully owned subsidiary of Endeavour Mining plc.

On 26 November 2020, a sale of the exploration permit agreement was entered into between LMCI and SODEMI. This sale agreement states that the Exploration Agreement will terminate on the date of the Ministerial Order transferring PR 329 to LMCI. SODEMI is entitled, inter alia, to:

- a payment of XOF 10 520 100 000;
- 10% of SML, the operating entity which will operate the Lafigué Mine; and

USD 3/oz for every additional ounce of 'Reserves' identified, over and above the Reserves associated with the 2.471 Moz of Measured and Indicated (M&I) gold Resource initially defined on PR 58; and any future mining license issued within the perimeter of PR 329.PR 329 (249.8 km²) was transferred to LMCI by Ministerial Order n°00174/MMG/DGMG, dated 18 December 2020. Based on the exploration works conducted and studies completed, LMCI submitted a request:

- for an environmental permit, granted by Ministerial Order n°00044/MINEDD/ANDE dated 18 February 2021; and
- for a mining license, granted by Decree n°2021-538 dated 22 September 2021 to LMCI under PR n°58, for a total surface of 64.08 km² and a duration of 12 years (including two years of construction).

Finally, PR 58 was transferred from LMCI to Société des Mines de Lafigué SA by Ministerial Order n°018/MMPE/DGMG, dated 12 January 2022.

4.5.2 Encumbrances

All mining licenses carry a 10% free carried interest in favour of the Government of Côte d'Ivoire (CI) GoCI and, as a result, the GoCI holds a 10% interest in SML.

Also, according to the sale agreement of PR 329 entered between LMCI and SODEMI in November 2020, SODEMI holds a dilutable 10% interest in SML; and SODEMI is also entitled to a complementary price of USD 3/oz for every additional ounce of 'Reserves' identified, over and above the Reserves associated with the 2.471 Moz of Measured and Indicated (M&I) gold Resource initially delineated on PR 58; and any future mining license issued within the perimeter of PR 329.Payments.

4.5.3 Free Carry Interest (FCI)

Under the NMC, Mining Permits are subject to a 10% free carry ownership interest to the benefit of the Government of CI (the 'State' or GoCI). The NMC limits the additional participation of the State in these companies to a contributory participation that cannot exceed 15% of the share capital. SODEMI a state mine development company and JV partner for the development of PE 58, does not form part of the State's participatory interest (Endeavour, 2022).

As stated, the additional payment terms noted in Section 4.5.2 are also applicable.

4.5.4 Royalties

In accordance with 'Ordonnance n° 2014-148 du 26 mars 2014 fixant les redevances superficiaires et les taxes proportionnelles relatives aux activités régies par le Code minier' (Article 5), once an exploitation permit is awarded, a 'Mining Convention' is signed, and the mine is in production, an 'Ad Valorem' (or proportional) tax is applied to gross sales revenue, after deductions for transport (FOB) and refining and/or smelting costs and penalties (Endeavour, 2022).

The tax rate for gold is as defined below:

- 3.0% if gold price is ≤ USD 1000/oz;
- 3.5% if gold price is >USD 1000 and ≤USD 1300/ozt;
- 4.0% if gold price is > USD 1300 and ≤ USD 1600/ozt;
- 5.0% if gold price is > USD 1600 and ≤ USD 2000)/ozt;
- 6.0% if gold price is USD (>2000)/ozt.

The tax rate for other metals is as defined below:

- 4.0% on silver; and
- 3.5% on copper.

Royalties are paid to the Ministry of Mines on a quarterly basis.

4.5.5 Surficial Fees

In accordance with Ordonnance n° 2014-148 du 26 mars 2014 fixant les redevances superficiaires et les taxes proportionnelles relatives aux activités régies par le Code minier (Article 2), annual payments on a surficial area basis, are payable for exploration and exploitation permits. The fixed and annual fees payable by permit type are as noted below (Endeavour, 2022).

Fixed fees based on granting and renewals, are as noted below:

- Exploration Permits: Granting: XOF 1 M; and Renewal: XOF 2 M.
- Exploitation Permits: Granting: XOF 0 M; First Renewal: XOF 1 M; and Second Renewal: XOF 2 M.

An annual payment based on a unit rate per km²:

- Exploration Permits: Granting: XOF 3000/km²; 1st renewal period: XOF 4000/km²; 2nd renewal period: XOF 6000/km²; and Exceptional renewal period: XOF 15 000/km².
- Exploitation Permits: XOF 250 000/km² (granting and renewal).

4.5.6 Central and Commercial Bank Payments

Central and commercial bank payments, totalling approximately 1.6% of the FOREX[31] transaction are payable (Endeavour, 2022).

4.5.7 Community Levies

In accordance with Ordonnance n° 2014-148 du 26 mars 2014 fixant les redevances superficiaires et les taxes proportionnelles relatives aux activités régies par le Code minier (Article 7), an Ad Valorem contribution of 0.5% of gross sales revenue after deductions for transport (FOB) and refining and/or smelting costs is applicable.

[31] Estimate/approximation and excludes transactions involving the EURO.

4.5.8 Bonds

As per Article 144 of the NMC, a rehabilitation bond is payable by the holder of the exploitation permit. The annual amount payable is based on the total rehabilitation costs stated in the official studies submitted to the Administration. The basis of payment is defined more fully in Section 4.2.6.

4.5.9 Taxes

4.5.9.1 Overview

The basis for the application of taxes during construction and production are summarised below. In CI, taxes payable are subject to the definitions outlined in the NMC for 'Production', namely:

The 'First commercial production date', is the date at which the mine reaches a continued period of production of sixty days at 80% of its production capacity as drawn up in the 'feasibility study' forwarded to the mining administration or the date of the shipment of the mining production for commercial purposes'.

Importantly, taxes payable by SML if different to the official tax basis outlined in Sections 4.5.9.2 to 4.5.9.13 following, will be as a result of any amendments to the tax basis in the Mining Convention (not signed as of the 'Effective Date' of this Report). Any amendments to the tax terms, will be to the benefit of SML.

If the tax basis in the Financial Model is different to that presented in the Financial Model, this will be stated in Section 21 and 25 of this Report and Section 14 of the DFS report.

4.5.9.2 Construction

During construction, the permit holder is exempt from import duties, except for the Regional/ECOWAS levy of 2.5% CIF (Port). Said exemption excludes duties on chemical products and fuel (Endeavour, 2022).

Unless otherwise agreed in the mining convention, sub-contractors will pay the Regional/ECOWAS Levy and be subject to full duties, typically 0 to 20%[32] of the CIF value (Endeavour, 2022).

4.5.9.3 Production

Unless otherwise agreed in the 'Mining Convention', the permit holder will in addition to the 'Regional/Ecowas' levy, be subject to full import duties as defined in the tax code for equipment and consumables, typically (0 to 20)%[32] of the CIF value (Endeavour, 2022).

Chemical products (including fuel) are exempt of duties and only subject to the Region/ECOWAS Levy of 2.5% (Endeavour, 2022).[33]

There are no duty exemptions for sub-contractors, unless otherwise agreed in the mining convention (Endeavour, 2022).

4.5.9.4 Carbon Taxes

There are no carbon taxes applicable in CI (Endeavour, 2022).

[32] Can be as high as 35%.

[33] Production equipment only. Non-production equipment (i.e., light vehicles, buses etc, are not exempt).

4.5.9.5 Withholding Taxes (WHT)

Subject to the jurisdiction of the service provider, withholding taxes are applied at a rate of 0 to 20% (Endeavour, 2022).

4.5.9.6 VAT

Only the Permit holder is VAT exempt for Construction. For Production, the rate will be 18% unless negotiated otherwise in the Mining Convention. The exception being chemical products[34] which are VAT exempt during production (Endeavour, 2022).

4.5.9.7 Tax on Insurance Premiums

Subject to the type of Product procured, tax on insurance premiums varies between 0.1 and 25% (Endeavour, 2022).

4.5.9.8 Dividend Payments

The policy for the payment of dividends will be as defined in the 'Mining Convention'. In general, a sliding scale is applied to cover the first year of commercial production, the period of repayment of the debt, and the final period after the debt has been repaid (Blot, J, 2022b).

4.5.9.9 Inspection Fees

Inspection fees will be payable for the explosives store (100 000 XOF/quarter) and to CIAPOL for environmental monitoring (sliding scale based on surficial area) (Blot, J, 2022b).

4.5.9.10 Employer Labour Taxes

The employer is subject to; a payroll tax for expatriates and nationals, and employer; retirement, family, and worker contribution/compensation payments. Said taxes are built into each employees total cost to company (TCTC) (Endeavour, 2022).

4.5.9.11 Business Tax (Patente)

Exemption during first 'three years' after production, then 15% payable on the calculated annual rental value of plant and buildings (Rental value is a determined as a function of the gross capital value of fixed assets over a defined term) (Endeavour, 2022).

4.5.9.12 CI Training and Capacity Building

As per Article 135 of the 2014 Mining Code, an annual; payment of XOF 25 M is payable to assist in building in-country institutional capacity (Endeavour, 2022).

4.5.9.13 Corporate Income Tax

As per the NMC, corporate income tax is set at 25% (Endeavour, 2022).

[34] Excludes fuels used in buses and light vehicles.

4.6 Environmental and Other Liabilities

LMCI retained the services of Enval[35], to undertake baseline studies and environmental research for the Project. These studies began in November 2019 and resulted in the publication of the Environmental and Social Impact Assessment (ESIA) at the end of September 2020.

The ESIA was submitted to a government committee for validation on 20 January 2021, and the study received a favourable opinion. The Environmental Permit (Arrêté number 00044) was obtained on 18 February 2021 and formed part of the submission requirements for the Exploitation Permit application.

On PE 58 there has been significant artisanal mining post the base line assessment, which formed the basis of the ESIA submission to government. This artisanal mining activity continued over different areas of PE 58 (Section 20 of the Report), both during and post the ESIA process, with consequential damage to; flora, fauna, soils and water/water courses.

The establishment of a fenced off area for the Lafigué Mine in 2022, stopped artisanal mining within the perimeter of the fence line. The artisanal miner removal process, and any potential related issues are discussed in Section 20 of this Report and not here.

To exclude the consequential damages arising from artisanal mining from SML's long-term closure liabilities, the Cote d'Ivoire Agency against pollution (CIAPOL) was invited to PE 58 in November of 2021. The results of the work undertaken to define the environmental impact of artisanal mining with CIAPOL and other stakeholders, is noted in Table 4-4.

Table 4-4: Management of Artisanal Mining Environmental Liabilities

Date	Reference	Issued by/to	Purpose:	Outcome
January 2021 to April 2022	(SML, 2022d)	SML	Drone surveys to capture extent of artisanal mining activity, and damage	Aerial survey imagery for the Area of Interest (AoI), over the following dates: 27/01/2021, 06/05/2021, 29/05/2021, 05/06/2021, 17/08/2021, 04/03/2022, 04/04/2022.
9 November 2021	(Biotitiale, 2021), (CIAPOL, 2022)[36]	CIAPOL/SML	CIAPOL Site Visit 09/11/2022 and Environmental Review of ASM sites and specifically, sampling of water from illegal miners ponds.	The results showed a high[37] Level of Cyanide (WAD, Free and Total). CIAPOL recommended that SML conduct crop and soil analysis
March 2022 to February 2023[38]		SML/CIAPOL	Environmental Baseline Monitoring. Analysis by: BIOTITIALE (water) and SGS (noise and air) in local villages (monthly data)	Data used for monthly reports indicated below.
July 2022	(SML, 2022c), (CVGCS-CI, 2022)	SML/CIAPOL	Address CIAPOL request to assess impact on soils and crops. Analysis by GVGCS-CI.	Most elements were below the analytical detection levels in crops, except; Zn (0.07) mg/kg, copper (0.11 to 0.8 mg)/kg and Total Cyanide (0.04 mg/kg).

[35] www.enval-group.com
[36] No further feedback from CIAPOL.
[37] Outside of limits set by Order No. 01164 /MINEF/CAB/SIIC of 4 November 2008.
[38] Contract duration with SGS and Biotitiale.

Table 4-4: Management of Artisanal Mining Environmental Liabilities

Date	Reference	Issued by/to	Purpose:	Outcome
				Soil samples (two of) indicated free cyanide levels of (0.11 to 0.8) mg/kg and total cyanide levels of (0.98 to 1.92 mg/kg total. The soil samples (around old cyanide ponds) were below the 'Trigger' values, which in Europe, are typically around 5 mg/kg (db) for Total Cyanide[39].
August[40] to September 2022	Ongoing	SML/CIAPOL, ANDE and Local Authorities)	Monthly environmental base line monitoring and reporting	Acknowledgment receipts from relevant stakeholders where applicable, after each monthly submission.
Ongoing	Not applicable	SML/CIAPOL	To keep CIAPOL Informed of new ASM sites	None

Table 4-4 notes:

- CIAPOL (Cote d'Ivoire Agency against pollution) - Regional Director.
- ANDE (National Agency for Environment) - General Director.
- Local Authorities: Environment - Regional Director; Dabakala Prefet (Prefect); Dabakala Sous-prefet (sub-prefect); and Bonieredougou Sous-prefet (sub-prefect).

SML has undertaken work with government authorities to document as a function of time, the extent of the damage done by artisanal miners; and thus, there should be sufficient information available to establish a baseline before mining commences, thereby mitigating any potential future liabilities. Additional recommendations for further environmental and social baseline work, are made in Section 20 of the Report.

With the exception of what has been stated in Section 4.6, the QP for Section 4 and 11 of this Report and the DFS respectively, is not aware of any other legal or financial liabilities that are relevant to the development of the Lafigué Mine on PE 58. Further the QP has followed up with all relevant stakeholders to confirm that this is the case, no issues were raised by the various parties (Thomson, 2022).

4.7 Legal and Permitting

4.7.1 Overview

SML has followed due process in having the requisite permits in place to start developing the Lafigué Project on PE 58, and to continue to do exploration work on PR 329. However, it is notable that:

- The renewal Permit for PR 329 is outstanding.
- Whilst key permits are in place to start developing the Project on PE 58, the signing of the 'Mining Convention' is significantly outside of the timelines[41] defined in the CI 2014 Mining Code. This goes to defining the tax/payment derogation basis for SML and its contractors.

[39] **Invalid source specified.**.

[40] First monthly report submission, covering the period from March to August 2022

[41] Should have been signed by 15 December 2021.

- The approved ESIA and MRCP was based on the pre-feasibility study results. Whilst the DFS is not significantly different, as per Article 6 of the Environmental authorisation No. 00044/MINEDD/ANDE, dated 18 February 2021, ANDE must be notified accordingly of scope changes to the original ESIA (outstanding as per the 'Effective Date' of this Report).
- The Mine Closure and Rehabilitation Bond Basis needs to be finalised, specifically the:
  - escrow account is to be opened within 20 days following first commercial production; and
  - bank guarantee to be put in place within 120 days from date of first commercial production.
- A Permitting register has not yet been fully developed for the construction, operational and closure phases of the Project's/Mine's life cycle. Until such time as this is completed and aligned to the construction and operations schedule, it is not possible to say with certainty that all relevant permits will be in place in time. Notwithstanding this, there is likely sufficient time to address, if acted upon expediently.

The status of the primary and secondary permits and compliance requirements to develop, operate and close the Lafigué Mine are summarised in Section 4.7.2 and 4.7.3. Importantly, this should not be seen as an exhaustive list of all the permitting requirements.

4.7.2 Primary Permits, Agreements and Compliance Requirements

The status of the primary permitting activities associated with PE 58 and PR 329 are summarised below; and in part, in Table 4-3.

- PR 329 Exploration Permit.
  - Second renewal expired 6 June 2022.
  - Exception renewal application submitted/received by the authorising authority on 3 March 2022. If and when renewed, will expire 6 June 2024.
- PE 58 Exploitation Permit.
  - Granted to LMCI on 22/09/2021. Decree No. 2021-538 of 22 September 2021, first expiry 21 September 2033 (Presidency of the Republic, 2021).
  - Permit ownership was transferred from LMCI to SML on 12 January 2022.
  - In the granting of PE 58, it is a requirement that the:
    - ESIA be approved (approved 18 February 2021);
    - ESMP and MCRP be approved (approved in the granting of PE 58); and
    - Minister of Mines, Petroleum and Energy be notified of any change in the mine plan/production schedule (not done as per the 'Effective Date' of this Report).
- PE 58 Mining Agreement/Convention - Defines the tax derogation basis for SML and its contractors.
  - To be in place within (60 working days) of the exploitation permit being signed (Chapter II, Article 12 of the 2014 mining code).
  - The mining convention between SML and the State should have been signed by 15 December 2021.
  - As of the 'Effective Date' of this Report, the 'Mining Convention' is not signed.

- Commencement of work notification sent by SML to Directeur Général de Mines et de la Géologie (DGMG)/Ministere Des Mines, Du Pétrole et De L'Energie (MMPE).
  - First issued by SML on 31 January 2022 stating that work has commenced on site access infrastructure, and that work on the main mine infrastructure is to commence July 2022, finishing June 2024; and the customs administration should be notified accordingly for the exoneration of duties on imported goods and equipment (SML, 2022a). The MMPE duly notified (No. 0911/MMPE/DGMG/DDM) the Customs Directorate on 8 February 2022.
  - SML issued a second update to the DGMG and MMPE, requesting that the July 2022 start date be bought forward to 1 April 2022.
- As of the 'Effective Date' of this Report;
  - ANDE have not been notified of commencement to 'work activities', as per Article 10 of the environmental authorisation No. 00044/MINEDD/ANDE, dated 18 February 2021.
  - The Minister of Mines, Petroleum and Energy has not been notified of the change in LoM/Production schedule, as per Article 9 of Decree n° 2021-538 of September 2021.

4.7.3 Secondary Permits and Compliance Requirements

The full list of the secondary permits required to operate the Lafigué Mine is currently being developed/verified and are not ready to publish in this Report. Notwithstanding this, some of the key permitting/compliance requirements are as noted in Table 4-5.

Other permits that will be required but are not specifically listed in Table 4-5, include:

- General Waste Landfill, managed/authorised by ANAGED (National Agency for Waste Management), CIAPOL and ANDE.
- Radiation declaration: for importation and storage on site radiation equipment. Specific training required for identified people by the Radiation Safety, and Nuclear Security Authority (ARSN). Radiation Permit and Training to be done every two-years.
- Autorisation environnemtale d'exploiter by CIAPOL. This permit will indicate the amount to be paid to CIAPOL (Environmental Tax) - paid biannually.
- Plan d'Operation Interne (POI); Interministerial Order No. 4862 of July 14, 1999, making the establishment of an Internal Operation Plan (POI) compulsory in certain classified establishments. This plan is to be prepared by an external consultant and authorised by the relevant authorities.
- Site Clinic (construction and operations) governed/authorised by the Ministry of Health; and
  - Decree No. 96-877 of 25 October 1996 on the classification, finishing and organization of private health facilities.
  - Decree No. 96-878 of 25 October 1996 laying down the conditions of authorization and registration for the installation of health professions in the private sector.
  - Ministerial Order No. 255/MSHPDGS/DEPS of 04 April 2019 registering private health facilities.
- Permitting of the airstrip by Autorité Nationale de l'Aviation Civile (ANAC).

Table 4-5: Permitting and Compliance Requirements

Thematic area	Legal Texts	Articles or Chapters Concerned by the Project	Digby Wells/ENVAL Comment
Hazardous materials handling	Law No. 92-469 of July 30, 1992, on the repression of fraud in the field of petroleum products and violations of technical safety requirements.	Article 2: The import, export, transformation, storage, transport and distribution of petroleum products are subject to prior authorization, under conditions defined by decree.	SML must get prior authorisation for the import, export, transformation, storage, transport and distribution of petroleum products where applicable.
Natural Resources	Law No. 96-766 of October 3, 1996, on the Environmental Code.	Article 39: Any project likely to have an impact on the environment must be the subject of a prior impact study.	SML has/will assess the impacts of mining and associated activities on the receiving environment and manage these impacts in accordance with the ESMP
Water use	Law No. 98-755 of December 23, 1998, on the water code.	Article 12: Withdrawals from waters in the public hydraulic domain and the construction of hydraulic facilities or structures are subject, depending on the case, to prior authorization or declaration.	Any abstraction of surface- or groundwaters required for the Project must be authorised. Permitting of hydraulic structures to be investigated.
Permitting procedure	Decree No. 98-43 of January 28, 1998, relating to Installations classified for the protection of the environment.	Article 1: Are subject to the provisions of this decree, factories, depots, construction sites, quarries, underground storage, shops, workshops and in general, installations operated or owned by any natural or legal person, public or private, who may present disadvantages for the convenience of the neighbourhood, for health, safety, public sanitation, for agriculture, for the protection of nature and the environment and for the conservation of sites and monuments.	The nature of the planned activities requires that SML obtains authorizations from the Minister of the Environment and Sustainable Development and undertakes to implement measures aimed at controlling environmental risks.
Permitting procedure	Decree No. 98-43 of January 28, 1998, relating to Installations classified for the protection of the environment.	Article 3: Installations presenting the dangers and disadvantages referred to in Article 1 are subject to prior environmental compliance authorization from the Minister for the Environment.· The authorization can only be granted if these dangers or inconveniences can be prevented by the execution of the measures specified by order of the Minister in charge of the Environment.	
Compliance	Decree No. 98-43 of January 28, 1998, relating to Installations classified for the protection of the environment.	Article 32: The facilities referred to in Article 1 of this decree are subject to a half-yearly control and inspection fee, the basis and rates of which are set by Finance Law No. 73-573 of December 22, 1973.	Provide fees for carrying out checks and inspections.
Health and safety	Decree No. 2014-397 of June 25, 2014, determining the terms of application of the law No. 2014- 138 of March 24, 2014, on the mining code.	Article 146: The import, export, transport, sale, transfer, use and storage of explosive substances require the prior authorization of the Mines Administration. The conditions for importing, exporting, transporting, selling, transferring, using, destroying, storing and any other movement of explosive substances are defined by joint order of the Minister in charge of Mines, the Minister in charge of Budget and the Minister responsible for Trade.	Permit/authorisation is required for the transport and storage of explosives.

Table 4-5: Permitting and Compliance Requirements

Thematic area	Legal Texts	Articles or Chapters Concerned by the Project	Digby Wells/ENVAL Comment
Compliance	Decree No. 2015-346 of May 13, 2015, determining the list of breaches of the water code that may give rise to a transaction and breach.	Articles 2 and 3: The offenses are among others: • the abstraction of water from the public domain in excessive quantities, without prior authorization or declaration; • the wastage of water; • the discharge, discharge or runoff into surface waters, groundwater or waters of the territorial sea, of waste or substances whose effects are harmful to health or cause damage to flora or fauna, or alter the normal flow of water; • the degradation of the quality of water or hydraulic facilities or structures; and • the supply to the public of water, which does not comply with hygiene and public health standards, for human or animal consumption, free of charge or against payment.	Appropriate water management infrastructure must be in place, accommodating flood events. Where water is abstracted from public domain, flow meters must be installed and monitored in accordance with a monitoring procedure. The water harvest dams alter the normal flow of water, and thus any permitting requirements need to be determined.
Hazardous materials handling	Order No. 13 SEM. CAB. DH. of February 27, 1974, regulating the development or extension of oil depots and establishments.	Article 1: The creation, development or extension of a depot or an oil establishment are subject to the prior authorization of the Secretary of State for Mines.	SML must have the required authorizations for the storage of hydrocarbons on its site.
Hazardous materials handling	Order No. 13 SEM. CAB. DH. of February 27, 1974, regulating the development or extension of oil depots and establishments.	Article 5: The oil establishment or depot, must comply with the technical and safety regulations in force.	
Hazardous materials handling	Order No. 13 SEM. CAB. DH. of February 27, 1974, regulating the development or extension of oil depots and establishments.	Article 6: The commissioning of a depot or an oil establishment is subject to obtaining an operating permit issued after verification of the compliance of the depot or establishment with the plan and the provisions specified in the request as well as the technical regulations in force.	
Waste management	Order No. 0012/MINEDD/DGE/PFCB of 15 March 2012 on the authorization border movement and transfer of waste.	Article 4: When waste is intended to be treated outside Côte d'Ivoire, the producer of this waste must obtain a cross-border transfer authorization for waste, before any waste leaves the national territory.	In the event that some waste streams such as hazardous waste must be treated or disposed of at an appropriate facility outside of CI, the required permits must be obtained.

4.8 Data Verification

The approach taken to verifying the data used and presented in Section 4, is discussed in Section 12 of this Report.

4.9 Independent Reviews/Audits

As part of the definitive feasibility study work programme, no independent Audits/Reviews have been undertaken on the properties in question (PR 329 and PE 58), from a permitting/ownership/legal/compliance perspective.

4.10 Comments on Section 4

The QP considers that for PE 58, the level of reporting herein is largely in accordance with requirements of a definitive feasibility study, and meets the requirements as set out in the 'NI 43-101 Standards of Disclosure for Mineral Projects', Form 43-101F1 Technical Report (June 24, 2011), specifically Item 4, points (a) to (h). Further, the information presented herein has been used as presented, in the relevant sections of this Report.

There are concerns that a permitting/agreement/notification register has not been developed in detail for construction and operations, albeit this is underway and there is likely still time to address any issues.

4.11 Interpretations and Conclusions

Interpretations, conclusions and risks for Section 12, are presented in Section 25 of this Report.

4.12 Recommendations

Recommendations for Section 4 are presented in Section 26 of this Report.

4.13 References

References cited in the preparation of Section 4 are presented in Section 27 of this Report.

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Introduction

This section outlines the geophysical, political and economic environment that the proposed Lafigue Project operates in; the enabling infrastructure and services; and the rights of the Issuer to develop mine infrastructure on the Lafigué Mining License (PE 58), in the north-central region of Côte d'Ivoire (CI).

5.2 Location

CI is located along the coast of West Africa and is bordered to the east by Ghana, to the north by Burkina Faso and Mali, and to the west by Guinea and Liberia. The location of PE 58 relative to neighbouring countries, key cities, enabling infrastructure and the Issuer's interests, is illustrated in Figure 5-1 following.

Figure 5-1: Côte d'Ivoire (Google Earth®, 2022)

The Issuer's gold interests in CI, namely Ity and Lafigue, are supported by the Issuer's head office support functions in London, United Kingdom and a regional West African office in Abidjan, CI.

The Lafigue Mining Licence (PE 58) lies within the:

- Issuer's Fetekro exploration permit, which is approximately 330 km north-northwest of the port city of Abidjan (approximately 470 km by road) the economic capital of CI and 175 km north-northeast of the political capital of CI, Yamoussoukro (approximately 230 km by road).
- Dabakala Department[42] in the Hambol Region[43], which is approximately 55 km east of the A3 (highway) and rail line that connects Abidjan with Burkina Faso.
- The Sous-Prefecture of Bonieredougou (Cabinet Enval, 2021).

The Hambol Region is one of 31 regions in CI, whose administrative capital is Bouaké, CI's second largest city, with a population of over 536 000 persons (2014) (Wikipedia, 2022a). Bouaké lies approximately 80 km southwest of PE 58 (approximately 130 km by road)

The 'Seat' of the Hambol region is Katiola, a city of over 40 000 persons (2014) (Wikipedia, 2022b), which like Bouaké lies on the A3, the main arterial route between Abidjan and Burkina Faso. Katiola lies 54 km west-southwest of PE 58 (approximately 70 km by road)

The position of PE 58 and the Fetekro Exploration Permit in relation to the main highways (A3 to Burkina Faso and A10 to Ghana), rail line, arterial roads (B412 and B417), and regional population centres and villages is illustrated in Figure 5-2.

As illustrated in Figure 5.2, there are no local villages on PE 58, however there are number of larger towns in the area and a number of small and medium sized villages to the north and south of the B412. In additional to Katiola, larger regional centres include Dabakala[44] (Seat of the Dabakala Department), Satama-Sakoura, Niemene, Timbe and Bonieredougou.

The Lafigue Project, takes its name from the Lafigue village which whilst lying on the Fetekro Exploration Permit, is just north of PE 58.

[42] Third-level administrative subdivision (109 departments)

[43] Second-level administrative subdivision (31 regions)

[44] Population 56 000**Invalid source specified.**, 45 km by road from PE 58

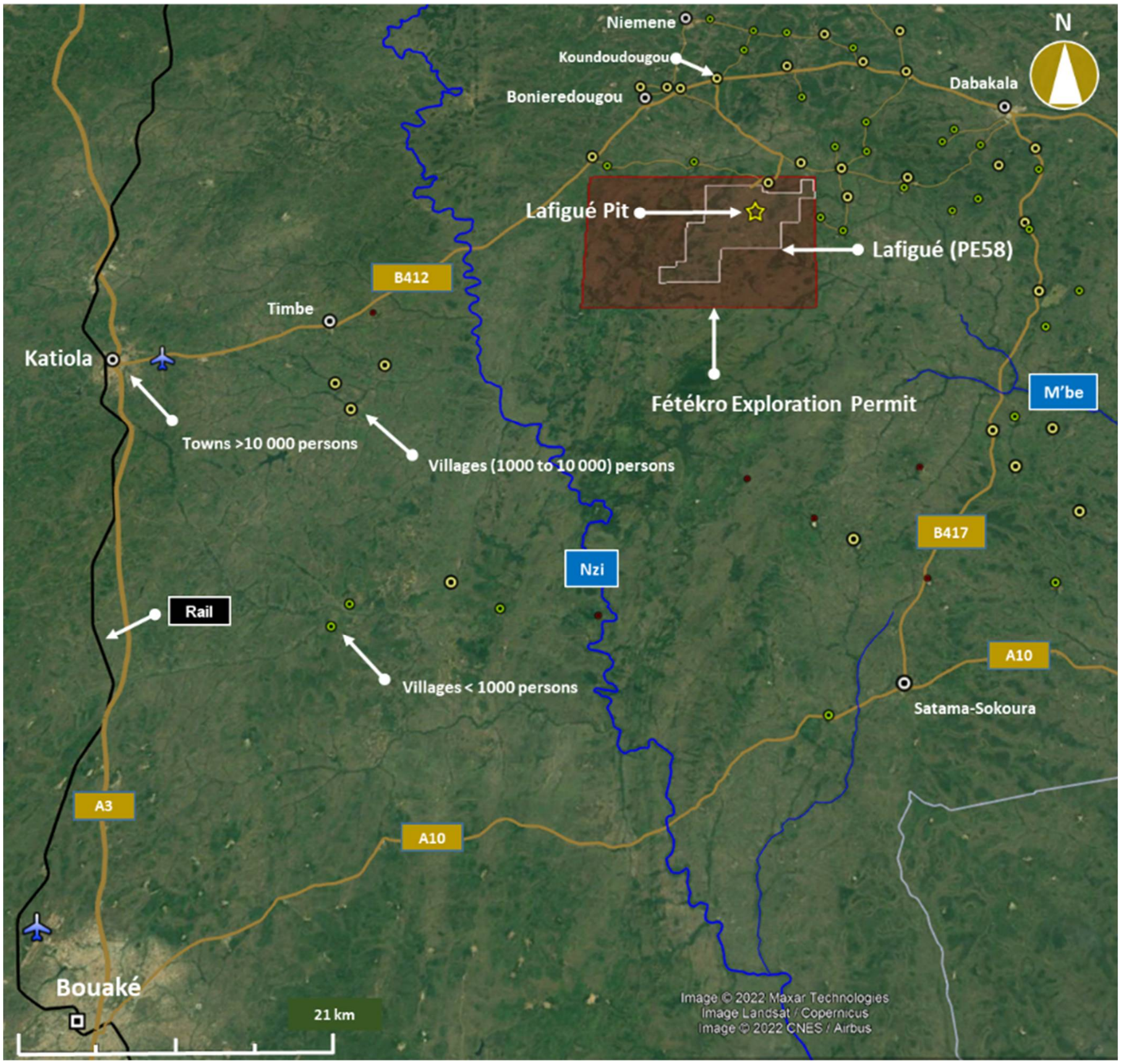

Figure 5-2: PE 58, Population Centres, and Logistics Infrastructure (Google Earth®, 2022)

5.3 CI Political and Economic Environment

CI went through a period of instability following a coup d'état in 1999, and two civils wars between 2002 and 2007, and 2010 and 2011. Since 2012 CI has gone through a period of relative political stability and high economic growth, largely driven by; a series of successful National Development Programmes (NDP or PND)[45;] robust export demand for agricultural products; and financing from intergovernmental organisations (IGOs) (World Bank/IMF, African Development Bank and others).

[45] PND 2012 to 2016**Invalid source specified.**, PND 2016 to 2020**Invalid source specified.**, PND 2021 to 2025

As a consequence of the PND's, outstanding public debt which was 37.9% of GDP at the end of 2019, is expected to grow to 41.7% of GDP in 2020 and stabilize at an average of 42.5% of GDP during 2021 and 2022, below the 70% threshold set by the West African Economic and Monetary Union (African Development Bank, 2021).

With the exception of the impact of the COVID crisis of 2020, since 2012 CI has maintained an annual GDP growth rate in excess of 5 %/a, with an annual GDP per capita in 2020 of USD 2300 (Real) and USD 5181 (PPP) (World Bank, 2020). CI's GDP in (2020) stood at USD 61.4 billion/a, ranking it 76th in the word. In June of 2022, Moody's affirmed Côte d'Ivoire's Ba3 rating, and changed its outlook to positive from stable. Further, Moody's believe real GDP growth rate is likely to be of the order of 7%/a from 2022 to 2025 (Moody's, 2022).

CI is also one of the 15 members of the Economic Community of West African States (ECOWAS). This union seeks to create a single economic block and develop infrastructure (road, rail, power and communications) and policies that are to the benefit of all stakeholders.

5.4 Country Population and Demographics

The most reliable population and demographics data for CI, is based on a census survey undertaken by the National Institute of Statistics (www.ins.ci) of CI in 2014 'Recensement Général de la Population et de l'Habitat 2014 (RGPH 2014). The most recent census was undertaken in 2021, however this data has not yet been released. Importantly, given the time intervals between census surveys and concerns around data accuracy, population and demographics data presented herein, should be seen as a guide only.

The population of CI is estimated to be 27.0 M persons, with a population growth rate of 2.5%/a and a fertility rate of 4.5 children per woman (2021) (World Bank, 2021e).

Additional key data is provided below:

- fifty-one per cent of the population lives in an urban setting (2020) (Worldometer, 2020);
- the median population age is 18.9 years, with a life expectancy of 58 years (Worldometer, 2020);
- country literacy rates for persons over 15 (1988 to 2014), is circa 40% (RPGH 2014); and,
- unemployment rates are low, circa 3.5% (World Bank, 2021e).

5.5 Local Topography, Elevation and Vegetation

5.5.1.1 Topography and Elevation

The topography in the area surrounding Dabakala is relatively flat with an average elevation of approximately 260 mamsl, with local variations of approximately 60 m (Cabinet Enval, 2021). There are low lying hills to the north of PE 58, where the elevation increases to just above 400 mamsl and on PE 58, the elevation varies from approximately 310 mamsl in the northeast corner of the license, to approximately 230 m in the southeast corner. In the area of the proposed pit, the elevation increases to approximately 400 mamsl. The majority of the infrastructure that supports mining is to be constructed at an elevation of approximately (300 to 330) mamsl (Google Earth®).

5.5.1.2 Soils

Ferralitic[46] soils, are generally located on the lower and high plateaus and have a structure in which the mineral alteration is complete due to the influence of paleo-climatic factors and very old types of vegetation. Mineral and organic hydromorphic soils are generally found in the vicinity of streams and marshy areas. Tertiary formations or neogenic sands consist of clay-sandy soils, sandy-clay soils (Cabinet Enval, 2021).

5.5.1.3 Vegetation

According to Guillaumet and Adjanohoun (1971), the centre-north of Côte d'Ivoire to which the study site belongs is the domain of clear forests and savannas (wooded, shrubby and grassy savannas) that derive from it (Cabinet Enval, 2021).

Gallery and riparian forests typically run along the permanent or temporary rivers and their tributaries and whose species are subservient to the forest islands (Cabinet Enval, 2021).

5.6 Local Climate

5.6.1 Introduction/Overview

CI is located in the transition zone between the humid equatorial climate that characterizes the southern part of the country, and the dry tropical climate of the north. The country generally experiences a rainy season from June to October and average annual temperatures range from (24 to 28)°C (World Bank, 2020a). The Köppen-Geiger Climate Classification for CI from 1991 to 2020 is As[47]/Aw[48] 'Tropical Savanna Climate' (World Bank, 2020b)

As illustrated in Figure 5-3 (left image), there are three distinct climatic zones in CI, the southern climatic zone, the central climatic zone and the northern climatic zone, which goes from the northwest of CI up into Burkina Faso. PE 58 falls within the central climatic zone.

The central climatic zone has a warm and humid climate, with a daily mean temperature of 27° C and mean annual total rainfall of 1440 mm/a (Figure 5-3 (image on right)) with relatively low to moderate interannual variability, but quite high spatial variability, with rainfall decreasing towards the northeast. Rainfall occurs in one very long rainy season from March to October, peaking at just over 200 mm/month during September.

There has been no discernible trend in annual rainfall over the past 35 years in the central zone, however all regions do show a statistically significant decrease in the frequency of rainfall days, but an increase in the frequency of extreme rainfall events. In the central zone there has been a drop in rainfall days (-4.7 d/decade) and an increase in extreme rainfall days (+2.4 d/decade). With respect to temperature and climate change, a (1 to 3.5) °C increase is expected through to 2050 (African Development Bank, 2019).

[46] High iron and aluminium, with low silica

[47] Group A, Tropical savanna climate with dry-summer characteristics

[48] Group A: Tropical savanna climate with dry-winter characteristics

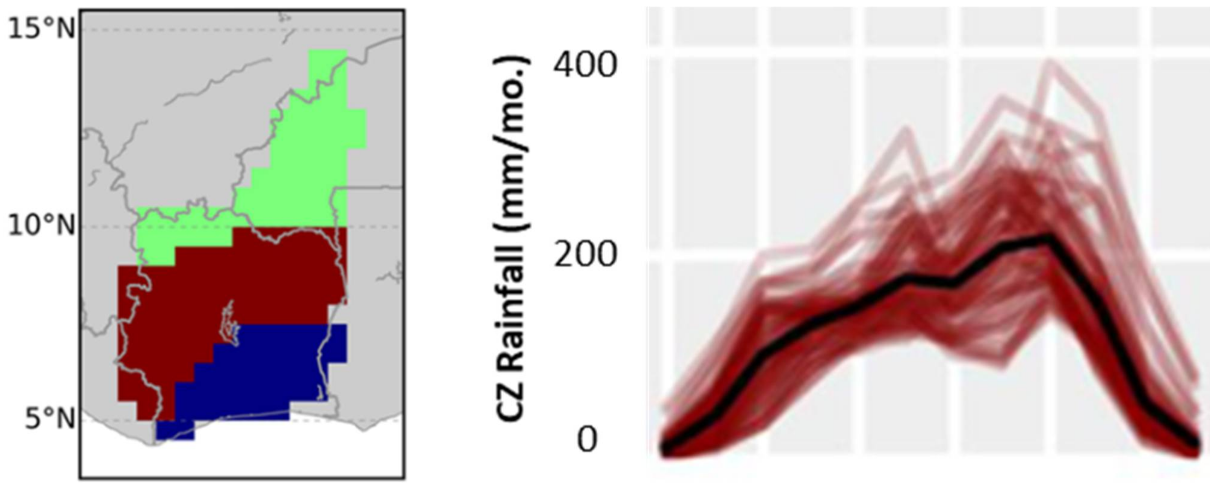

Figure 5-3: CI Rainfall Regions and Monthly Rainfall 1979 to 2013 (African Development Bank, 2019)

Figure 5-3 notes.

- The dark red zone in the left image represents CI's Central Zone (CZ), with the light green and blue zones representing CI's northern and southern zones respectively.
- The rainfall by month over the time period monitored for the CZ is presented in the image on the right.

5.6.2 Temperature

The temperatures are generally high all year round, with minimum and maximum mean monthly average dry bulb temperatures of (19 and 36) °C respectively. The mean average monthly dry bulb temperature varies between (21 (August) and 29.5 (March)) °C[49]. A range of average dry bulb temperature readings by month from 1991 to 2020 for the Vallée du Bandama District, is presented in Figure 5-4 following.

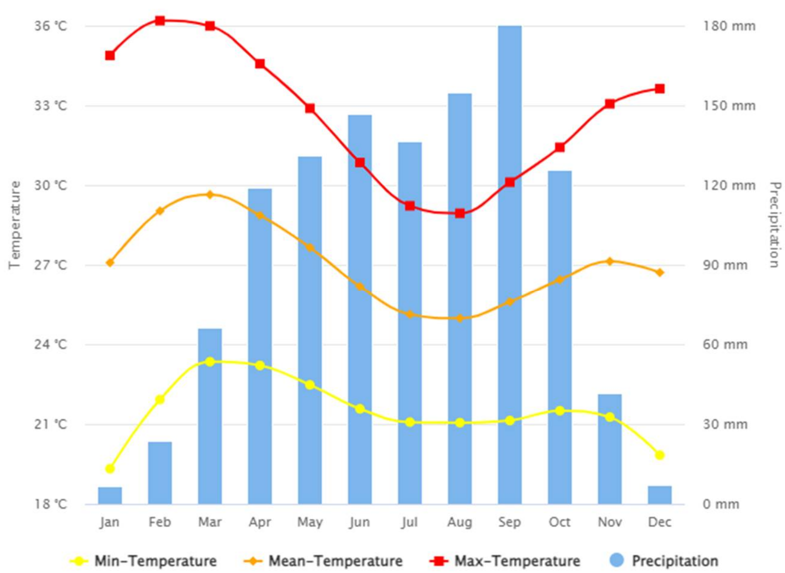

Figure 5-4: Vallée du Bandama Monthly Temperature and Rainfall Variation, (World Bank, 2020a)

[49] Minimum and maximum daily temperatures can be outside this range.

Whilst regional humidity has be reported by (Cabinet Enval, 2021)[50], the coincident dry and wet bulb, and dew point temperatures have not. However, in general, humidity is high from May to November (monthly average greater than 70% RH) (Weather Atlas, 2022).

5.6.3 Rainfall & Evaporation

The Lafigue Mine will be reliant on the collection of water (surface and underground) from seasonal rainfall (Section 5.8.1) and thus, the accurate prediction of seasonal rainfall variability and lake evaporation rates is important.

Monthly and annual rainfall data was derived and reported by Knight Piesold from a weather station at Dabakala (1922 to 2000)[51], whilst evaporation data is based on data obtained by Knight Piesold from Ferkessédougou (Elevation approximately 330 mamsl, approximately 160 km north-northwest of PE 58) and Zuenoula (elevation approximately 200 mamsl, approximately 180 km southwest of PE 58). The mean monthly and annual evaporation rate for PE 58 was derived by factoring[52] the data from the two sites. For Ferkessédougou and Zuenoula, the mean annual evaporation rates were approximately 1530 mm and 1196 mm respectively[53] (Knight Piesold, 2021).

For Dabakala, the highest and lowest annual rainfall events occurred in 1957 (1742 mm) and 1974 (605 mm) (Knight Piesold, 2021).

Rainfall data sourced from the World Meteorological Organisation (WMO, 2022)[54] and Knight Piesold is presented in Table 5-1 following. It is not noted that the mean annual evaporation (MAE) rate over a year exceeds the mean annual precipitation (MAP), meaning the that the site has a negative annual water balance. The monthly exceptions are June to August, where the rainfall exceeds evaporation.

Table 5-1: Rainfall and Evaporation Data

Month	Knight Piesold	Project Design Basis (Knight Piesold, 2021)[55]					(WMO, 2022)
	Dabakala[56] Average Rainfall (mm)	Average Rainfall (mm)	100 ARI Wet Annual Rainfall (mm)	100 ARI Dry Annual Rainfall (mm)	Average Pan Evaporation (mm)	Average Lake Evaporation (mm)	Rainfall days per month[57]
January	8.9	0	0	30	201	144	0.8
February	30.7	56	4	0	208	146	2.5
March	64.6	64	130	19	218	155	6.5
April	123.0	83	258	57	182	131	8.9

[50] Data originally sourced from https://weatherspark.com/y/34006/Average-Weather-in-Bouak%C3%A9-C%C3%B4te-d%E2%80%99Ivoire-Year-Round

[51] 25 km from Lafigue pit. Elevation at Lafigue circa (300 to 360) m, Dabakala~ 250 m. A few years of data was removed, based on incomplete data sets

[52] Based on location differences

[53] Pan to lake conversion factor of 0.73 was applied.

[54] Sourced from 'Direction de la Météorologie Nationale (SODEXAM)'

[55] It is noted that the synthetic wet years may have months where the precipitation is less than the average or the synthetic dry years. Similarly, the synthetic dry year may have months where the precipitation is greater than the average or the synthetic wet years. This is not an error but is due to the rainfall pattern within the specific year selected to develop the synthetic precipitation scenarios.

[56] **Invalid source specified.**

[57] Mean number of rain days = Mean number of days with at least 0.1 mm of rain (1981 to 2010).

Table 5-1: Rainfall and Evaporation Data

Month	Knight Piesold	Project Design Basis (Knight Piesold, 2021)[55]					(WMO, 2022)
	Dabakala[56] Average Rainfall (mm)	Average Rainfall (mm)	100 ARI Wet Annual Rainfall (mm)	100 ARI Dry Annual Rainfall (mm)	Average Pan Evaporation (mm)	Average Lake Evaporation (mm)	Rainfall days per month[57]
May	132.4	120	113	77	167	121	10.3
June	138.8	135	166	132	136	100	10.7
July	102.1	111	138	41	117	87	10.9
August	143.1	270	387	85	116	86	12.5
September	210.4	181	331	84	121	89	14.4
October	126.0	67	209	54	137	101	10
November	30.5	30	9	4	136	100	3.1
December	11.7	0	27	0	153	112	1.1
Total		**1119**	**1772**	**583**	**1891**	**1373**	

Given that rainfall and the collection/storage thereof from water harvest dams and boreholes is the only source of water for construction and operations. The impact of dry and wet season on water supply to the mine is discussed in Section 18.

High intensity rainfall days have been considered in mining, with five days of production lost per annum. No rain stoppage days have been allowed for in the process plant.

5.6.4 Wind

Based on data obtained from SODEXAM, (Cabinet Enval, 2021) state that winds in the Hambol area, originate from the south to close to west-northwest (180 to 280)°, with the 'mode' of the data from 2014 to 2018, being 220° (southwest). The average monthly wind speed over this period varied from (1 to 3) m/s, with an average of 2 m/s. Average monthly wind speeds of 3 m/s most frequently occur in May and June, and again in October and November. Instantaneous hourly air speed, 3s gusts at 10 m, and variability with respect to variation from the mean, are not stated.

Data from the UN (1961 to 1990) broadly corroborates the wind direction, with the exception of January, where it is stated the wind is from the North (360°) (United Nations, 1961 to 1990). Other sources such as Weatherspark (www.weatherspark.com), indicate that for Bouake, the wind can also come from the north and the east, and the 75th and 90th percentile average wind speed for the period from (July to August) can be as high as (4.2 and 5.0) m/s respectively.

On a Beaufort scale, the regional average wind speed would be classified as a '2' or 'Light Breeze' and in context, average wind speeds of (4 and 5.8) m/s would be required to drive small and utility scale wind turbines respectively (U.S Energy Information Administration, 2022).

In summary, average wind speeds are moderate and not sufficiently high enough to use for the generation of power. There is no evidence to suggest that the area is subject to wind gusts that would physically damage infrastructure.

The closest villages to the north and west of PE 58 operations that could be impacted by dust and noise carried by the wind are Lafigue (approximately 3 km from the mine) and Toledougou (approximately 4 km from the mine) respectively.

5.6.5 Dust

Background/seasonal dust levels in the area will not impact the design or operation of the facilities and the area is not subject to dust storms/events. Dust fallout from operations should be considered in any solar installation.

5.6.6 Solar Radiation and Sunshine Hours

The Hambol Region and by association PE 58, is subject to moderately high[58] levels of solar insolation (Table 5-2). Subject to permitting and techno-economic considerations, solar energy could be used in heat and power applications. In the integration of solar, consideration needs to be given to seasonal variability in daily sunshine hours and ambient dust levels.

Table 5-2: Solar Insolation (World Bank & International Finance Corporation, 2022)

Description	Abbreviation	Value
Specific photovoltaic power output	PVOut (Specific)	3.8 to 4.27 kWh/kWp[59]
Direct normal irradiation	DNI	2.59 to 3.46 kWh/m^2
Global horizontal irradiation	GHI	4.82 to 5.40 kWh/m^2
Diffuse horizontal irradiation	DIF	2.83 to 2.91 kWh/m^2
Global tilted irradiation at optimum angle	GTIopta	4.88 to 5.50 kWh/m^2

The region typically has seven to eight hours of sunshine per day from December to May (six months), six to seven hours of sunshine per day over October and November (2 months) and between 2.5 and four hours of sunshine per day from June to September (4 months) (World Data Info, 2022)

Over the year, sunrise varies between (0600 and 0630) hours, whilst sunset varies between (1755 and 1840) hours (Redwoods, 2022). Construction hours will be structured accordingly.

5.6.7 Extreme Weather Events

The Hambol region is not subject to extreme weather events that would likely stop production for long periods, or cause physical damage to infrastructure. One hundred year dry and wet Annual Return Intervals (ARI's) have been considered for rainfall in the design of the facilities and in the operation of the mine.

Lightning intensity during the wet season is moderate, and likely to be of the order of 10 to 30 strikes.km^{-2}.y^{-1} (Wikipedia, 2001). Suitable operating practices will be put in place during construction and operations to manage lightning risks.

[58] Values above 4.5 kWh/kWp are considered excellent**Invalid source specified.**

[59] P_{50} 3.93 kWh/kWp.

5.7 Seismicity

PE 58 is located within the West African Craton (WAC), one of five large masses or cratons, of Precambrian[60] basement rock, that make up the greater African Plate. The WAC stretches from the Little Atlas Mountains in Morocco to the Gulf of Guinea and is bounded by mobile belts of much younger rocks to the north, east and west. The seismicity of much of West Africa is typical of an intra-plate region, characterised by low levels of seismic activity and earthquakes randomly distributed in location and time. The correlation between recorded earthquakes and geological features is typically not well known or understood (Knight Piesold, 2021b).

As a guide and as indicated in Figure 5-5, PE 58 lies within an area of low seismicity, with an estimated Peak Ground Acceleration (PGA)[61] of less than 0.01 g or 0.1 m/s^{2}.

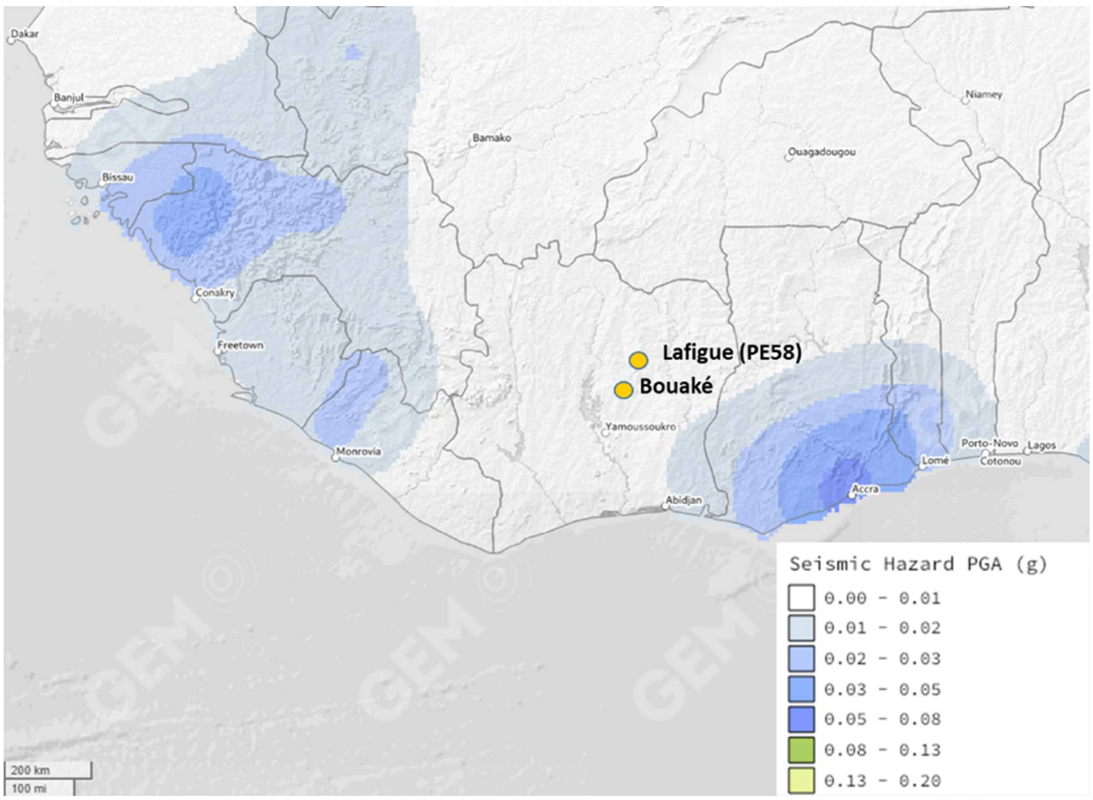

Figure 5-5: Seismic Hazard PGA (Global Earthquake Model, 2018)

For the purposes of the design of the tailings dam and associated structures, Knight Piesold undertook a site-specific hazard assessment to determine the relevant design parameters (Knight Piesold, 2021b). These parameters are reported in the Infrastructure section of this report.

[60] Formed 4.6 billion years ago.

[61] Peak Ground Acceleration (PGA) with a 10% probability of being exceeded in 50 years

5.8 Project/Mine Enabling Infrastructure/Facilities/Services

5.8.1 Rivers and Water Supply

Whilst there are two rivers in the area in close proximity to PE58, namely the N'zi (perennial) and M'bé rivers (Figure 5-2), water is not sourced from these rivers, nor is water discharged from the mine into these watersheds[62]. The N'zi is approximately 24 km east of the proposed pit, whilst the heads of the M'bé tributaries are approximately 21 km southwest of the pit.

Water for the project and operations will be sourced from a water harvest dam constructed on one of the non-perennial water courses that flows southeast through the property. This water will be supplemented with borehole water when required.

5.8.2 Ports

CI is serviced by two autonomous ports, one in Abidjan and one in San Pedro. Whilst both ports service the Issuer's interest in CI, only the Autonomous Port of Abidjan (APA)[63] will be used to provide a logistics function to the Lafigue Project/Mine.

The APA is one the most important ports in West Africa, is a major contributor to the economy of CI, and serves as the primary sea entry point for goods being transported to Burkina Faso, Mali, Niger, Chad, and Guinea. It is also serves as a key export point for CI's agricultural (cocoa beans/paste/butter/powder, rubber, cotton, nuts, fruits and timber) and mineral products (manganese ore).

Since 2013, a number of upgrades have been made to the APA (APM Terminals, 2022), (Bollore Ports, 2022) including:

- deepening and widening of the Vridi Canal and the reclamation of 45 ha of land (completed 2019);
- upgraded of the existing container terminal in 2015, which increased capacity from (0.8 to 1.5) TEUs and the construction of a new container terminal (TC2), which will expand container handling capacity to 2.5 M TEUs/a[64];
- construction of a new RoRo facility; and,
- construction of a mineral loading terminal (Ship Technology, 2020).

With the aforementioned upgrades, the APA will be the only port in West Africa capable of handling the new generation of cargo ships (14 000 TEUs and 350 m in length). Such ships should reduce logistics costs and reduce the CO_2 footprint of the logistics function.

With respect to port operators; Bollore Ports/APM terminals have a concession for the operation of the container terminals, whilst Terra have the concession for the RoRo terminal.

Road and rail links that facilitate the movement of goods from the APA to PE 58 are described more fully in Sections 5.8.3 and 5.8.4 respectively.

[62] The exception being clean rainfall run-off, after passing through sediment control structures.

[63] https://portabidjan.ci

[64] To be commissioned in November of 2022 (www.apmterminals.com/en/cit/about/our-terminal)

From the time of arrival of the ships at the Abidjan harbour entrance, the delivery time for goods to PE 58 is expected to be between (14 (standard scenario) and 21 (worst case scenario)) days.

5.8.3 Road Access

The distance by road from the APA/Abidjan to PE 58 is approximately 470 km, of which approximately 453 km is paved. The last 16 km of the route is via a public laterite road, which will likely be maintained by the Issuer. The route followed is as defined below and illustrated in Figure 5-1 and Figure 5-2.

- From the APA/Abidjan routing is in a north-northwest direction via the A3[65] from Abidjan to Katiola (393 km).
- From Katiola one follows the B412 northeast to Koundoudougou (60 km) and then south via a laterite road to PE 58 (16 km).

As part of CI's National Development Programmes (Section 5.3), the A3 from Abidjan to Burkina Faso is being upgraded and will likely eventually be a dual carriage, double line highway connecting Abidjan with Burkina Faso. The upgrade of the A3 is being rolled out in phases, as defined below. It is likely that by the time mine enters operation, 340 km of the route to site will be via a dual carriage, double line highway (Traore, 2022).

- Dual carriage highway from Abidjan to Yamoussoukro - complete.
- Dual carriage highway from Yamoussoukro to Tiebissou started in October 2017 and is to be completed by the end of 2022.
- Dual carriage highway from Tiebissou to Bouaké started in November 2018 and is likely to be complete by the end of 2023.

5.8.4 Rail

As illustrated in Figure 5-1, the 1245 km, 1000 mm gauge rail line connecting the APA and Ouagadougou in Burkina Faso lies approximately 55 km west of PE 58. The closest railway station to the proposed mine is in Katiola, 76 km by road from PE 58. Rail infrastructure at Katiola would likely need to be upgraded to include a rail siding if this station is to be used by the Issuer for the transport of goods.

The two largest rail stations/sidings in the area are in Bouake (80 km southeast of PE 58) and Ferkessédougou (160 km north-northwest of PE 58)[66]. The railway station at Ferkessédougou is being developed as a fully equipped dry port and transit point for goods destined for the cities of Korhogo, Boundiali , Odienné and the northern part of the country, but is also a very promising interface for the import and export of goods into and from Mali (World Food Programme, 2022).

Whilst originally in state hands, rail since 1995 has been managed by 'SITARAIL'. Since 2017, both rail infrastructure and rolling stock have been undergoing a refurbishment/upgrade process, with the objective being to transport 1 Mt of goods and 300 000 passengers per annum by 2023 (Oxford Business Group, 2022) and (Traore, 2022).

SITARAIL is private consortium led by the Bollore group and is jointly owned by the governments of CI and Burkina Faso (15% each), the Railway Employees Trust Fund 3% and a French/Danish business group (freight operators Bollore, SAGA and MAERSK, consultancy bureaux SYSTRA and TRANSURB) for 67% (World Food Programme, 2022).

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[65] The A3 is the primary access corridor for the movement of goods by road between the APA/Abidjan and Ouagadougou in Burkina Faso (1189 km)

[66] Circa 250 km by road

Whilst not currently incorporated in the basis of construction or operations, the service provided by SITARAIL could be used for the transport of goods and people by the Issuer.

5.8.5 Power

5.8.5.1 Country Overview

With an installed generation capacity of 2199 MW (2018), CI's electrical generation capacity, is the third largest in West Africa, after Nigeria and Ghana. Power capacity is dominated by natural gas-fired generation (1320 MW)[67] and hydropower (879 MW) (IFC, 2018).

Despite significant hydropower capacity, the generation mix (energy generated in megawatt hours (MWh)) is dominated by gas. Natural gas Independent Power Producers (IPPs) represent 60 per cent of total Ivorian production (with a 68 per cent capacity factor), while hydro production is 40 per cent (with a 27 per cent capacity factor) (IFC, 2018).

Traditionally, contract structures have favoured the gas producers. As gas capacity increases under the same contractual framework, it has been difficult to displace gas production with renewables. Therefore, while in the near-term gas is cheap because of offshore supplies, in the longer-term more diversification will be required (IFC, 2018).

Whilst CI has both proven oil and natural gas reserves (100 M barrels and 1 trillion cubic feet)[68] (U.S International Trade Administration, 2021b) these will only last approximately 10 years at current production rates (based on proven reserves only)[69]. Further, whilst Total Energies (Total) was to build an LNG Terminal and regassification facility in Abidjan to offset declining gas production, they have since exited the project, and Côte d'Ivoire's Ministry of Mines, Petroleum and Energy, via state electricity company Cote d'Ivoire Energies (CI Energies) issued a tender in June of 2021, for a provider and operator of an FSRU in the Port of Abidjan (Gobal Energy Monitor, 2022). Gas producers in CI are PETROCI, CNR, FOXTROT and CNR (Kone, 2022).

The 2011 National Strategic Action Plan for the Development of the Electricity Sector aimed to increase total installed capacity to 3000 MW by 2020[70]. This was to be achieved through public private partnership arrangements, with hydropower and solar as priority technologies (IFC, 2018). The Government has also committed to meeting demand growth by increasing installed generation capacity by approximately 150 MW/a, mostly through IPPs (USAid, 2022).

Electricity pricing is regulated by the government, with average tariffs closely grouped in the range of 0.10 EUR/kWh across different voltage categories (Get, 2022). Whilst there is some pressure to rise prices along the value chain, the electricity price has largely been constant in 'XOF' terms from 2016 to 2021 (Kone, 2022). With respect to electricity pricing in 2021 and CI's global competitive position, CI ranks 72nd out of the 147[71] countries surveyed[72] (GlobalPetrolPrices.com, 2021).

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67 1390 MW end 2021**Invalid source specified.**

68 13.9 M barrels produced and 0.074 trillion cubic feet extracted in 2020 (Endeavour calculation based on U.S International Department of Trade Data). See also www.worldometers.info/gas/cote-d-ivoire-natural-gas

69 A new oil (1500 to 2000 M barrels) and gas (<2.4 trillion cubic feet) find was announced in 2021 **Invalid source specified.**

70 Not met

71 Most expensive country

72 40% higher than China, but 26% less than the US.

Whilst CI is in control of the majority of their energy feedstocks (water, solar and gas), diminishing proven gas reservices and global increases in PPI/ CPI, is likely to put pressure on electricity pricing over the longer-term.

As of 2018, electrification rates in urban and rural areas was 95% and 32% respectively (Get, 2022). Rural electrification is expected to increase over the coming years as detailed more fully herein.

In CI, significant actors in the electricity market include:

- Ministry of Mines, Petroleum, and Energy (MMPE) - policy, oversight, and provides some regulatory functions;
- The Côte d'Ivoire Electricity Regulation Authority (ANARE-CI) - regulates the country's three independent power producers (IPPs).
- Côte d'Ivoire Energies (CI-ENERGIES): the state-owned entity responsible for monitoring and managing of electrical energy production. It also manages projects for which the State is acting as the conceding authority. In 2020, the government awarded a 12-year electricity distribution concession to Compagnie Ivoirienne d'électricité (CIE)[73].

CI's country energy generation and energy usage is illustrated in Table 5-3, whilst CI's energy development plans through to 2030 are illustrated in Table 5-4 and Table 5-5 following.

In this context:

- power demand is expected to continue to grow at more than 7 per cent per year until 2025, driving demand for limited natural gas, which is needed for industry and gold mining. (IFC, 2018);
- CI's power usage per capita is low at 364 kWh/a, when compared with western developed countries, which typically consume greater than 5000 kWh/a per capita (Our World in Data, 2021);
- As part of CI's climate commitment to reduce CO_2 by 28% by 2030[74], the Master Plan for the Production and Transport of Electric Power (2014 to 2030), calls for USD 14.3 billion investment, with an addition investment in the mining sector of USD 2.87 billion. (African Development Bank, 2019)

Table 5-3: Country Energy Generation and Use (U.S. International Trade Administration, 2021)

Description	2018	2019	2020
National Demand for Electricity (GWh)	8180	9095	10 020
Total Export of Electricity (GWh)	1078	1256	1333
Total gross national production of electricity (GWh)	997	10 613	11 210
Total Installed (MW)	2199	2249	2249

[73] Managed by CI-Energies and owned by Eranove with 54% shareholding, is involved in generation, transmission and distribution of electricity. They also manage IPPs**Invalid source specified.**

[74] Submission to UNFCCC relating to Nationally Determined Contributions (NDC), includes detailed submissions of the country's major commitments. The 28% is an unconditional reduction target. Base Level (BAU) 35.78 Mteq CO_2/a (date of Base Line not stated).

Table 5-4: CI Power Sector Requirements and Commitments (World Bank, 2017)

Programmes	Required Investment USD (M)	Commitments USD (M)	Donors/Financiers
Transmission Master Plan (2016 to 2030)	2000	1081	BOAD (West African Development bank), AfDB, World Bank, China
Distribution Master Plan (2016 to 2030)	680	177	AfDB, World Bank, EU, China
Rural Electrification Master Plan (2016 to 2020)	675	219	World Bank, EU, China
E4All Programme[75]	270	21	EU, AFD (French Development Agency), World Bank

Table 5-5: Current and Planned Generation Projects (African Development Bank, 2019)

Installed Capacity (MW)	2017	2030 Plan	2030 Pipeline	Cost USD/W
Thermal - gas	1320	2548	2728	
Thermal- coal		1400	1400	
Hydro	879	1891	1891	3.40[76] to 3.84[77]
Solar		420	320	1.54
Biomass		500	236	1.82
Total Installed Capacity (MW)	**2199**	**6759**	**6575**	
Total thermal	1320	3948	4128	
Total renewable	879	2811	2447	
Shortfall versus plan			-13%	

A 700 MW thermal coal plant at San Pedro was to be commissioned in 2021, is yet to break ground and is unlikely to do so (Global Energy Monitor, 2022b). This power plant would have provided additional base load capacity, and would have assisted in addressing seasonal issues associated with hydro power as an energy source. Notwithstanding this loss in thermal base load generation capacity, approximately 1340 MW of new combined cycle gas turbine capacity is still planned (IFC, 2018).

It is unclear whether the 1000 MW of hydropower dams and run-of-river schemes proposed over the next 10 years will improve the capacity factor of hydro, which has been low to date (circa 27%).

Whilst the government's commitment to reduce CO_2 emissions will drive solar and hydro installations, additional gas generation capacity will likely be required to meet daily/seasonal shortfalls in energy production. With the failure to build the San Pedro Power Station and the Abidjan LNG terminal, the power rationing noted in 2021, could become more frequent over the next 10-years. However, subject to development cycles times, the recent gas finds in CI may mitigate this, with additional combined cycle gas capacity bought online.

[75] Endpoint distribution programme to consumers
[76] Run of river (RoR)
[77] Dam

It is unclear how switching on/off natural gas thermal production to make-up for daily/seasonal shortfalls in renewable generation, will affect energy pricing over the longer-term. It will likely impact the developer's business case for gas fired power. Similarly, low gas prices, coupled with low levelised costs for hydro power, will likely impact the business case for renewables.

5.8.5.2 Mine Power Supply

The Project benefits from CI's investment in 225 kV transmission infrastructure noted in Table 5-4 and Figure 5-6 following. Whilst there is a 225 kV switch yard at Katiola, the Project will tie into a new 225 kV switchyard in Dabakala, and a new 32 km 225 kV line will be installed from Dabakala to the mine. The Issuer will build/fund the transmission line, with handover to CI Energies. Change of ownership will largely be at the 225 kV incoming busbar (also CI's place of power measurement).

The new Dabakala switchyard is on a new 225 kV ring main that forms part of the main link from Yamoussoukro to Ferkessédougou (ECG Engineering, 2021).

ECG Engineering have indicated that on the 225 kV transmission network, power quality should be good, and power availability should be in excess of 98%. Whilst low rainfall/dam levels and other factors led to 'load shedding/'power rationing' in-country in 2021, ECG state that heavy industry are likely the last to be load shed (ECG Engineering, 2021).

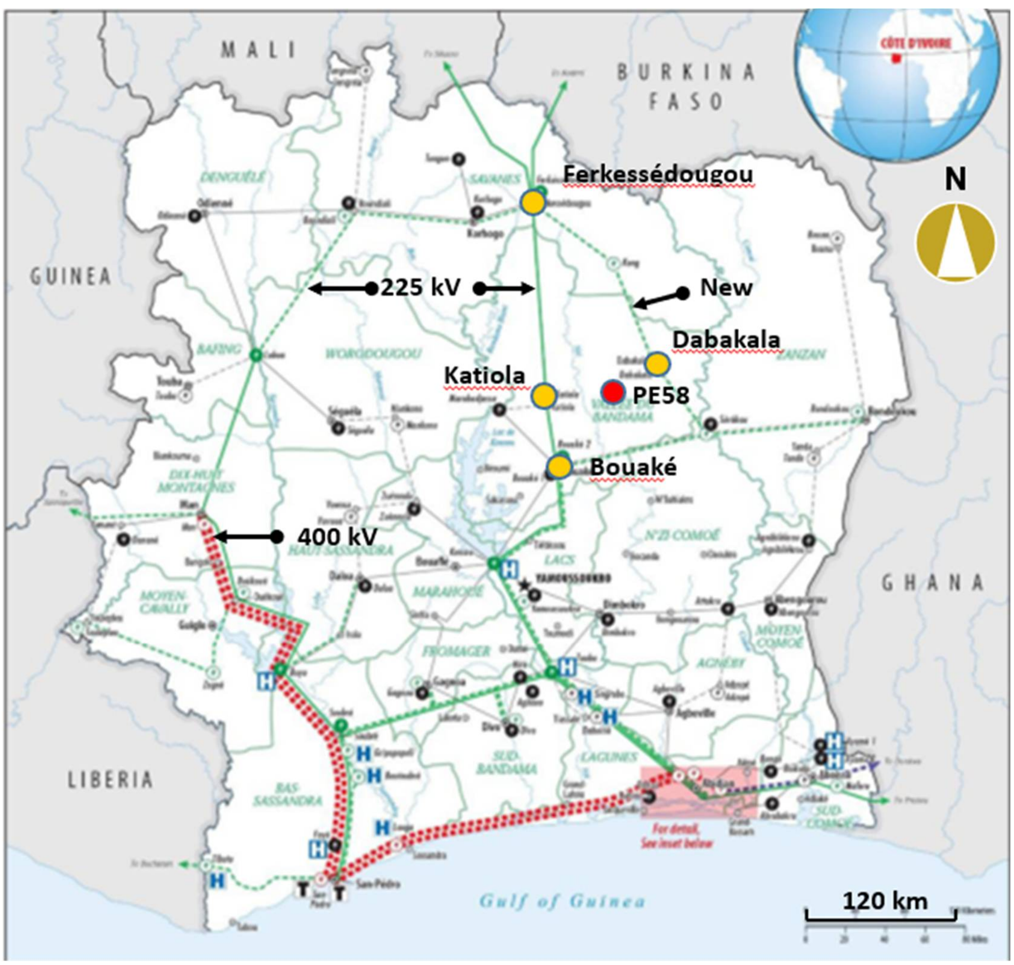


Figure 5-6: Cote d'Ivoire Electrical Distribution Network (World Bank, 2017)

5.8.6 Fuel Supply

CI is largely self-sufficient in fuel (gas and liquid) with offshore oil, and refining capacity in Abidjan. Light fuel oil (diesel) and petrol (gasoline) is distributed from Abidjan by either; road, rail (to Bouake and Burkina Faso) or pipeline (to Yamoussoukro)[78] (Deyres, 2022). Notwithstanding this, CI is both an importer and exporter of refined petroleum products (Observatory of Economic Complexity (OEC), 2022).

Abidjan is home to four fuel storage terminals, owned by four different companies. Three companies can supply the CI market, whilst the fourth company Puma Energy, can only supply to neighbouring countries. Additional fuel storage depots are located in Yamoussoukro and Bouake (Deyres, 2022).

Until such time as fuel supply agreements are signed, the sourcing/depot for fuel supply is not defined. For operations, the Issuer's policy is to have 15 to 20 days of fuel storage capacity at the mine to cover for unplanned fuel supply interruptions.

Based on in-country capacity, no fuel supply issues are foreseen.

5.8.7 Airports and Airstrips

CI has only one international airport (Félix-Houphouët-Boigny International Airport(IATA: ABJ, ICAO: DIAP)), which is located 16 km southeast of Abidjan. The airport is a modern international airport, typically handling 2 M passengers per annum. Whilst there is a Civil Aviation Authority (CAA) registered laterite airstrip at Katiola (IATA: KTC) and a regional paved airport (3300 m runway) at Bouaké Airport (IATA: BYK, ICAO: DIBK), these are unlikely to be used by the Issuer given that a laterite airstrip (1000 m long) is currently being built on PE 58. Air Côte d'Ivoire is the only airline flying domestically, and flights to Katiola or the Issuer's airstrip on PE 58 will either be by; the Issuer's aircraft (Pilatus PC-12), charter aircraft (MEDEVAC or other) or Brinks (gold transporter).

5.8.8 Communications

5.8.8.1 Overview

CI has one of the most developed telecommunications sectors in West Africa. The sector constitutes approximately 10 percent of GDP and annually, contributes USD 800 M in tax revenue. The national telecom regulator, Telecommunication Regulatory Authority of Côte d'Ivoire (ARTCI), is actively issuing decrees, to update the 2012 Telecom Code (U.S International Trade Administration, 2021).

CI formally adopted a broadband roll out in policy in 2016 'Le Réseau National Haut Débit (RNHD)', where RNHD will roll out some 7000 km of fibre across CI (UNESCO, 2017). In addition, MTN of South Africa are also rolling out a fibre network in CI (5500 km installed, 4400 km planned) (MTN, 2021).

Although there are two fixed network operators, the market is dominated by Orange Group's local unit, Orange Côte d'Ivoire. The mobile market is more competitive, with Orange Côte d'Ivoire operating alongside MTN Côte d'Ivoire, and Moov (a subsidiary of Maroc Telecom), all of whom currently provide 2G/3G/4G coverage.

[78] A fuel pipeline is planned between Yamoussoukro and Ferkessédougou.

5.8.8.2 Fibre

CI is connecting with the rest of the world via four independent submarine communication cables (Africa Coast to Europe (ACE), South Atlantic 3/West (SAT-3/WASC), GLO-1 and Main One) (International Telecommunication Union (ITU), 2022).

With respect to PE 58, there are two fibre lines (one owned by MTN and the other by RNHD(state)) that run adjacent to the A3 highway connecting Abidjan to Ferkessédougou, with the closest connection point to PE 58, being at Katiola. RNHD are currently constructing fibre lines between Bouaké, Basawa and Dabakala, whilst a fibre line between Dabakala and Katiola is planned.

Until such time as the Issuer can establish a direct fibre connection, a microwave link with either Orange or MTN, will be the primary basis for connection to CI's fibre backbone. Said link is discussed more fully in the Infrastructure section of this report.

CI Energies the state power utility, install OPGWs (optical ground wires) on their transmission lines. However, the use of these wires for communications, is currently reserved for CI Energies.

5.8.8.3 Cellular

For the Dabakala/Katiola region, it is likely that MTN/Orange will be the preferred service provider. The communication system between the service provider and the Mine is discussed more fully in the Infrastructure section of this report.

5G services are currently in their infancy in CI, albeit this is expected to become more mainstream from 2023 onwards (General Confederation of Businesses of Côte d'Ivoire (CGECI), 2022).

5.8.8.4 Radio

Radio coverage for the mine operations and aircraft is discussed more fully in the Infrastructure Section of this report.

5.8.9 Location of Towns and Supporting Service Centres

5.8.9.1 Overview

As illustrated in Figure 5-2, there are number of medium sized towns of greater than 10 000 persons within a 50 km radius of PE 58, namely; Katiola, Timbe, Bonieredougou, Niemene and Dabakala and to the south, Satama-Sokoro. Additionally, there are a number of medium sized villages (1000 to 10 000 persons) and small villages of up to 1000 persons, largely concentrated in an area from the north-northwest to the east of PE 58.

The local economy is largely agrarian in nature, supplemented with money earned through trading and artisanal mining.

Whilst there are educational facilities in the area, few persons will have completed a secondary education, and literacy rates are low, circa 20% (RGPH 2014).

5.8.9.2 Other Industries and Supporting Services

The regional economy is largely agrarian, with some textile industries, government institutions, and higher education facilities in Bouake. Whilst the region is slowly re-building after the civil wars, it is likely that mine supporting services will be sourced out of Abidjan and/or abroad. Notwithstanding this, the Issuer will through its foundation, engage in training and development and setup new businesses for the provision of goods and services to the mine as part of the Issuer's Environmental, Social, and Governance (ESG) initiatives.

5.8.9.3 Medical Facilities

Outside of primary care, there are limited medical facilities/services in the surrounding towns/villages. A clinic is provided on site for employees, with referral facilities identified in Bouake, Yamoussoukro and Abidjan for secondary/tertiary care. Where air MEDEVAC is required, flights are organised out of Abidjan. Further work is required to ensure that employees dependants are adequately provided for.

5.9 Land Use and Conditions

On PE 58, the issuer has sufficient land to construct and operate the Lafigue Mine. There are no villages on PE 58, that would impact the mine's operations. Additionally:

- the Mining License for PE 58 has been obtained (22/09/2021).
- the laterite road from the B142 to PE 58 is a public road which is being upgraded by the issuer, and no issues are foreseen in its use.
- the air strip on PE 58 will likely be licensed by the CI's Civil Aviation Authority (CAA) by end 2022;
- the power supply ESIA is complete and an agreement with CI Energies is in progress;
- there are no seismic, faults, or geotechnical issues known that would influence the cost of construction and operations; and
- the use of a non-perennial natural water course on PE 58 for filling of a water harvest dam and the supply of water has been modelled by Knight Piesold, and no issues have been identified based on ARI 100-year dry and wet season assumptions.

5.10 Data Verification

The approach taken to verifying the data used and presented in Section 5, is discussed in Section 12 of this Report.

5.11 Comments on Section 5

The QP for Section 5 is of the opinion that the Issuer has the rights to develop and operate the Lafigue Project/Mine on PE 58 and there is the requisite supporting infrastructure (transport/logistics, fuel, communications and power) in place to support the development and operation of the mine.

PE 58 is not in a high-risk seismic area, nor subject to storm events that are likely to materially impact operations. As long as Knight Piesold's assumptions and modelling are correct, no water supply issues are foreseen.

There are some concerns with respect to the local availability of skilled human resources. Notwithstanding this, the Issuer/mine and its various service providers will need to invest in the training and development of local people and be largely self-reliant, with respect to maintenance.

5.12 Interpretations and Conclusions

Interpretations, conclusions and risks for Section 5, are presented in Section 25 of this Report.

5.13 Recommendations

Recommendations for Section 5 are presented in Section 26 of this Report.

5.14 References

References cited in the preparation of Section 5, are presented in Section 27 of this Report.

6. HISTORY

6.1 Introduction

This section describes the discovery, ownership and early exploration and resource definition history of the Project, prior to Endeavour Mining plc ('Endeavour' or the 'Issuer'), taking an active role in ownership and exploration from 2017 onwards.

6.2 Historical Ownership

Historical property ownership and ownership changes are summarised in part in Section 6.3, and detailed more fully in Section 4 of this Report (Lycopodium, 2022).

6.3 Historical Exploration and Development

The earliest exploration work across the Project area commenced in 1935, when the 'Bureau Minier of the France d'Outre-mer' (Bumiform) conducted geological mapping. Its successor Bureau de Recherches Géologiques et Minières (BRGM)[79] and Société pour le Développement Minier de la Côte d'Ivoire (SODEMI) conducted airborne geophysical surveys during the late 1960s and early 1970s.

In 1996 an exploration, development and operating agreement was entered into by SODEMI, the Permit title holder and GENCOR Limited (GENCOR), through its Ivoirian company GATRO-Côte d'Ivoire (GATRO-CI) for PR 57 (the Exploration Agreement). According to this Exploration Agreement, the exploration campaigns were undertaken by GATRO-CI, SODEMI, BRGM, and by the Australian mining group Normandy Mining Ltd (Normandy), through its Ivoirian company subsidiary La Source. Through the agreement, GATRO-CI completed a series of regional stream sediment and soil geochemistry surveys, exploration pits and trenches, and a small amount of drilling (14 diamond core drillholes and 37 reverse circulation holes), and defined four main targets, including Lafigué.

In 1999, Compagnie Minière Or (COMINOR) took over La Source's participation and the GATRO-CI contractual commitments under the Exploration Agreement.

Between 1999 and 2002, COMINOR conducted exploration works, including exploration drilling in 2002 comprising of 1803 m of rotary airblast (RAB) drilling, 1281 m of reverse circulation (RC) drilling and 461 m of diamond core (DD) drilling, which demonstrated mineralisation was not continuous between Lafigué Centre and Lafigué North and that locally, felsic dykes play a role in controlling some mineralisation.

Due to the civil war affecting Côte d'Ivoire (CI), exploration works were suspended from 2002 until 2010. When COMINOR recommenced exploration works in 2010, a further 11 RC holes (1109 m) and 4 DD holes (396 m) were drilled to assess the down-dip extents of mineralisation.

In 2014, La Mancha Côte d'Ivoire SARL (LMCI) replaced COMINOR in the partnership with SODEMI, leading eventually to a transfer of PR 329 from SODEMI to LMCI in 2020 and the granting of PR 58 to LMCI (and then Société des Mines de Lafigué SA (SML)) in 2021.

[79] Formed from the merger of four French Mining Research Departments, namely: the Bureau of Geological, Geophysical and Mining Research (BRGGM), the Mining Bureau of Overseas France (BUMIFORM), the Bureau of Mining Research in Algeria (BRMA) and the French Guiana Mining Bureau (BMG) (Jesus M, 2017) (Wikipedia, 2022).

LMCI conducted further exploration on PR 329 in 2014, comprising of 23 DD holes (1864 m) and 54 RC holes (4634 m), focusing on Lafigué North, as well as obtaining structural data to better understand mineralisation controls. The majority of historical boreholes were resurveyed by differential GPS in 2014 by Environnement Technologie Côte d'Ivoire, with the exception of the RAB holes and three RC drillholes completed in 1997, which could not be located (R2087, R2997, R30B97).

A summary of exploration drilling completed on PR 57 and PR 329 prior to Endeavour ownership is provided in Table 6-1, whilst other exploration activities are summarised in Table 6-2 following. The location of the GATRO CI stream sediment (1994) and GATRO CI Lafigué soil geochemistry (1995) surveys and the location of the Lafigué South, Centre, and North drilling programmes (1997) are illustrated in Figure 6-1 and Figure 6-2 following.

Table 6-1: Summary of Exploration Drilling Prior to Endeavour Ownership

Year	Drilling Type	Number of Drillholes	Metres (m)
1997	DD	14	1447
	RC	37	1549
2002	DD	11	461
	RAB	94	1803
	RC	32	1281
2010	DD	4	396
	RC	11	1109
2014	DD	23	1864
	RC	54	4638
Total		**280**	**14 548**

Table 6-1 Note: Drilling Types: DD = Diamond Core Drilling; RC = Reverse Circulation

Table 6-2: Summary of Historical Exploration Activity at Lafigué and Surrounding Areas

Company	Year	Activity	Primary Results
BRGM. Canadian Aero Mineral Surveys Ltd	**1965 to 1968**	**Régional magnetic airborne survey**	
		Kenting Pty Ltd mission 1973 to 1976	
GATRO CI	**1994**	**Stream sediment geochemistry** 2143 samples taken (1 per 1.2 km²) 1970 samples analysed for gold 1006 ICP analyses	4 gold anomalies: Sandérékro, Tibéguélé, Lafigué, Sarakakro
GATRO CI	**1995**	**Soil geochemistry (Lafigué anomaly)** 200 x 100 m spacing (locally (100 x 50) m) 1862 samples Au analyses (detection limit : 5 ppb) **Rock samples** 35 samples taken and Au Analysed.	4 gold anomalies: Lafigué A: Main anomaly, 1700 m long and 250 m wide Lafigué B: 2 values >2 g/t Au but not validated. Work stopped in this area Lafigué C: Close to a granodiorite, 300 m long and 100 m wide Lafigué D: Flat area. Work stopped in this area Lafigué A: 3.4 g/t and 6 g/t Au in rock samples
GATRO CI	**1996**	**LAFIGUÉ A anomaly** **Soil geophysics** **(magnetics, VLF-EM, PP)** Mag. and VLF: 130 km lines with a 100 x 30 m spacing Induced polarisation: 4 profiles (7.4 km) Trenches - 26 trenches (3800 m) 2154 samples taken (horizontal grooving) 264 samples (vertical grooving)	Mineralisation typically low grade and disseminated (0.4 to 0.8 g/t Au). Highest grade (up to 50 g/t Au) associated with quartz tourmalines veins. Channel samples in the overburden returned an average grade of 3.8 g/t and 1.3 m average length. Some mineralisation was intersected in vertical groove samples (1.3 m @ 3.8 g/t).

Table 6-2: Summary of Historical Exploration Activity at Lafigué and Surrounding Areas

Company	Year	Activity	Primary Results
GATRO CI	**1997**	**Diamond core drilling campaign**	Gold associated with weathered schists and mylonites in metavolcanic rocks. The rocks are locally sheared.
		14 drillholes (1447 m)	Coarse gold associated with quartz veins with sericite and tourmaline.
GATRO CI	**1997**	**LAFIGUÉ 'A" Anomaly** **Exploration pits:** (100 x 100) m spacing. Infilled to (50 x 50) m 313 pits completed (depths from (0.45 to 8.7) m) 1018 grooved samples **RC drilling campaign** 37 RC drillholes (1549 m) **Mineral processing test** 3 oxidised samples were tested: • One from the overburden quartz • One from an in situ quartz vein • Disseminated (rock-hosted) gold Initial gravity separation (Knelson concentration), with the rejects cyanide leached.	First calculation of the mineralized quartz potential close to the surface and in the saprolite: Estimate (using a density of 1.57 g/cm³): Tonnage (t): 978 797 Grade (g/t Au): 2.54 Au metal (t): 2.5 Tested mineralisation continuity in the centre and southern areas. **Results:** Gravity recovery: • 84 to 88% recovery • 86 to 89% recovery • 17 to 35% recovery Cyanide leaching and gravity recovery: • quartz vein: 98 to 99% • disseminated (rock-hosted) gold: 64 to 83%
COMINOR	**2001**	**Metallurgical Testwork – Lafigué** Inventory of RC 1997 duplicate samples. 40 samples were sent to SGS for analyses. 'A' and 'Z' samples generated for insertion during the RAB and RC drilling campaign.	Mineral processing test (bottle test) of 36 samples from overburden mineralisation (12 holes, 21.35 m).

Table 6-2: Summary of Historical Exploration Activity at Lafigué and Surrounding Areas

Company	Year	Activity	Primary Results
COMINOR	**2002**	**Lafigué 'A' anomaly drilling:** • RAB: 94 holes, 1803 m, 12 profiles • RC: 32 drillholes, 1281 m, 17 profiles • DD: 11 drillholes, 461 m, 8 profiles	RAB Targets: • Continuity between the centre and the northern area, search for extension, 'C' anomaly checking. Results: No continuity, 200 m Eastern extension, no positive results on C. • Centre area extension, increase the resources. Results: discontinuity on the centre zone linked with a felsic dyke N125°E, North: discontinuity also explained by a felsique dyke N160°E. • DD and RC Targets: Main mineralisation confirmation, density measurements. Mean density (t/m^3): saprolite: 2.0, Oxidised zone: 2.1, transition zone: 2.5, sulphide zone: 2.8.
COMINOR	2010	Drilling campaign (Lafigué): • RC: 11 drillholes, 1109 m • DD: 4 drillholes, 396.30 m	Targets: - Check the mineralisation extension downdip on the centre area, check the extensions. Results: LFDD10 and LFRC10 cf. Table 6: Lafigué best mineralised intercepts (cut off: 0.5 g/t, intercepts with an average grade superior at 1 g/t, trenches excluded).
LMCI	**2014**	**Drilling campaign (Lafigué):** • RC: 54 drillholes, 4634 m • DD: 23 drillholes, 1864 m	Testing of Lafigué deposit extensions.
LMCI	**2015**	DGPS survey of collars	
		LIDAR survey done by AOC	

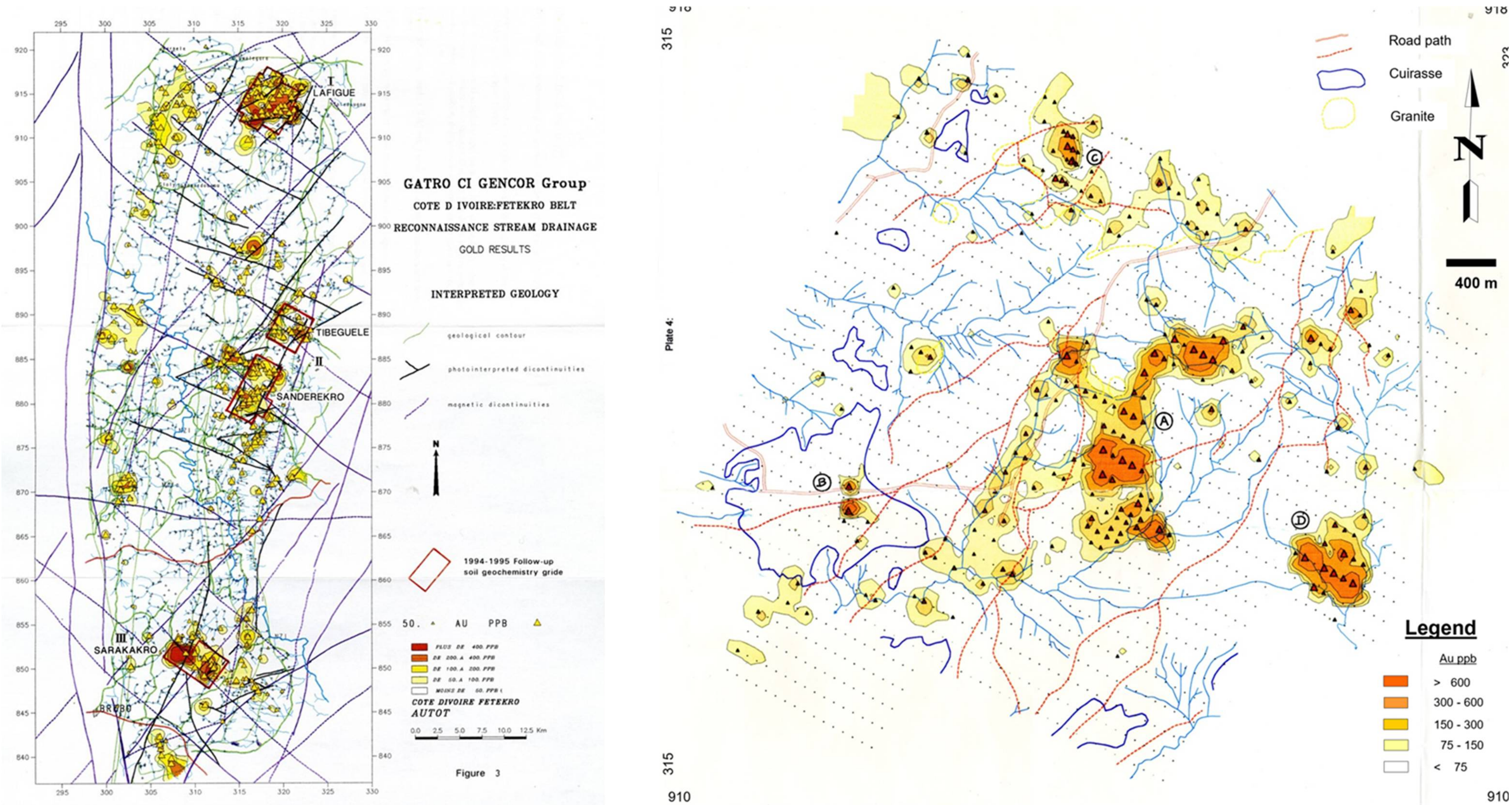


Figure 6-1: GATRO CI Stream Sediment (1994 - left) and Lafigué Soil Geochemistry (1995-right) Surveys

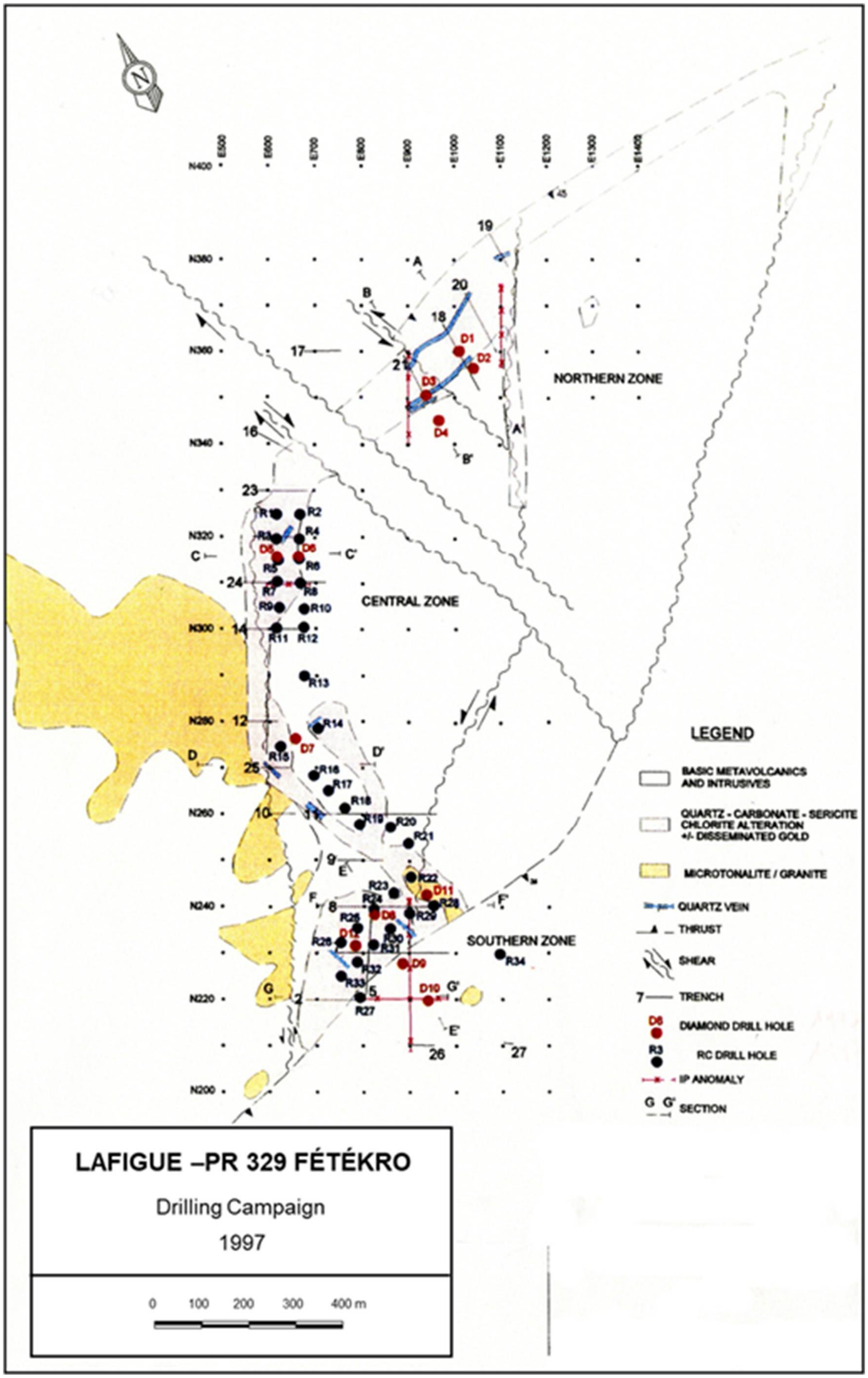


Figure 6-2: Lafigué South, Centre and North and GATRO CI Drilling (1997)

6.4 Previous Mineral Resource Estimates

A historical Mineral Resource estimate was completed for the Lafigué deposit by GATRO-CI, however the Issuer understands the estimate was for internal use only and was not reported publicly, or within any regulatory environment.

The last Mineral Resource estimate conducted prior to Endeavour's ownership of Lafigué was completed in 2003 by COGEMA, based on an updated geological model and density measurements obtained after the 2002 estimate was issued. The 2003 Mineral Resource Statement was not classified, but was split into North, Centre and South zones, as detailed in Table 6-3. This estimate was not reported publicly or in accordance with any internationally recognised codes or regulations. The 2003 estimate has not been reviewed by the authors and should not be considered a current Mineral Resource Estimate.

Table 6-3: COGEMA 2003 Preliminary Mineral Resource Estimate for Lafigué

Area	Oxide Zone			Sulphide Zone			Total		
	Tonnes (kt)	Grade (g/t Au)	Metal (t)	Tonnes (kt)	Grade (g/t Au)	Metal (t)	Tonnes (kt)	Grade (g/t Au)	Metal (t)
North	655	1.81	1.22	299	1.87	0.56	914	1.94	1.78
Centre	550	2.49	1.37	606	3.89	2.36	1157	3.23	3.73
South	315	1.50	0.47	-	-	-	315	1.50	0.47
OVB*	1288	2.30	2.96	-	-	-	1288	2.30	2.96
Total	**2769**	**2.17**	**6.02**	**905**	**3.22**	**2.91**	**3674**	**2.43**	**8.94**

Table 6-3 Notes:

- *OVB = Overburden
- Reported above a 1 g/t Au cut-off grade
- Rounding may result in apparent summation differences between tonnes, grade and contained metal content

As detailed in Table 6-4, subsequent Mineral Resource Estimates have been completed by Endeavour on an annual basis between 2017 and 2020, and by SRK Consulting (UK) in 2021,. Endeavour highlights that each of the Mineral Resource estimates completed between 2017 and 2021 is superseded by the Mineral Resource Statement presented herein.

Table 6-4: Mineral Resource Estimates for Lafigué from 2017 to 2021

Author	Date	Indicated			Inferred			Basis
		Tonnes (kt)	Grade (g/t)	Au (koz)	Tonnes (kt)	Grade (g/t)	Au (koz)	
Endeavour	October 2017	4981	2.34	375	898	2.19	63	USD 1500/oz pit shell, cut-off 0.5 g/t Au
Endeavour	October 2018	6833	2.25	494	3039	2.25	225	USD 1500/oz pit shell, cut-off 0.5 g/t Au
Endeavour	October 2019	14 577	2.54	1190	867	2.17	60	USD 1500/oz pit shell, cut-off 0.5 g/t Au
Endeavour	October 2020	32 030	2.40	2471	820	2.52	66	USD 1500/oz pit shell, cut-off 0.5 g/t Au
SRK	September 2021	44 805	2.02	2917	3559	2.36	269	USD 1500/oz pit shell, cut-off 0.4 g/t Au (oxide) and 0.5 g/t Au (transition and fresh)

6.5 Mine Production History

PE 58 has not been mined on a commercial scale, however, there has been significant artisanal mining works, primarily targeting the quartz-tourmaline vein-hosted mineralisation. Since September 2021, Endeavour alongside the Dabakala Gendarmes, have been undertaking an eviction exercise, whereby the majority of artisanal miners have been removed from site.

6.6 Project Milestones and Development Status

As outlined previously, an historical Mineral Resource estimate was completed by GATRO-CI (approximately 2002). The estimate was for internal use only and was not reported publicly or within any regulatory environment. In 2003, COGEMA issued an MRE update which incorporated new geological and density information, however the estimate was not classified.

Endeavour later issued four MRE updates in 2017, 2018, 2019 and 2020, all associated with new drilling results. In 2021, SRK issued an MRE update, which formed the basis for the Lafigué Project Pre-feasibility Study (PFS) and associated NI 43-101 Report (Lycopodium, 2021).

In 2022 SRK issued the latest MRE, which forms the basis for the current Definitive Feasibility Study (DFS) and NI 43-101 Technical Report.

6.7 Comments on Section 6

The QP has reviewed all available documentation regarding the historical ownership, exploration, and development activities for the Project and where applicable, historical technical work has been incorporated into the current geological interpretation and modelling workflows for the Lafigué Project. Historical Mineral Resource estimates reflect incremental advancements in the geological understanding of the deposit as more data was acquired, and modelling approaches evolved.

6.8 Interpretations and Conclusions

Interpretations, conclusions, and risks for Section 6 are summarised in 25 of this Report.

6.9 History

Recommendations pertaining to Section 6 are summarised in Section 26 of this Report.

6.10 References

References cited in the preparation of Section 6, are presented in Section 27 of this Report.

7. GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The majority of known gold resources within the West Africa craton are hosted by the Paleoproterozoic lithologies of the Man-Leo shield (also referred to as the Baoulé-Mossi domain). The gold deposits are typically constrained to NNE-SSW-trending Birimian greenstone belts, formed from calc-alakline or tholeiitic volcanic rocks, with metasedimentary rocks filling adjacent sub-basins (Goldfarb et al., 2017). The greenstone belts themselves most likely represent juvenile oceanic arcs accreted onto continental margin, with adjacent sediments often derived from erosion of arc rocks into back-arc basins which developed into foreland basins during basin closure (Baratoux et al., 2011). Following the emplacement of intrusive and volcanic rocks of the Birimian Supergroup, and the deposition of the clastic sediments of the Tarkwa Supergroup, the region underwent regional greenschist facies metamorphism, with some localised higher-grade metamorphism. This is particularly associated with the largest intrusive centres (John et al., 1999; White et al., 2013).

Lafigué is located towards the northern end of the Birimian-age Oumé-Fetekro greenstone belt, a N-S-trending meta-volcano-sedimentary belt comprised primarily of bimodal metavolcanics and clastic metasedimentary rocks (Figure 3-5). The belt is developed along a northeast-trending shear zone and is intruded and surrounded by a series of granite and granodiorite complexes. Other notable gold deposits developed along the Oumé-Fetekro greenstone belt include Agbaou and Bonikro, both located to the south of Lafigué.

Past field studies (cf. Mortimer, 1990; Leake, 1992; Houssou, 2013; Ouattara, 2015) indicate multiple phases of deformation for the Oumé-Fetekro greenstone belt, as summarised:

- **D1** WNW-ESE compression resulting in the formation of NNE-trending upright to isoclinal folds (F1) with a penetrative axial-planar cleavage (S1).
- **D2** WNW-ESE to NW-SE compression produced isoclinal to upright NNE- to NE-trending folds (F2), a penetrative axial-planar cleavage (S2), and moderate- to high-angle reverse shear zones.
- **D3** NW-SE transpression marking a switch from a coaxial deformation regime to a non-coaxial regime and an evolution from ductile to brittle-ductile behaviour. This deformation phase is associated with the formation of a NE-trending spaced crenulation cleavage (S3) and the dissection of the Oumé-Fetekro greenstone belt by N- to NNE-trending sinistral shear zones.
- **D4** E-W shortening occurring at high crustal levels, responsible for the development of ENE-trending (dextral) and WNW-trending (sinistral) brittle strike-slip conjugate faults. This deformation episode is also associated with the formation of localised N-trending upright folds (F4) and associated axial-planar cleavage (S4).

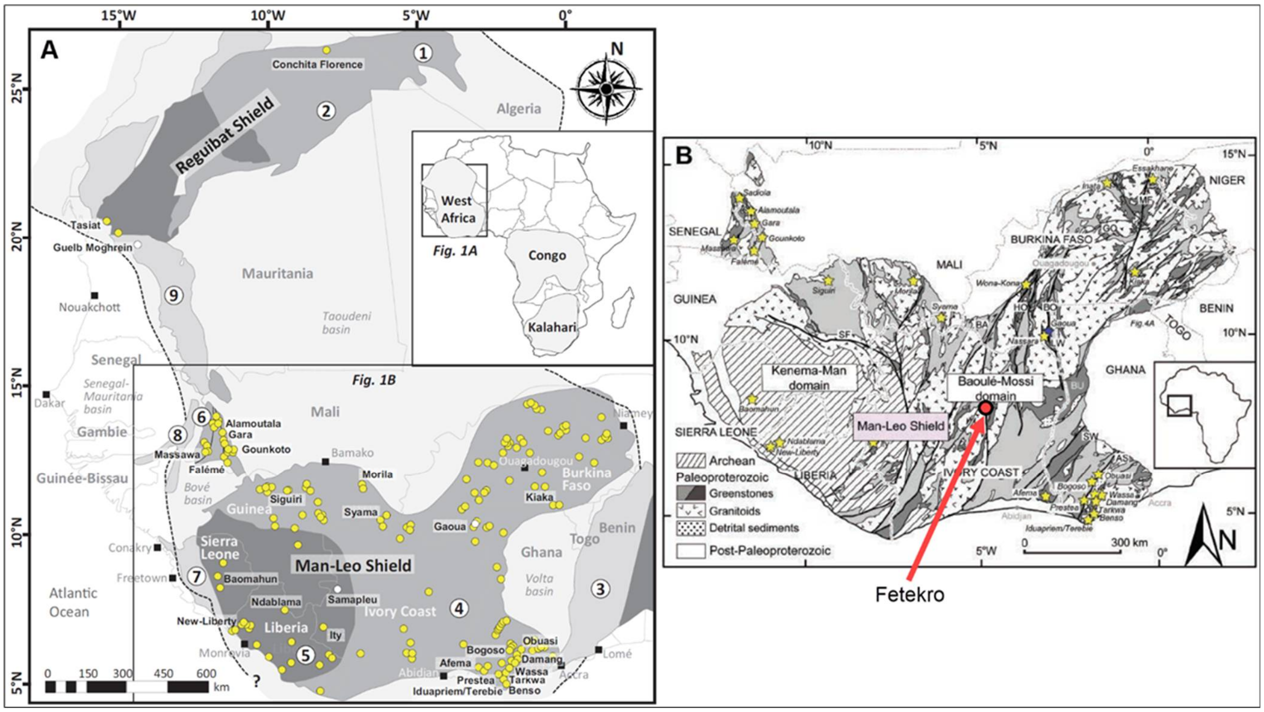


Figure 7-1: Schematic maps of the West African Craton, associated sub-domains and gold deposit locations

Notes for Figure 7-1.

- Figure (A) Schematic map of the West African craton (dashed line) showing the distribution of gold deposits (yellow circles) in the context of the various Archean, Proterozoic and Hercynian domains (1 = Eglab; 2 = Yetti; 3 = Daomeyan; 4 = Baoulé-Mossi; 5 = Kenema-Man; 6 = Kédougou-Kénébia Inlier; 7 = Rokelides; 8 = Bassarides; 9 = Mauritanides).
- Figure (B) Enlargement of the southern West African craton outlining the macro-lithological packages. Gold deposits are denoted by yellow stars.
- Modified after Goldfarb et al. (2017)

7.2 Local Geology and Mineralisation

Geologically, the Lafigué deposit is a Birimian volcanic complex predominately composed of mafic rocks (meta-gabbros/meta-norites and meta-basalts) with felsic intrusives (granodiorite or tonalite) in the western part of the prospect. This volcanic complex is affected by a transpressive deformation and is intruded by granodioritic bodies and quartz-porphyry dykes. Regional foliation varies in strike from N-S to N070° with gentle to intermediate/steep dips to the east and south (25 to 65°).

The Lafigué mineralisation is primarily controlled by an east-northeast-trending brittle-ductile thrust fault dipping 15° to 45° south-southeast. Gold mineralisation occurs as a network of Quartz-Carbonate-Tourmaline-Pyrite-Pyrrhotite veins within sheared and altered brittle-ductile deformation zones of variable thickness. The alteration assemblage comprises Biotite-Sericite-Tourmaline-Chlorite-Carbonate and various amounts of disseminated Pyrite and Pyrrhotite (up to 5%). The shear zones show a typical C-S geometry, with the CS at some angle of the S (schistosity) fabric due to shearing (Figure 7-2). The veins are emplaced in the deformation corridors both along CS and S planes. The shear foliation and most of the veins dip shallowly to the south or south-southeast. Quartz-Carbonate-Tourmaline-Pyrite-Pyrrhotite veins occur in the shear zones (Figure 7-3), show crack-seal textures or have a breccia texture with strongly altered xenoliths of sheared host rock. They appear to dilate the sheared SC foliation of the host rock at various stages of the shearing history, which demonstrate their syn-deformation emplacement (Ciancaleoni, 2018).

At Lafigué, a prominent deformation zone is typically located at the contact zone between a mafic intrusive (gabbro) and mafic volcanics, whereby the contact also occurs with a felsic intrusive at Lafigué North. The shear zones (and associated mineralisation) are better developed at or near lithological contact zones, where competency contrasts favour the localisation of brittle-ductile shearing with a permeability increase and enhanced hydrothermal fluid flow; however, these shear zones also appear in the core of massive intrusive or metavolcanic units (Figure 7-4 to Figure 7-7).

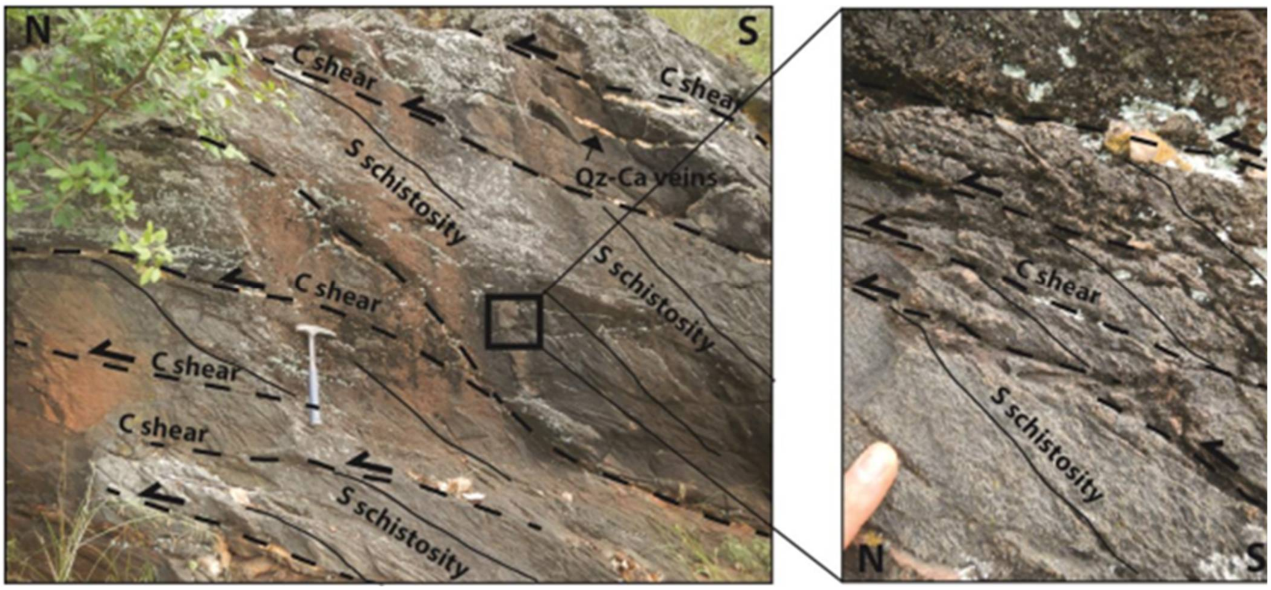


Figure 7-2: Typical Outcrop Showing the Deformation Style within the Mineralised Zone at Lafigué.

Figure 7-3: Typical Mineralisation Styles at Lafigué

Notes for Figure 7-3:

- Figure A - Sheared and altered gabbro (Bt-To-Ser±Chl ±Po ±Py) and Qz-Cb-To veins;
- Figure B - Shear (S/C) foliation with subparallel to parallel Qz-Cb veinlets;
- Figure C- lineation (Bt-To-Chl-Ser-Po); D- S/C fabric;
- Figure E - Visible gold in Qz-Cb-To vein;
- Figure F - Shear (S/C) foliation and quartz-carbonate veins relationship plotted on schmidt stereonet.

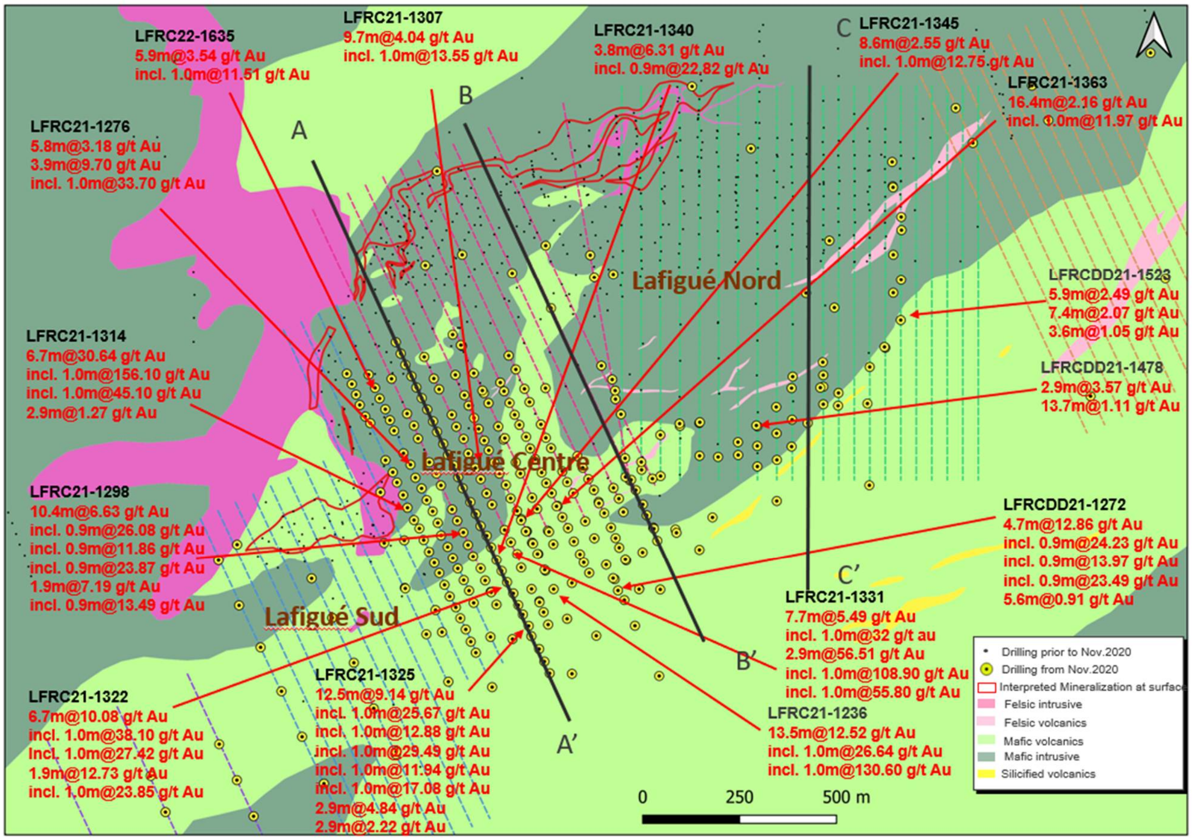


Figure 7-4: Lafigué Geology Interpretation and Mineralised Intercepts

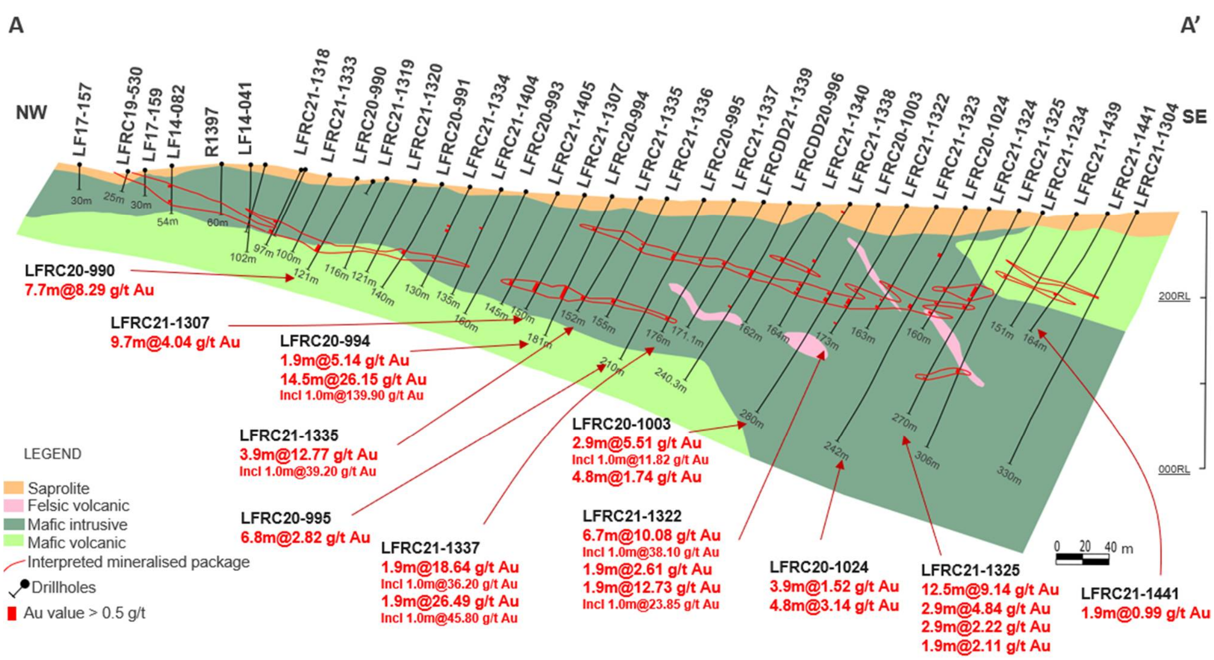


Figure 7-5: Section A-A' showing geology and key mineralised intercepts

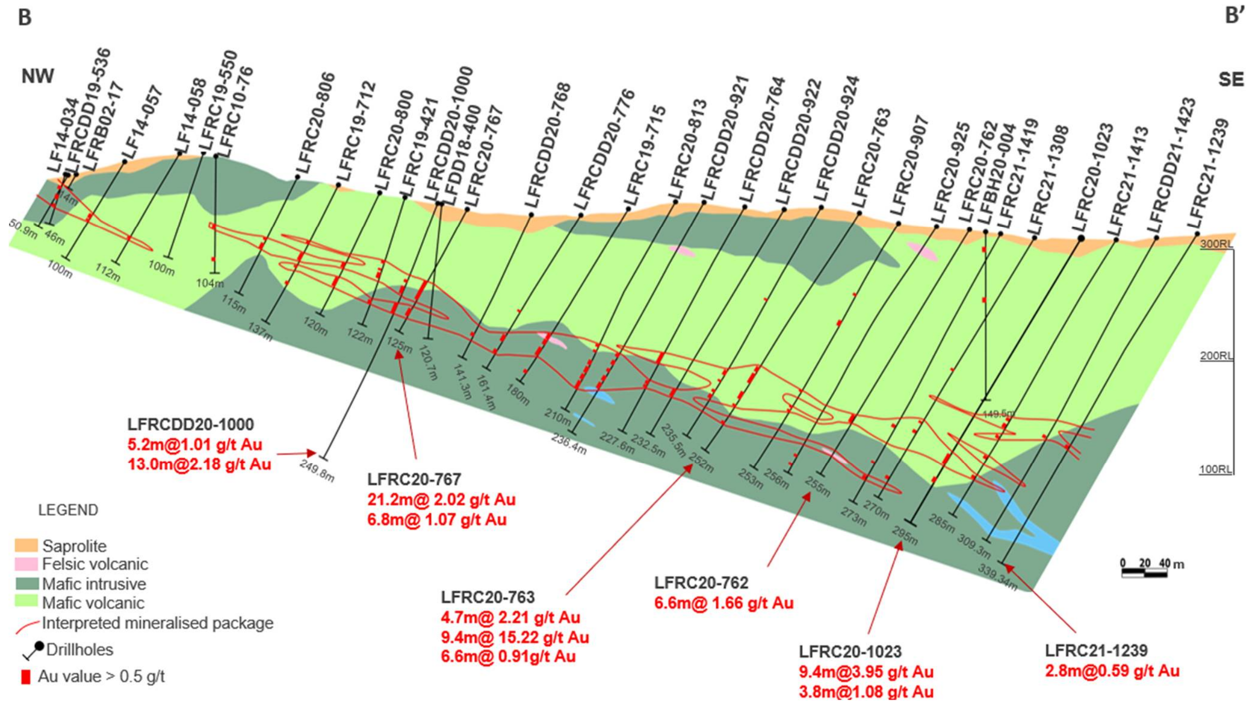


Figure 7-6: Section B-B' showing geology and key mineralised intercepts

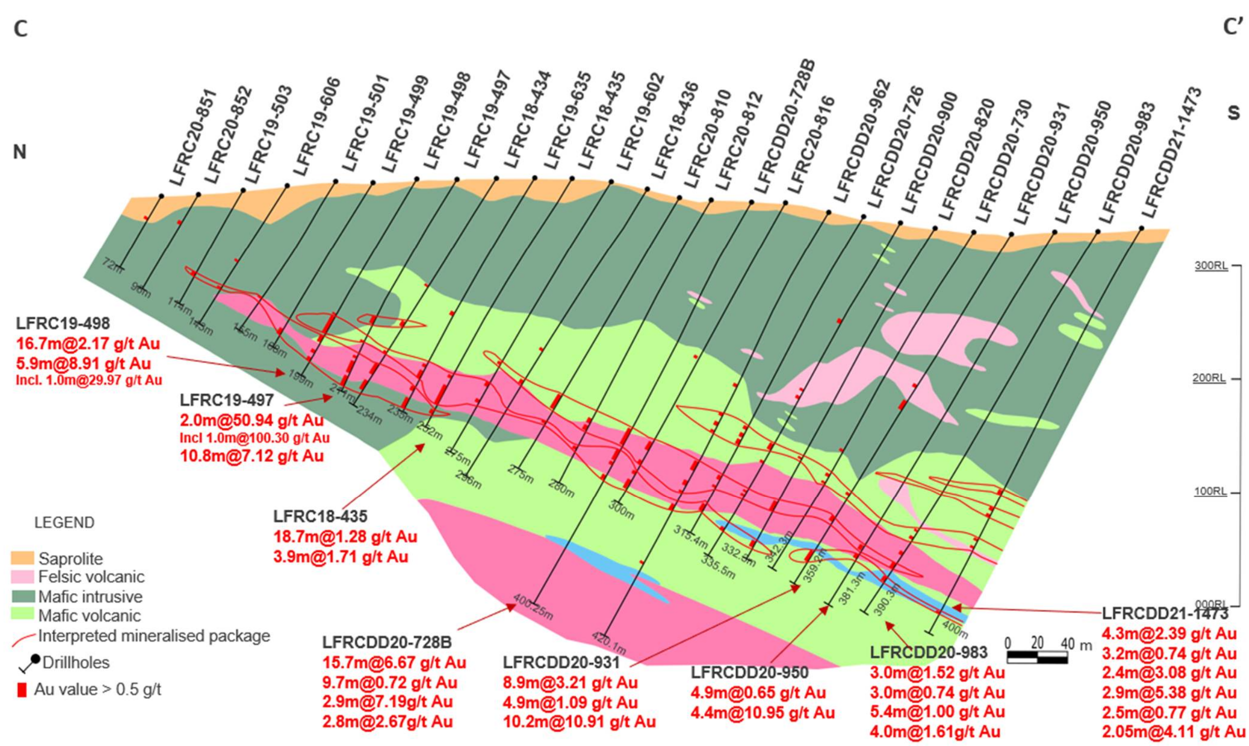


Figure 7-7: Section C-C' showing geology and key mineralised intercepts

Mineralisation is often hosted by quartz-carbonate-tourmaline-pyrite-pyrrhotite-gold veins as well as the associated biotite-tourmaline-sericite-chlorite-carbonate alteration zones (Figure 7-8 A), where these veins typically exploit the gently dipping brittle-ductile reverse shear zones. Although the quartz-tourmaline lodes commonly host mineralisation at the primary lithological contacts across the Lafigué project area, where quartz veins are barren or low grade, they can form planes of rheological contrast which have focussed auriferous fluids along vein contacts, mineralising the hanging wall or footwall rocks (Figure 7-8 B). Gold is also hosted within broader zones of altered, stacked shear zones in the hanging wall (and to a lesser degree, the footwall) of the main lithological contacts (Figure 7-8 C and D and Figure 7-9). In particular, the entire thickness of the granodiorite body in Lafigué Nord is often mineralised, including disseminated pyrite, pyrrhotite and gold, along with a similar alteration assemblage to that associated with the quartz lodes. In the broader zones of stacked shears, there is a tendency towards higher grades at the footwall contacts which likely accommodated the greatest strain and associated fluid flow.

Figure 7-8: Photos of drillcore from PE 58 Showing Different Styles of Mineralisation

Photos of drill core from Lafigué showing:

- (A) A typical high-grade quartz-tourmaline vein with associated alteration (including mineralised quartz selvages) in the hangingwall and footwall foliated metagabbro;
- (B) Finely disseminated mineralisation hosted within metabasalt footwall of an unmineralized quartz vein;
- (C) Gold mineralisation hosted as fine disseminations and within narrow quartz stringers (white circles) within the broader granodiorite package; and
- (D) Pyrite-pyrrhotite-gold associated with carbonate-sericite-biotite alteration assemblage within the sheared metagabbro unit in Lafigué Centre.

Figure 7-9: Drillcore Photograph Showing a Typical Broad Mineralized Intercept Around a Central Quartz Vein

Figure 7-9 illustrates typical broad mineralised intercepts around a central quartz vein. Shearing was focused along the lithological contact between metagabbros and metavolcanics, with the zone of mineralization directly correlating with the most intense shear fabric and associated alteration (Source: Ciancaleoni, 2018)

Mineralization at Lafigué has been interpreted to have a strike length of approximately 2 km, trending east-northeast and dipping moderately to the south-southeast. Mineralization has been intersected to depths of approximately 440 m below the surface in Lafigué North, which is approximately 700 to 900 m of down-dip extension. The continuity of the mineralization reduces towards Lafigué Centre and to the south and west. The deposit remains open at depth and, in some areas, along strike.

7.3 Comments on Section 7

The QP considers the current level of understanding of the Oumé-Fetekro Greenstone Belt, which hosts the Lafigué deposit, relatively robust. Controls on mineralization at the Lafigué deposit itself are very clear in some areas, where rheological contrasts focused shearing and attendant fluid flow, for example; and less clear in other areas (e.g. Lafigué Centre), where mineralisation shows shorter-scale continuity and less obvious direct litho-structural controls.

7.4 Interpretations and Conclusions

Interpretations, conclusions and risks pertaining to Section 7 are summarised in Section 25 of this Report.

7.5 Recommendations

Recommendations pertaining to Sections 7 are summarised in Section 26 of this Report.

7.6 References

References cited in the preparation of Section 7 are presented in Section 27 of this Report.

8. DEPOSIT TYPES

8.1 Overview

The West African Lower Proterozoic greenstone belts are often referred to as Birimian Greenstone Belts, which include a collection of Paleoproterozoic metasedimentary and metavolcanic units and associated intrusive complexes that are the dominant hosts of gold mineralization in West Africa. The Birimian Greenstone Belts host multiple world-class gold deposits situated within countries, including; Côte d'Ivoire; Burkina Faso; Ghana; Guinea; Mali; Niger, and Senegal. These deposits can broadly be classified into the following types:

- Structurally-controlled, epigenetic lode or stockwork style mineralization related to major shear zones with native gold (Poura, Burkina Faso; Kalana, Mali);
- Structurally-controlled, epigenetic lode or stockwork mineralization related to major shear zones and characterised by the inclusion of gold in the crystal structure of the sulphides, often locked in arsenopyrite (Ashanti type: Obuasi, Ghana);
- Stratiform deposits hosted in tourmalinised turbidites (Gara Deposit Loulo, Mali);
- Disseminated sulphides hosted in volcanic or plutonic rocks (Syama, Mali; Yaouré, Côte d'Ivoire; granitoid-hosted, Ayanfuri, Ghana); and,
- Paleo-placer deposits: Auriferous quartz-pebble conglomerates (Tarkwa, Ghana) and modern placers (eluvial, alluvial).

More specific to the Birimian, two major styles of gold mineralization occur, which include:

- structurally controlled quartz vein style deposits; and,
- chemical sediment hosted deposits.

The Lafigué deposit resembles a typical shear zone-hosted deposit of the West African Paleoproterozoic greenstone terrane (Man-Leo Shield) and can be associated with the low-sulphide quartz gold deposit model of Laurence J. Drew (Drew, 2003). The deposit is related to the N-S-trending Oumé-Fetekro greenstone belt, and more specifically, to a Birimian age complex of bimodal metavolcanics and meta-volcanoclastic rocks intruded by a series of felsic intrusions. Mineralization is spatially and genetically related to shearing and fluid ingress along zones of competency contrast between different lithologies. There is a further spatial relationship between some mineralization and felsic intrusive bodies. Gold is often free, occurring in quartz-carbonate-tourmaline veins or associated alteration haloes. Zones of shearing and alteration (mineralised or otherwise) can reach 10s of metres thick, pervading the hanging wall and footwall rocks away from recognised lithological contacts.

8.2 Comments on Section 8

The QP considers that the deposit type at Lafigué is well understood in the context of the wider Oumé-Fetekro greenstone belt. The deposit type targeted by exploration is well aligned with the mineralisation styles outlined at Lafigué and, as such, the QP considers the deposit type to be well understood.

8.3 Recommendations

Recommendations pertaining to Section 8 are presented in Section 26 on this Report.

8.4 References

References cited in the preparation of Section 8 are presented in Section 27 of this Report.

9. EXPLORATION

9.1 Overview

Following a strategic assessment of Endeavour's exploration tenements company-wide, the Lafigué deposit was subsequently ranked as a top priority target. Endeavour began intensive exploration on the exploration permit PR 329 in March 2017, a vertical tilt-angle derivative ('VTEM') geophysical survey was flown in 2017, which helped to better define the structural context of the permit. The survey area was flown in a northwest to southeast (N135°) direction with a traverse line spacing of 150 m, at a mean altitude of 84 m above the ground. A total of 1858-line kilometres of geophysical data was acquired during the survey over an area of 257 km^2. A structural interpretation of the VTEM survey data (Ciancaleoni, 2018) highlighted four tectonic domains (Figure 9-1), namely:

- Western tectonic domain marked by N020 sinistral shear zone and N040 regional foliation;
- Central tectonic domain, a transitional domain;
- Compressive relay domain marked by ENE trust; and
- Eastern tectonic domain, similar to western domain (sinistral N020 shear zone).

The Property is transected by sinistral N120° and dextral N060° faults, with an overall structural framework inferred to be formed in response to northwest-southeast-directed shortening during the D2 and D3 regional tectonic events under a compressive/transpressive regime of deformation.

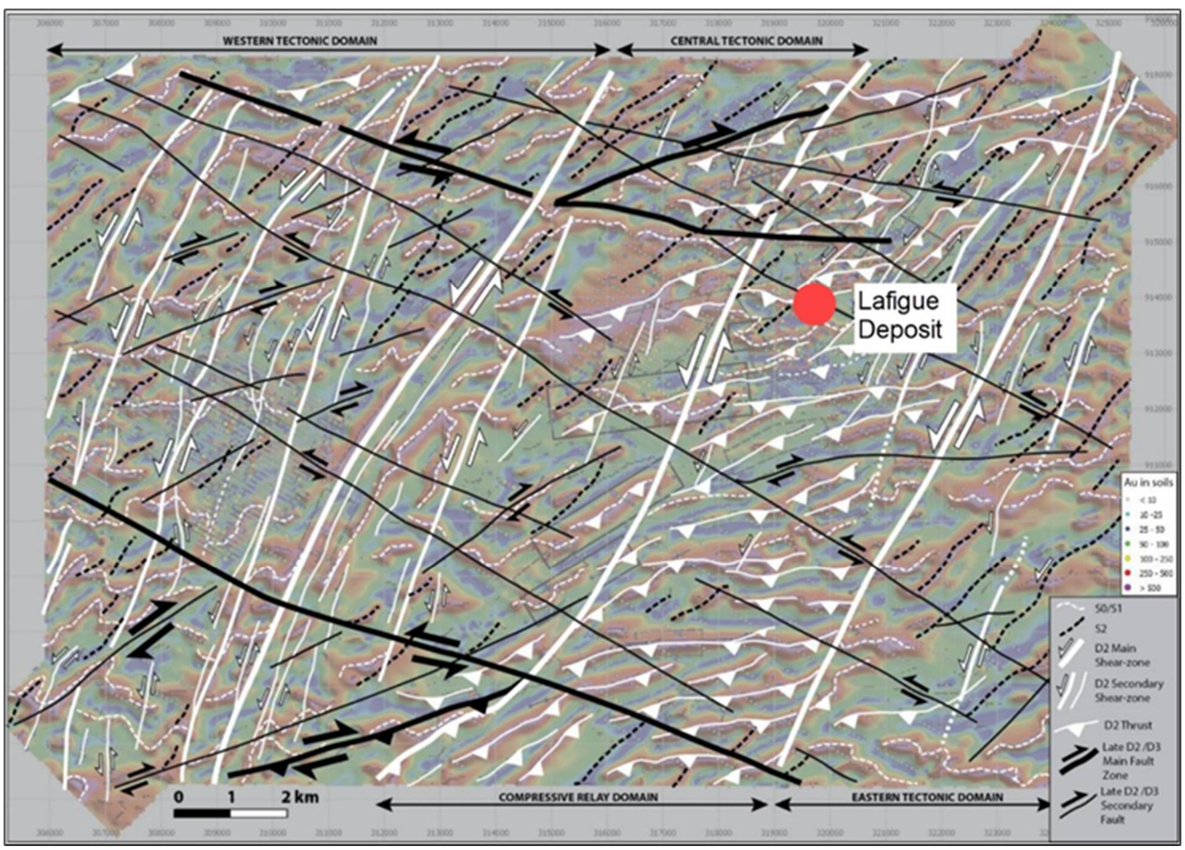


Figure 9-1: Interpreted Structural Framework Across the Fétékro Permit Area

Figure 9-1 Note: Based on Tilt-angle derivative (VTEM) image by Geotech, 2017A and overlain with interpreted structural trends (Modified after Ciancaleoni, 2018).

Approximately 20 targets across the Lafigué deposit and in the western part of the PR 329 exploration permit were identified by a gold in-soil sampling campaign (6844 samples) in 2017. These targets are denoted by the acronyms 'Target' and 'WA' in Figure 9-2, respectively. Given the subsequent focus of exploration on Lafigué, relatively little additional exploration has been carried out on the targets not immediately adjacent to the Lafigué deposit.

In 2017-2018, Induced polarisation (IP) pole-dipole and gradient surveys were carried out on Lafigué North, Target 2 and Target 5 in order to better understand the mineralised structure and to find any similar additional structures or direct extensions. An IP anomaly showing a similar signature as that observed at Lafigué was observed at Target 2; this anomaly was tested by drilling, but no significant mineralisation was found.

Also during this period, detailed mapping works were undertaken to refine the existing geological map, to classify soil geochemical anomalies in a regolith regime, i.e. to clarify their 'in situ' or 'transported' character, to update the cartographic contours of artisanally worked areas and to establish correlations between airborne radiometric survey data and geological field observations. A total of 73 grab samples were taken and analysed during this programme.

In 2019, Endeavour conducted a regional soil geochemical survey on the central part of the exploration permit, and a detailed soil geochemical survey on anomalies >50 Au ppb previously highlighted on the western part of the exploration permit. A total of 3469 soil samples were taken, helping to identify five new targets based on well-structured N10° to N25° soil anomalies several hundred metres long.

For each of the Endeavour soil sampling campaigns samples were taken at regular intervals along parallel profiles, forming a sampling grid of 400 x 100 m, with local infill at 200 x 50 m and, in some areas, at 100 x 50 m.

Samples were taken from a depth of approximately fifty centimetres below the surface and given a brief lithological description. Approximately 2 kg of material was sampled and bagged with a label stating the station and profile number written in indelible marker. The number was then written on the outside of the bag. The position of the sampling point was recorded using a GPS, with all information recorded in the database.

Blank and Duplicate samples were inserted at regular intervals, as noted below:

- Blank Samples: 1kg of sand inserted directly into a sample bag, together the sample number label. Blank samples inserted regularly every 30 samples.
- Duplicate Samples: Every nineteenth sample is duplicated, such that the twentieth sample becomes the duplicate sample.

LMCI and Endeavour have standard operating procedure manuals that document all exploration activities in detail.

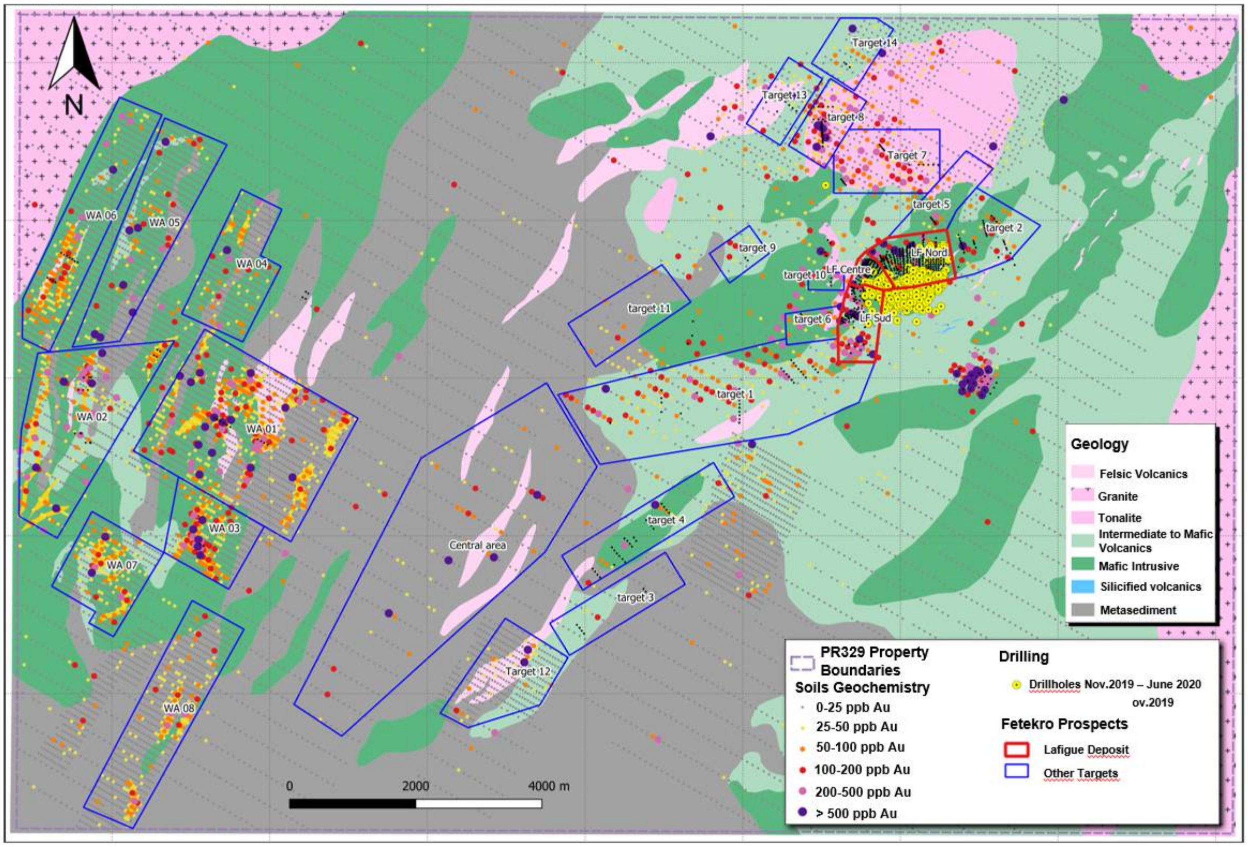


Figure 9-2: Plan Map (July 2020) Highlighting Various Exploration Targets Identified within the Fétékro Permit Area

Ten early-stage targets (eight denoted 'WA' and two denoted 'VMS' in Figure 9-3) across the PR 329 Exploration Permit, and ten early stage targets (denoted 'Target' in Figure 9-3) across the PE 58 Exploitation Permit were investigated by reconnaissance field programs during 2021 and 2022. The field work was conducted with the goal of identifying targets with potential to become additional satellite ore sources for the Lafigué Mine. Initial results have highlighted some targets warranting a further stage of more detailed exploration work (WA01, WA03, WA08, Central Area, Target 4 and Targets 9-11, all highlighted in red in Figure 9-3).

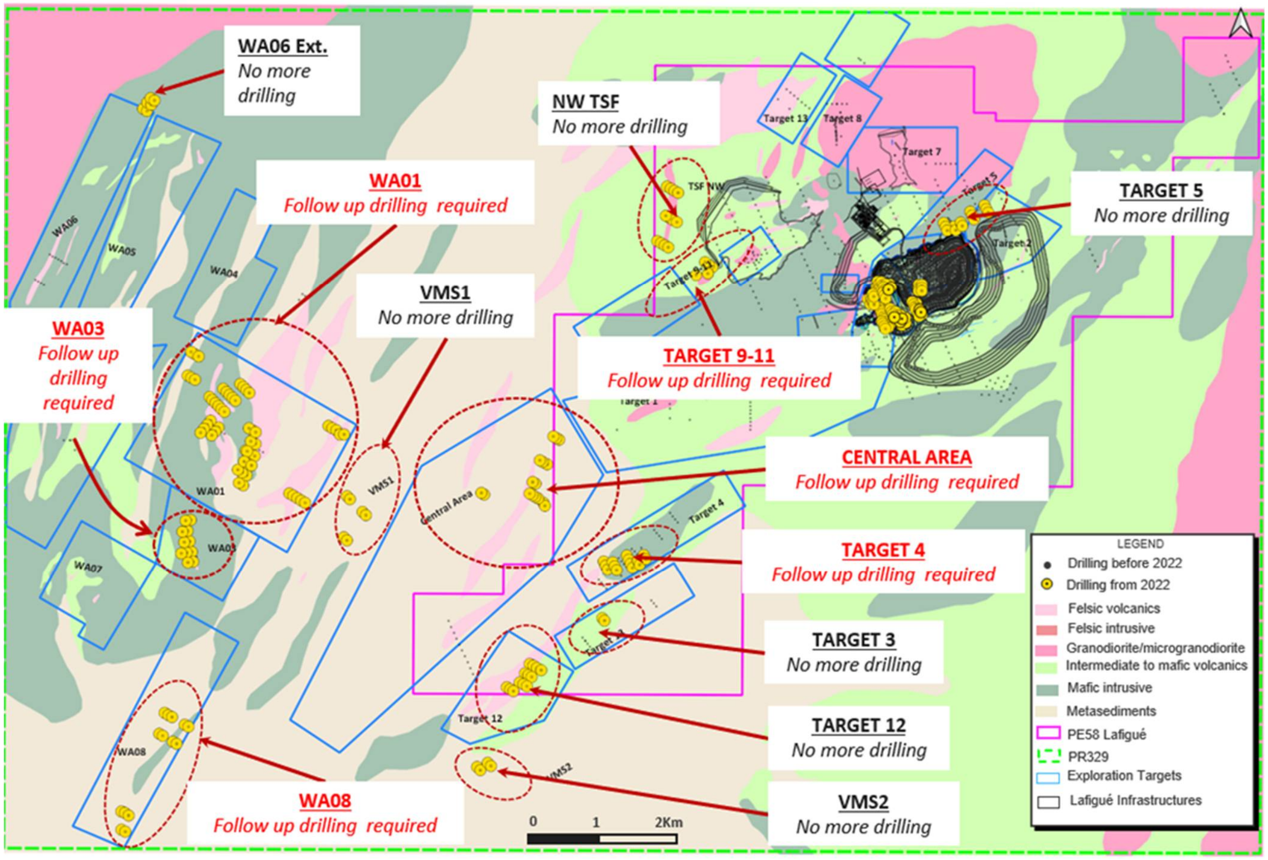


Figure 9-3: PE 58 and PR 329 Permits with Simplified Geology (June 2022), and Targets Warranting Further Exploration

Figure 9-3 note: Auger drilling is ongoing across the gold in-soil geochemical anomalies (>500 ppb) located in the Central Area, WA02, WA06, WA05, WA08 and Termitiere; 813 holes for 6678 m have been drilled to-date and full results are expected by September 2022.

9.2 Comments on Section 9

The QP considers the techniques and approaches used during the historical and ongoing exploration programmes broadly appropriate and relevant for the geology and style of mineralisation across the PE 58 Mining Permit and PR 329 Exploration Permit.

9.3 Interpretation and Conclusions

Interpretations, conclusions and risks for Section 9 are summarised in Section 25 of this Report.

9.4 Recommendations

Recommendations for Section 9 are summarised in Section 26 of this Report.

9.5 References

References cited in the preparation of Section 9, are detailed in Section 27 of this Report.

10. DRILLING

10.1 Introduction

Limited information is available regarding the drilling methods and procedures employed at Lafigué prior to LMCI involvement in 2014. Reliance upon historical drillhole data from before this period is discussed further in Section 12.

10.2 Endeavour Drilling (2017 to 2022)

The drilling programmes on the exploration permit PR 329 and in mining permit PR 58 have primarily been focused on developing the Lafigué deposit and to a lesser degree on testing priority targets in the vicinity of the Lafigué deposit and in the Western Area of the permit.

Drilling at Lafigué since 2017 has comprised of five separate campaigns aiming to delineate the full down-dip and along-strike extent of mineralization, as well as increase confidence in the geological and grade continuity through infill drilling. Additionally, some drillholes have been completed for various technical studies as part of the pre-feasibility and feasibility studies, including hydrogeological and geotechnical studies, as well as metallurgical testwork. Together, these total 17 RC, RC-DD and DD drillholes for 3217 m. A program of sterilisation drilling totalling 216 holes for 26 754 m was also conducted in 2021/2022 to test areas where mining infrastructure was planned. Only the Tailings Storage Facility (TSF) area showed significant mineralization; however, the potential of the area was considered too low to justify relocation of the planned infrastructure. Table 10-1 summarises all of the drilling completed under Endeavour ownership from 2017 until June 2022. The Lafigué drilling is also illustrated in Figure 10-1, with drilling across the wider PR 329 Exploration Permit shown in Figure 10-2.

Table 10-1: Summary of drilling completed between 2017 and June 2022 in PE 58 and PR 329

Period	Type	Number	Metres	Drilling Contractor
2017	DDH	17	2197	FORACO
	RC	179	12 464	FTE
2018	DDH	21	3861	FORACO- GEODRILL
	RC	105	14 647	GEODRILL
	RC-DD	8	2662	GEODRILL
2019	DDH	15	2543	FORACO- GEODRILL
	RC	228	37 633	GEODRILL
	RC-DD	27	7804	GEODRILL
2020	RC	169	35 941	GEODRILL
	RC-DD	130	41 556	FORACO- GEODRILL
2021	RC	416	62 572	GEODRILL
	DD	5	1468	GEODRILL
	RC-DD	61	19 844	GEODRILL

Table 10-1: Summary of drilling completed between 2017 and June 2022 in PE 58 and PR 329

Period	Type	Number	Metres	Drilling Contractor
2022	RC	242	27 162	GEODRILL
	RC-DD	7	1310	GEODRILL
Total		1630	273 664	

Table 10-1 notes:

- DD = Diamond core drilling; RC = Reverse Circulation drilling; RC-DD = Reverse circulation with a diamond core tail
- Includes drilling across all permit areas, and for all purposes, including sterilisation, hydrology, geotechnical and geometallurgical testwork purposes

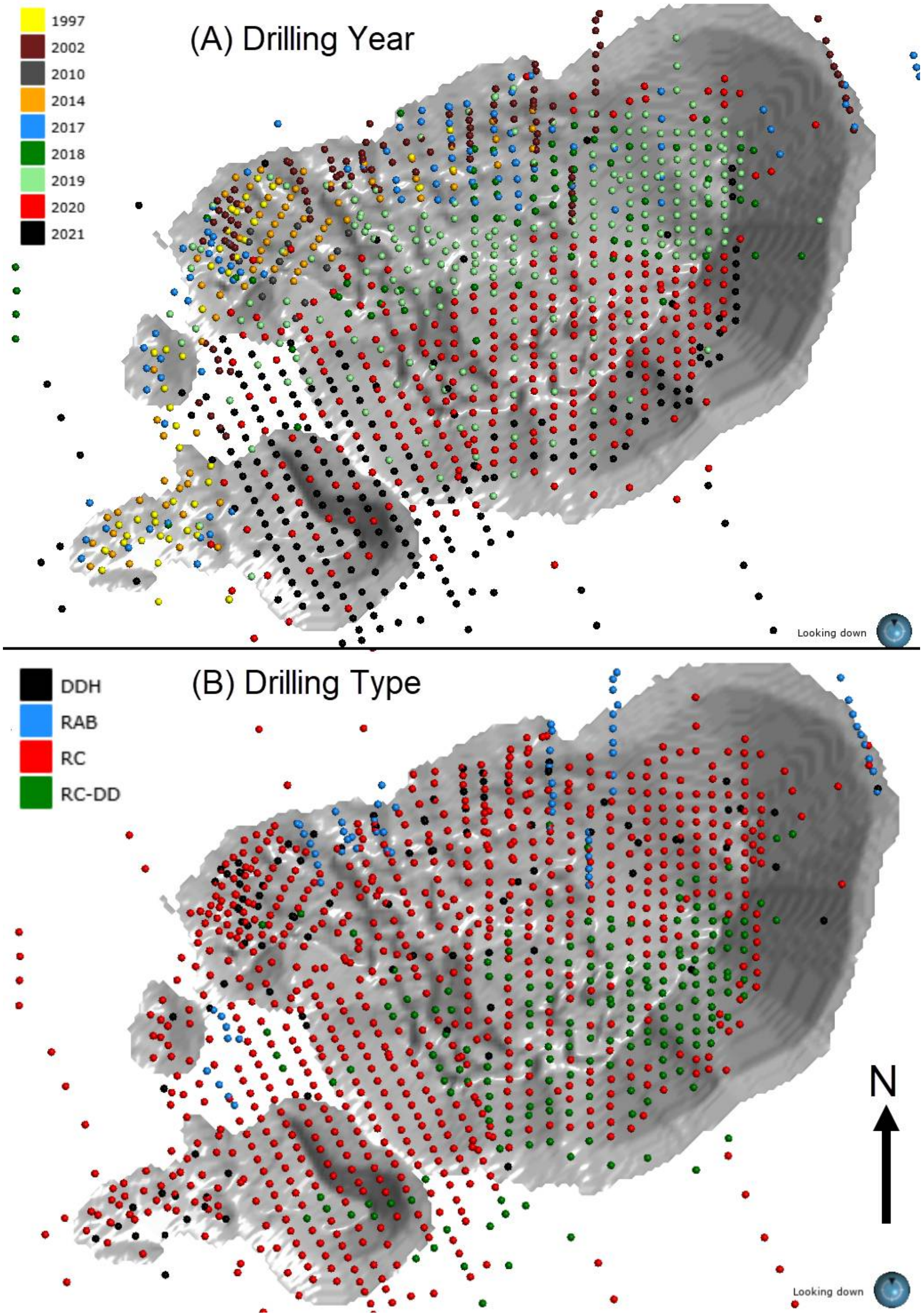


Figure 10-1: Plan Views of Drillhole Collars in the Context of the 2022 Optimised Resource Shell for the Lafigué Deposit

Figure 10-1 notes: Drillhole collars coloured by, (A) - drilling campaign (year); (B) - type of drilling method used

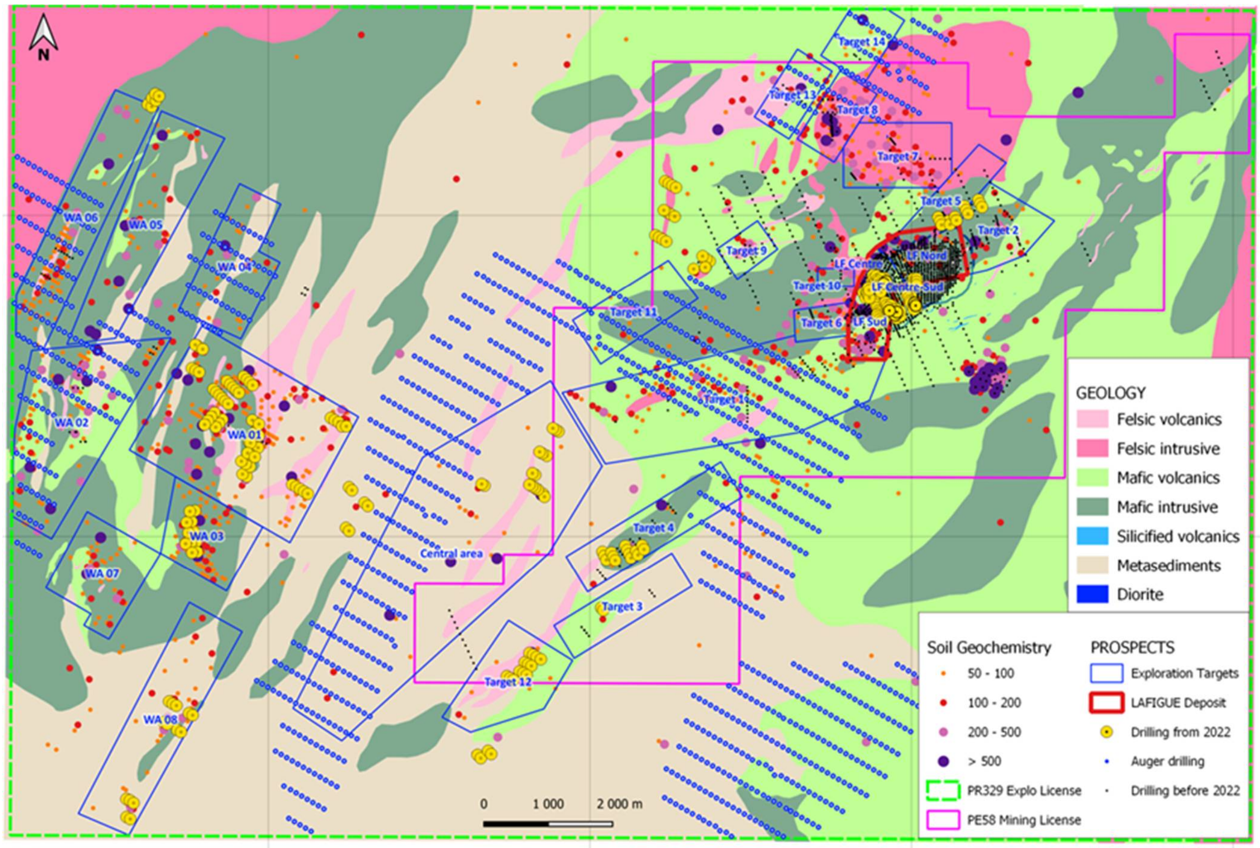


Figure 10-2: Lafigué Drilling Plan Map (including Auger drilling) with Exploration Targets

10.3 Drilling Methods

Drilling conducted during Endeavour's ownership of the Project has been carried out under the supervision of technically qualified personnel applying industry standard approaches. The drilling contractors used for each program are detailed in Table 10-2, with the current contractor being GEODRILL.

Figure 10-3: RC Drill Rig (Hole ID: LFRC21-1405) and Associated Sampling Setup

Drilling is carried out in two 12-hour shifts per drill rig, operating 6 days per week. A geologist supervises each drillhole, with geological technicians and other associated workers allocated to each drill rig for sampling purposes.

The paper logs for the majority of drillholes completed prior to 2017 have been located, reviewed, and digitised by Endeavour.

10.4 Core Recovery

Core recovery has been recorded for all diamond core drilling at Lafigué since the 2010 drilling campaign. Core recovery was measured based on the length of core recovered relative to the length of each core run, with a global average of 98.3% recovery across the eight drilling campaigns since 2010 (Figure 10-4). Histograms and associated summary statistics for core recovery, broken down by drilling campaign are provided in the appendices of the May 2022 SRK MRE Report.

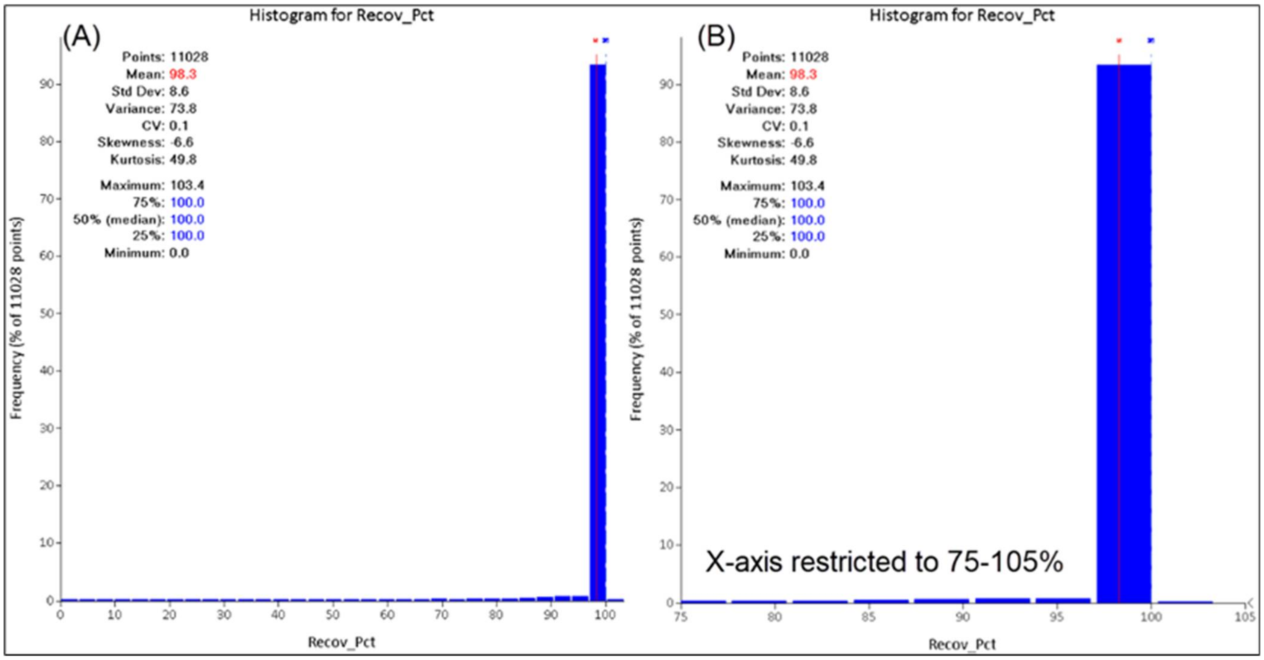


Figure 10-4: Histograms showing Core Recovery for drilling since 2010

Figure 10-4 notes:

- Histograms A shows diamond core recovery (%) for the global drillhole database (2010 to 2022).
- Histogram (B) shows the same data with the X-axis (% recovery) restricted to 75-105%. Stated statistics exclude seven samples with recorded recoveries >105%.

10.5 Drillhole Surveying

10.5.1 Collar surveys

For drillholes completed between 2017 and 2019, collar surveys were conducted by the sub-contractor Société Nationale de Topographie ("SNT") using a differential GPS. The 2020 and Q1 2021 drillhole collars were surveyed by the subcontractor Cabinet Kouamelan using a differential GPS, with Cabinet CGE-Kouroukan then surveying all drillhole collars since Q2 2021 by the same method.

With the exception of some very minor corrections in the elevation of some drillhole collars, which were set to the topographic surface for the purposes of geological modelling for the current MRE, no material issues were identified in the collar survey information.

10.5.2 Downhole surveys

Downhole surveys were completed in each drillhole using geographic north as a reference azimuth. Table 10-2 summarises the downhole survey tools used by each drilling contractor.

To better control downhole deviation in the deeper drillholes, stabiliser rods have been used since November 2019. The drilling grid azimuth has varied by campaign and area; drillholes in the Southern Area are oriented towards N300°; in the Central Area, towards N300°, N0° and N335°; and in the North Area, towards N180°. The holes are drilled at dips of -60° and -50°.

Table 10-2: Summary of Downhole Survey Tools Used, Split by Drilling Campaign/Contractor

Year	Drilling Contractor	Downhole Survey Instrument	Comments
2017	FTE	Gyro	
	FORACO	Reflex-EZ track	
	GEODRILL	Reflex-EZ track	
2018	FORACO	Reflex-EZ track	
	GEODRILL	Reflex-EZ track + EZ-Gyro	
2019	FORACO	Reflex-EZ track	
	GEODRILL	EZ-Gyro + SPRINT Gyro	TN14 AZI Aligner
2020	FORACO	Reflex-EZ track	
2021	GEODRILL	EZ-Gyro + SPRINT Gyro	TN14 AZI Aligner
2022	GEODRILL	EZ-Gyro + SPRINT Gyro	TN14 AZI Aligner

10.6 Logging and Photography

Diamond core drillholes were geotechnically logged and photographed at the drilling site, along with the marking up of an orientation line, where competent, oriented core has been recovered. Drillcore and RC chips were then transported to an LMCI sampling facility where detailed geological, structural and weathering logging was conducted. Each drillcore log includes:

- Lithology:
  - Rock code(s)
  - Sulphide intensity
  - Carbonate intensity
- Alteration:
  - Alteration mineralogy and intensity
- Oxidation:
  - Oxide, Transition ("oxide-sulphide"), Sulphide
- Weathering:
  - Weathering code (LATR, MTLZ, SAPR, SAPRK, OVBD, NRCV, BDRK)
- Structure:
  - Structure code (qualitative observation)

10.7 Drillhole Orientation Relative to Mineralization

The majority of DD and RC holes at Lafigué were drilled on a regular grid, dipping at 50° or 60° towards 000° or 335°. Mineralization typically dips at approximately 20° to 40° towards the S-SSE, resulting in drilling intersection angles of 90° to 110°. Overall, it is considered that drillhole orientations relative to mineralization are suitable to support the 2022 Mineral Resource estimate.

10.8 Drillhole Quantity and Spacing

The majority of DD and RC holes at Lafigué have been completed on a 20 to 40 m by 50 m grid, with some areas of closer drilling towards the up-dip portions of the deposit and wider spaced drillholes in down-dip areas (Figure 10-1).

The Lafigué drillhole database (PE 58) contains a total of 1189 DD, DD-RC and RC exploration holes, the majority of which support the main area modelled in support of the 2022 MRE. Although some areas, particularly in the western and down-dip portions of the deposit, would benefit from further drilling to increase confidence in geological and grade continuity, it is considered that the current drilling database is sufficient to support the 2022 Mineral Resource estimate.

10.9 Comments on Section 10

It is considered that the drilling procedures employed by LMCI/Endeavour, including collar and downhole surveys, logging, photography and core cutting typically conform to industry best practice. Where these details are not fully documented for historical drilling (i.e., pre-2010), this drilling comprises a relatively small component of the overall database supporting the MRE and is often supported by relatively close spaced younger drilling.

10.10 Interpretations and Conclusions

Interpretations, conclusions and risks for Section 10, are presented in Section 25 of this Report.

10.11 Recommendations

Recommendations for Section 10 are presented in Section 26 of this Report.

10.12 References

References cited in the preparation of Section 10 are presented in Section 27.

11. SAMPLE PREPARATION, ANALYSIS AND SECURITY

11.1 Introduction

Sample preparation, analysis and security for the Lafigué Project is currently under the supervision of Endeavour geologists, with LMCI responsible between 2013 and 2016. The following processes and procedures relate to the drilling and sampling campaigns managed by Endeavour since 2017, with similar protocols being followed by LMCI between 2013 and 2016. Information pertaining to historical drilling and sampling prior to 2013 is limited to some QAQC results for the 2010 drilling campaign only, however much of the historical drilling (completed prior to 2010) has been followed up with close-spaced drilling during the 2014 and 2017 drilling campaigns, run by LMCI and Endeavour, respectively.

11.2 Sampling Methods

11.2.1 RC Sampling

Reverse circulation ('RC') samples are collected in 1 m intervals in bulk bags directly from the cyclone discharge (Figure 11-1 A). Samples are riffle split into a labelled sample bag, producing a representative 2 to 4 kg sample split with a matching sample tag included in each bag. A duplicate 2 to 4 kg sample is retained for reference, alongside a small quantity of representative chips for geological logging purposes. The riffle splitters, sample tubs and other working surfaces are cleaned with compressed air between each sample. The sample rejects are bagged and either remain at the drill pad or are transported to the sample management facility. The riffle splitting and sampling methodologies are summarised in the flow charts in Figure 11-2 and Figure 11-3.

Figure 11-1: Photographs Showing Sampling Procedures at an RC Drill Rig

Figure 11-1: notes: Figure (A) - Sample collection from the RC rig cyclone; and Figure (B) - Riffle splitters used to produce a 2 to 4 kg sample split

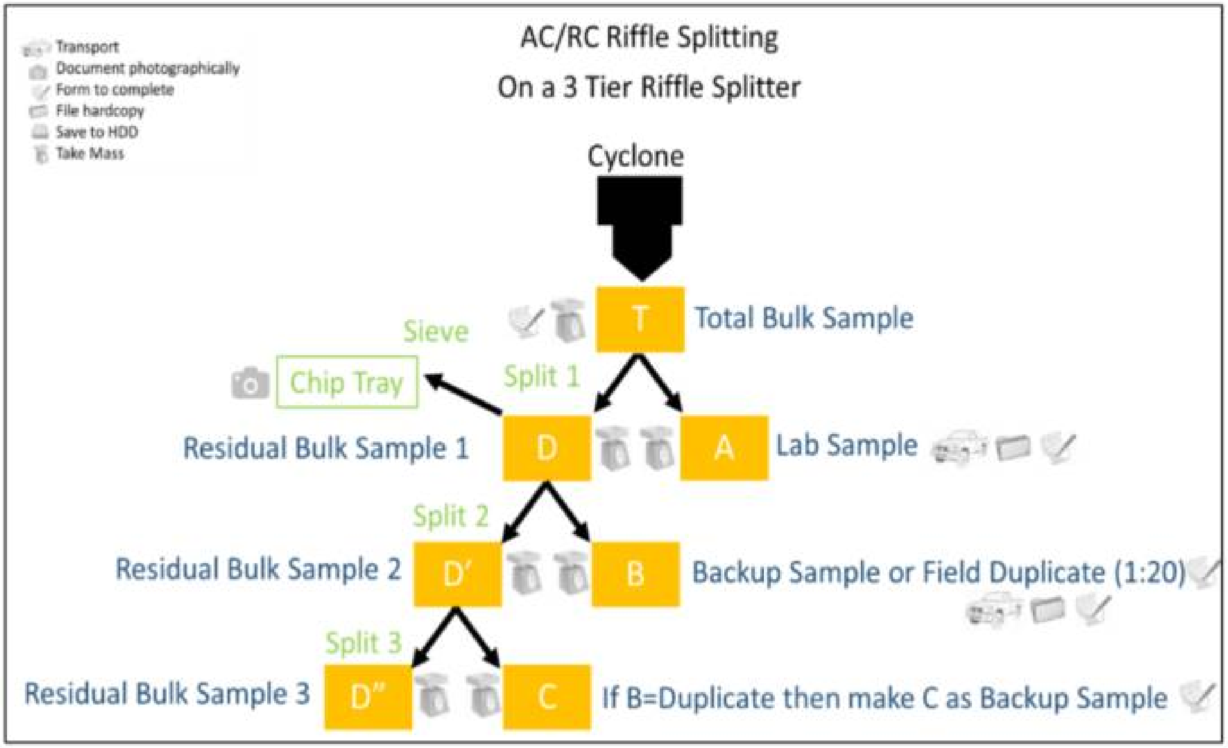


Figure 11-2: Sample Splitting Methodology Using a Riffle Splitter

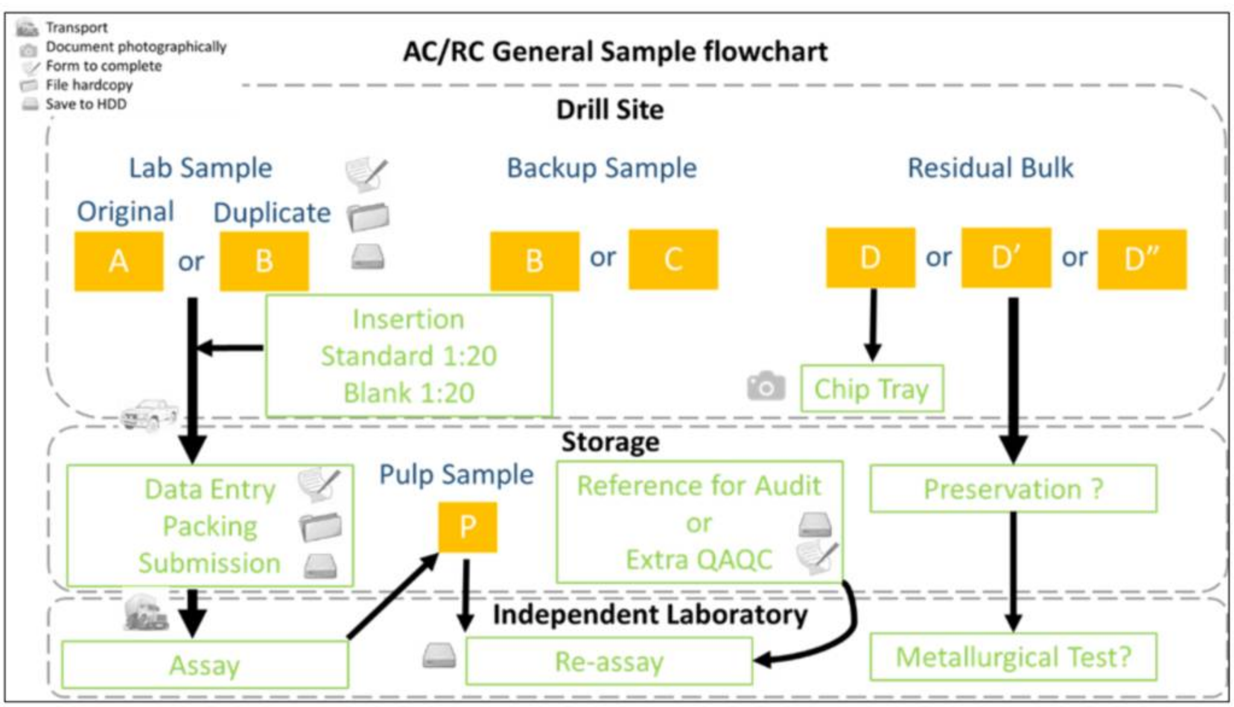


Figure 11-3: RC Sampling Flowchart

Polyweave bags containing approximately 30 samples are regularly transported to the Bureau Veritas laboratory in Abidjan, Côte d'Ivoire. The Bureau Veritas laboratory is not currently ISO17025 accredited. The laboratory does, however, work under the accreditation of the global Bureau Veritas group of laboratories including Australia and Canada and is covered by the group's ISO9001, ISO14001, ISO18001 and IFIA certificates.

11.2.2 Diamond Drilling Sampling

Drill core is placed in steel or timber core boxes, each marked with the borehole ID, and start and end depths for the corresponding core. Orientation lines are drawn on competent lengths of drill core immediately, and the core is geotechnically logged and photographed whilst still at the drill site. Core boxes are transported to the LMCI sampling facility where the core is geologically logged, and sampling intervals are marked. Drill core is cut along its longitudinal axis, with half core samples selected from the right-hand side of each interval (looking down hole). Samples are tagged, bagged and transported to the Bureau Veritas laboratory in Abidjan. The remaining half of each core is retained for reference. Figure 11-4 shows the typical sampling procedures flowchart for DD drillholes.

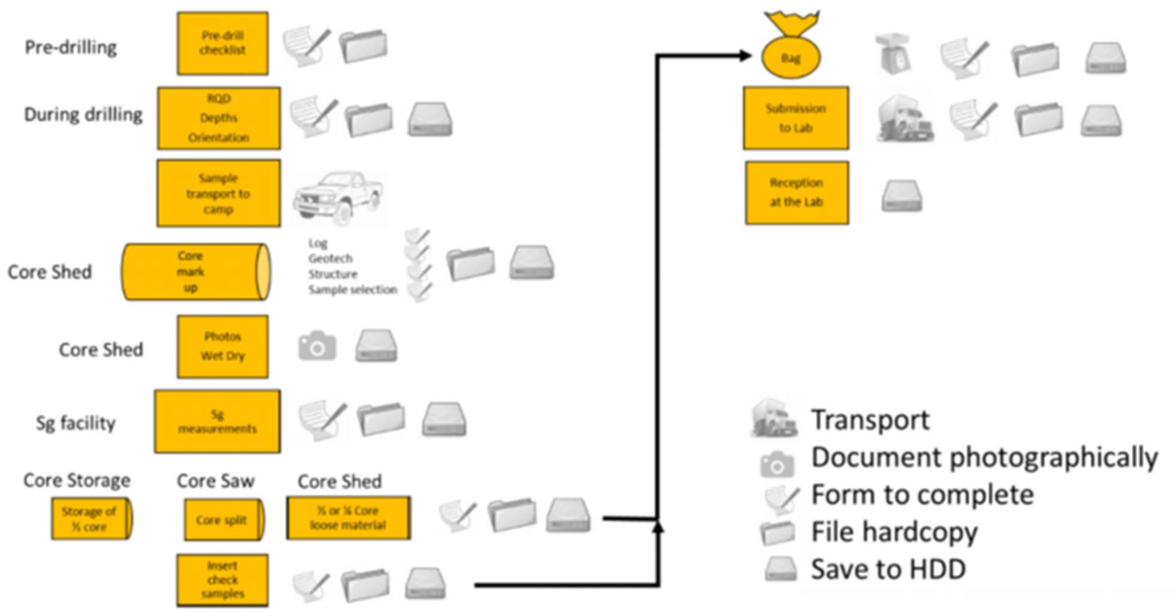


Figure 11-4: Diamond Core Drilling Sampling Flowchart

11.2.3 Sample Submission Methodology

Both RC and DD sample submissions to the Bureau Veritas laboratory in Abidjan are accompanied by a sample submission form detailing the sample numbers. Bureau Veritas staff cross-referenced the samples received with the sample submission forms to ensure all samples are received. Samples are logged into the Laboratory Information Management System ('LIMS').

11.3 Sample Security and Chain of Custody

All diamond drill core and RC drill samples are transported to Endeavour's secure (walled and lockable) Fetekro project sample management facility. QA/QC samples are inserted into the sample sequence for all diamond drill core and RC drill samples. The batches of samples are placed in sealed and numbered poly-weave or plastic bags for transport. The sample shipments are verified by laboratory personnel at the Fetekro Projects sample management facility with Endeavour personnel during loading on the laboratory truck.

A verification document is signed by both parties before departure. Upon reception at the laboratory, sample preparation personnel verify the shipment samples. All aspects of the sample collection and dispatch is conducted by Endeavour personnel, or under the supervision of Endeavour personnel.

11.4 Sample Storage

The Fetekro project has several enclosed buildings and fenced core yards which are locked and secured by security personnel. These facilities are used for sample management and storage management. Destructive (RC, AC, ARC) reference samples (Temoin) collected at the drill site as secondary samples to those submitted to the laboratory are stored at these secure sample management and storage management sites. Diamond drill core is stored at the same sites.

Course reject samples generated at the laboratory are stored at the laboratory until the QAQC results have been signed off, thereafter, they are destroyed. Pulps generated by the laboratory which are not used for first pass Fire Assay are stored at the laboratory for a short time before being transferred to the secure sample management and storage management sites.

11.5 Analytical and Test Laboratories

The primary independent laboratory used for assaying of samples collected at Lafigué is Bureau Veritas in Abidjan where fire assay analysis represents the majority of tests undertaken. The laboratory is currently certified to the following standards:

- ISO 9001 (Certified 2020-01-25; Certificate Registration No - 44 100 16014)
- ISO 14001 (Certified 2020-01-25; Certificate Registration No - 44 104 16014); and
- OHSAS 18001 (Certified 2020-01-25; Certificate Registration No - 4 116 160145)

11.6 Sample Preparation and Analysis

11.6.1 Sample Preparation

The sample preparation procedures, applicable to both diamond drill core and RC samples, undertaken at the Bureau Veritas Laboratory in Abidjan included:

- oven drying at (105 to 110)°C;
- crushing using a jaw crusher such that 75% passes a 2 mm diameter mesh;
- sub-sampling with a riffle splitter;
- pulverisation of approximately 0.5 kg with an LM2 pulveriser such that 90% passes a 75 µm mesh; and,
- homogenisation of a 250 g pulp split for transfer to the fire assay circuit.

11.6.2 Sample Analysis

All samples taken since 2017 were analysed by fire assay with an atomic absorption finish (BV code FA450) using a nominal 50 g charge. Samples returning a grade greater than 100 g/t Au were reanalysed by fire assay with a gravimetric finish (BV code FA550 or FAGRA01).

11.7 Density Determinations

The density sample database for Lafigué includes a total of 3312 measurements taken between 2014 and 2022. Density determinations were carried out using drill core samples representing the full range of lithologies and weathering intensities present at the Project. Competent sections of core (160 g to 1,000 g in mass) were cut and dried in the sun for 2 days prior to measurements being taken.

Sample densities were measured on-site using the Archimedes principle of first weighing the sample dry, and then submerged in water within a wax/plastic coating. Moisture content was not measured and is assumed to be negligible following drying of the sample. The following equation was used to generate specific gravity, which at room temperature correlates to density:

Equation 11-1: $$\boldsymbol{Archimedes\ SG} = \frac{\textbf{Weight of sample (g)}}{\boldsymbol{Weight\ in\ air\ (g) - weight\ in\ water\ (g)}}$$

The average density values, split by lithology and weathering intensity are shown in Table 11-1. Note these do not exactly correlate with the density values stated in Section 14.10, where these values are averages applied to simplified lithology groupings for the purposes of the tonnage estimate as part of the MRE.

Table 11-1: Average Densities, Split by Lithology and Weathering

Rocktype	Average of SG	Weathering	Average of SG
CHRT	2.70	LATR	2.02
DIOT	2.77	SAPR	1.76
DYKE	2.84	SPRK	2.54
IFEL	2.73	BDRK	2.80
IMAF	2.86		
MARK	2.81		
MVCB	2.83		
QZON	2.72		
QZTM	2.82		
QZVN	2.70		
VCSD	2.83		
VFEL	2.72		
VMAF	2.84		
Average All	**2.80**	**Average All**	**2.79**

11.8 Quality Assurance and Quality Control

11.8.1 Introduction and Summary

Quality Assurance/Quality Control (QAQC) sampling programmes are typically designed to identify and assess contamination or bias in the analytical results and allow analytical precision and accuracy to be quantified, providing confidence in the underlying sample data used for the purposes of estimating Mineral Resources.

This section comprises of a review of the QAQC sample analyses for the 2017 to 2022 Endeavour drilling campaigns as well as the 2010 and 2014 drilling campaigns prior to LMCI/Endeavour ownership. No QAQC sample results are available for earlier drilling (1997 and 2002 drilling campaigns) and as such, assay data from these drilling campaigns present a risk in terms of accuracy and precision of the associated assay grades. Given the relatively small proportion of drilling (<8% of total RC + DD drillholes used for the MRE), it is considered reasonable to include these data in the 2022 MRE.

A summary of QAQC sample insertion rates for drilling between 2010 and 2022 is provided in Table 11-2 following.

Table 11-2: Summary of QAQC Samples Inserted During the 2010 to 2022 Lafigué Drilling Campaigns

Sample Type	Drilling Campaign								Total	%
	2010	2014	2017	2018	2019	2020	2021	2022*		
Regular samples	1661	6497	6164	13 005	38 919	73 585	81 288	22 073	243 192	100%
Blank	12	93	92	61	123	209	424	215	1 229	0.51%
Blank (coarse)	12	428	191	535	1 683	3 212	3356	812	10 229	4.21%
CRM Combined	36	435	360	744	2250	2835	4730	1282	12 672	5.21%
G300-8			115	5					120	0.05%
G302-3								12	12	
G307-2	2	144							146	0.06%
G310-6				244					244	0.10%
G310-8			115						115	0.05%
G910-10								12	12	
G311-2	2	145							147	0.06%
G316-2			3	167					170	0.07%
G318-10								52	52	
G910-8	4	146	10	5					165	0.07%
G913-3							160	402	562	0.23%
G913-9			117	236	749	1439	1173		2275	0.94%
G914-2				59	750	1424	1718	391	4342	1.79%
G915-6				28	751	1411	1679	413	4282	1.76%
G998-8	23								23	0.01%
Std-UNKN	5								5	0.00%
Field duplicates	-	-	394	796	2511	4312	4794	1284	14 091	5.79%
Pulp duplicates	47	509	-	-	-	-	-	-	556	0.23%
Total QAQC Samples	**107**	**1465**	**1037**	**2136**	**6567**	**10 568**	**13 304**	**3593**	**38 777**	**15.95%**

Table 11-2 notes: *Drilling completed up until May 2022

11.8.2 Blank Samples

The insertion of blanks is intended to identify if there has been contamination during the sample preparation process. Additionally blank samples can capture contamination during Fusion and Cupellation, digestion and on AAS probes. Two types of blanks have been used at Lafigué:

- coarse crush blank – granite sourced from a quarry near Abidjan; and,
- fine blank – a fine fluvial sand sourced from a river in Abidjan.

Both blank materials are reported to have been tested at multiple laboratories within Côte d'Ivoire.

During the 2010 to 2022 drilling campaigns, coarse and fine blank samples were inserted at overall insertion rates of 0.5% and 4.2%, respectively, broadly in-line with Endeavour's policy of at least one blank sample insertion per 30 regular submissions. Endeavour considered a grade greater than 5x the limit of detection (i, e. >0.05 ppm) a failure. Overall, the performance of the blank samples was excellent between 2010 and 2022. Although approximately 10 to 15% of blanks samples returned Au grades greater than the detection limit (>0.005 ppm), none of these were greater than 0.05 ppm and therefore considered a failure. An example blank control plot is shown in Figure 11-5, with plots from each drilling campaign between 2010 and 2022, split by drilling type, presented in Appendix A.

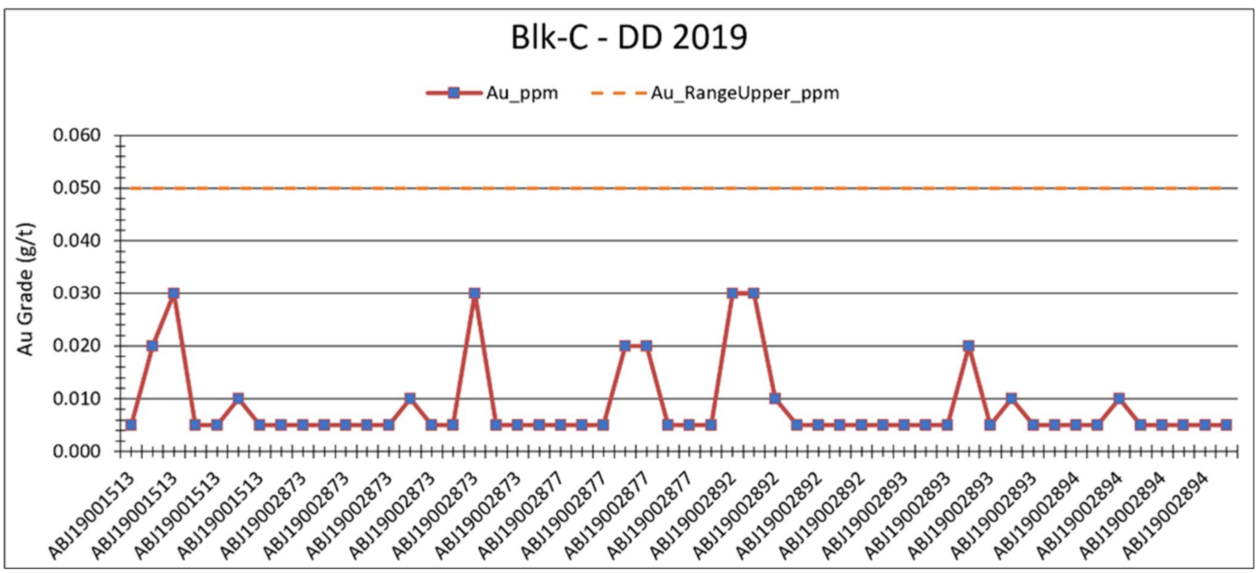


Figure 11-5: Example Coarse Blank Control Chart Showing Blank Sample Grades for the 2019 DD Campaign

11.8.3 Duplicate Samples

11.8.3.1 Background

The precision of sampling and analytical results can be measured by re-analysing a portion of the same sample using the same assay methodology. The variance between the original and duplicate result is a measure of the precision.

Precision is affected by mineralogical factors such as grain size and distribution and inconsistencies in the sample preparation and analysis processes. There are a number of different duplicate sample types which can be used to determine the precision for the sampling process, sample preparation and analyses. Field duplicates assess the variability of two samples taken across the same interval, indicating the overall repeatability of the assayed results. Field duplicates can also help detect sample number mix-ups and assess the natural local-scale grade variation or nugget effect.

11.8.3.2 2010 and 2014 Pulp Duplicate Evaluation

A relatively small number of pulp duplicates were inserted into the sample stream during the 2010 and 2014 drilling campaigns in order to assess laboratory precision. In total, 557 pulp/laboratory duplicate samples were submitted for analysis during the 2010 and 2014 drilling campaigns, equating to insertion rates of approximately 0.2% for pulp duplicates.

The 2014 DD laboratory duplicate results show a relatively poor degree of correspondence (R2 = 0.41) and there is little documentation detailing potential sources of imprecision in the sampling. Given the relatively small number of 2014 DD drillholes supporting the MRE (24 drillholes, or <2% of the total supporting drillholes), this is not considered to be a material issue.

11.8.3.3 2017 to 2022 – Endeavour Field Duplicate Evaluation:

In total, 14 091 field duplicate samples were submitted for analysis during the 2017 to 2022 drilling campaigns, equating to an average insertion rate of approximately 5.8%.

Endeavour Exploration uses tables of calculations and charts to evaluate duplicates. A set of calculations has been integrated into the standard Endeavour QA/QC report based on procedures used by some assay laboratories to evaluate internal duplicate and repeat samples. A tolerance value can be calculated for each individual duplicate pair based on the mean grade of the pair, the lower limit of detection for the analytical method used, and the method precision, as determined by the laboratory. The relative difference (RD) is then calculated for each individual pair. If the RD result exceeds the calculated tolerance result the duplicate pair is considered to have failed. Extracts of the tables for 2021 (with failures) and 2022 (no failures in 2022) are presented in Table 11-3 and Table 11-4.

Table 11-3: Example of 2021 duplicate data and calculations used to determine Pass-Fail

ORIG_SampleID	ORIG_Au	DUP_SampleID	DUP_Au	PairAvg	RelDiff	pctTolerance	pctDifference	Pass-Fail
329299050	0.005	329299051	0.005	0.005	0.000	2080.0	0.00	PASS
329299070	0.005	329299071	0.020	0.013	-1.200	880.0	84.85	PASS
329299090	0.020	329299091	0.005	0.013	1.200	880.0	84.85	PASS
329299110	0.020	329299111	0.005	0.013	1.200	880.0	84.85	PASS
329299130	0.730	329299131	0.060	0.395	1.696	105.3	119.94	FAIL
329299150	0.005	329299151	0.005	0.005	0.000	2080.0	0.00	PASS
329299170	0.005	329299171	0.005	0.005	0.000	2080.0	0.00	PASS
329351230	0.030	329351231	0.005	0.018	1.429	651.4	101.02	PASS
329351250	0.230	329351251	0.320	0.275	-0.327	116.4	23.14	PASS
329351270	0.800	329351271	0.750	0.775	0.065	92.9	4.56	PASS

Table 11-3: Example of 2021 duplicate data and calculations used to determine Pass-Fail

ORIG_SampleID	ORIG_Au	DUP_SampleID	DUP_Au	PairAvg	RelDiff	pctTolerance	pctDifference	Pass-Fail
329351290	0.040	329351291	0.005	0.023	1.556	524.4	109.99	PASS
329351310	0.500	329351311	0.380	0.440	0.273	102.7	19.28	PASS
329351330	0.110	329351331	0.010	0.060	1.667	246.7	117.85	PASS
329351350	1.900	329351351	0.050	0.975	1.897	90.3	134.17	FAIL
329351370	1.030	329351371	3.260	2.145	-1.040	84.7	73.51	PASS
329351390	0.030	329351391	0.070	0.050	-0.800	280.0	56.57	PASS
329351410	0.030	329351411	0.005	0.018	1.429	651.4	101.02	PASS
329351430	1.200	329351431	0.960	1.080	0.222	89.3	15.71	PASS
329351450	0.150	329351451	0.100	0.125	0.400	160.0	28.28	PASS

Table 11-4: Example of 2022 duplicate data and calculations used to determine Pass - Fail

ORIG_SampleID	ORIG_Au	DUP_SampleID	DUP_Au	PairAvg	RelDiff	pctTolerance	pctDifference	Pass-Fail
329378310	0.720	329378311	0.590	0.655	0.198	95.3	14.03	PASS
329378330	0.005	329378331	0.005	0.005	0.000	2080.0	0.00	PASS
329378350	0.005	329378351	0.005	0.005	0.000	2080.0	0.00	PASS
329378370	0.005	329378371	0.005	0.005	0.000	2080.0	0.00	PASS
329378390	0.005	329378391	0.005	0.005	0.000	2080.0	0.00	PASS
329378410	0.005	329378411	0.005	0.005	0.000	2080.0	0.00	PASS
58000170	0.005	58000171	0.005	0.005	0.000	2080.0	0.00	PASS
58000190	0.005	58000191	0.005	0.005	0.000	2080.0	0.00	PASS
58000210	0.005	58000211	0.005	0.005	0.000	2080.0	0.00	PASS
58000230	0.010	58000231	0.030	0.020	-1.000	580.0	70.71	PASS
58000250	0.010	58000251	0.010	0.010	0.000	1080.0	0.00	PASS
58000270	0.010	58000271	0.005	0.008	0.667	1413.3	47.14	PASS
58000290	0.010	58000291	0.020	0.015	-0.667	746.7	47.14	PASS
58000010	0.005	58000011	0.005	0.005	0.000	2080.0	0.00	PASS

Example of duplicate control plots from 2021 (with failures) and 2022 (no failures in 2022) are shown in Figure 11-6 to Figure 11-13, with the remaining plots, split by drilling type and campaign, presented in Appendix A

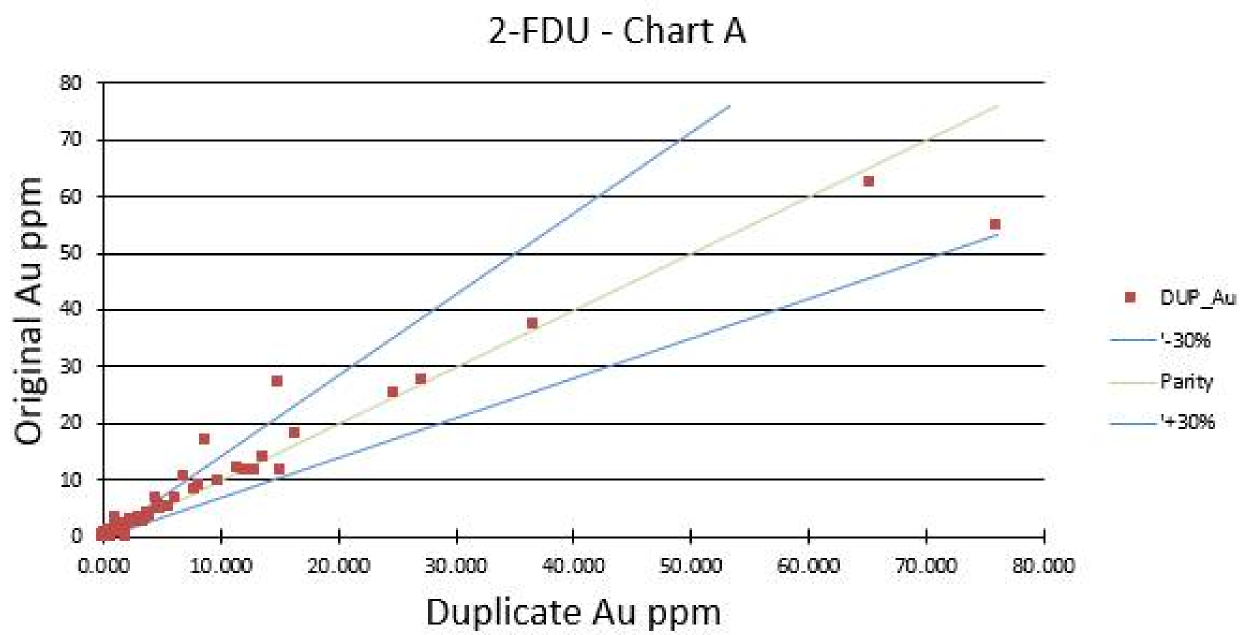


Figure 11-6: Field Duplicate Evaluation Chart A – 2021 - All data

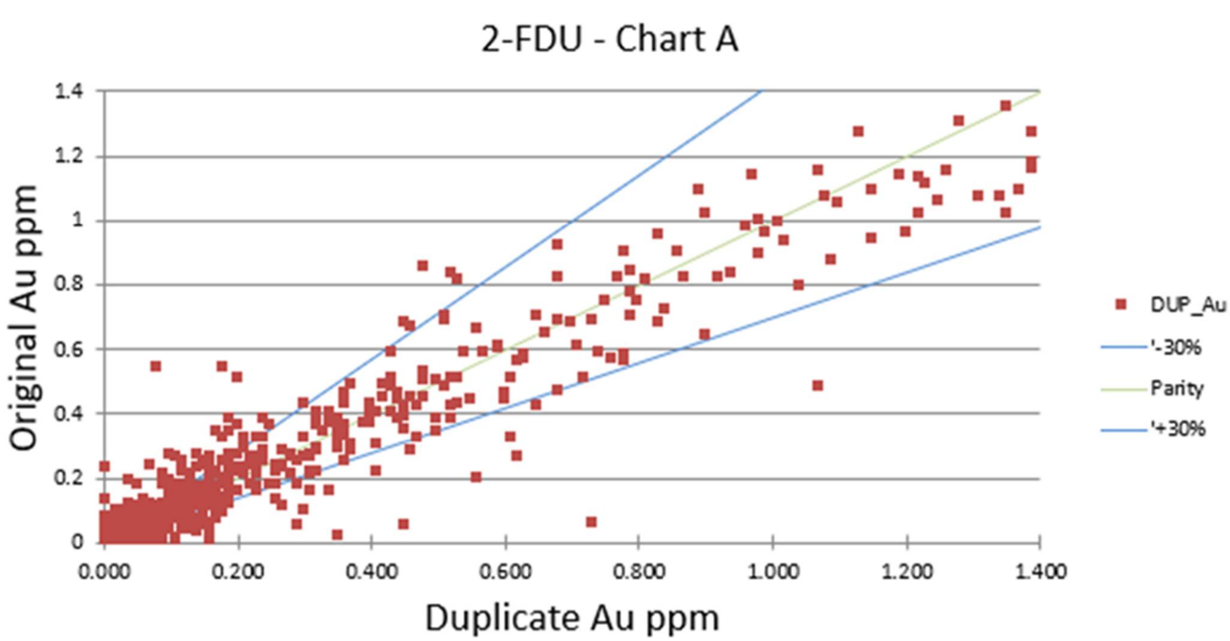


Figure 11-7: Field Duplicate Evaluation Chart A – 2021 - restricted to maximum 1.4 ppm Au

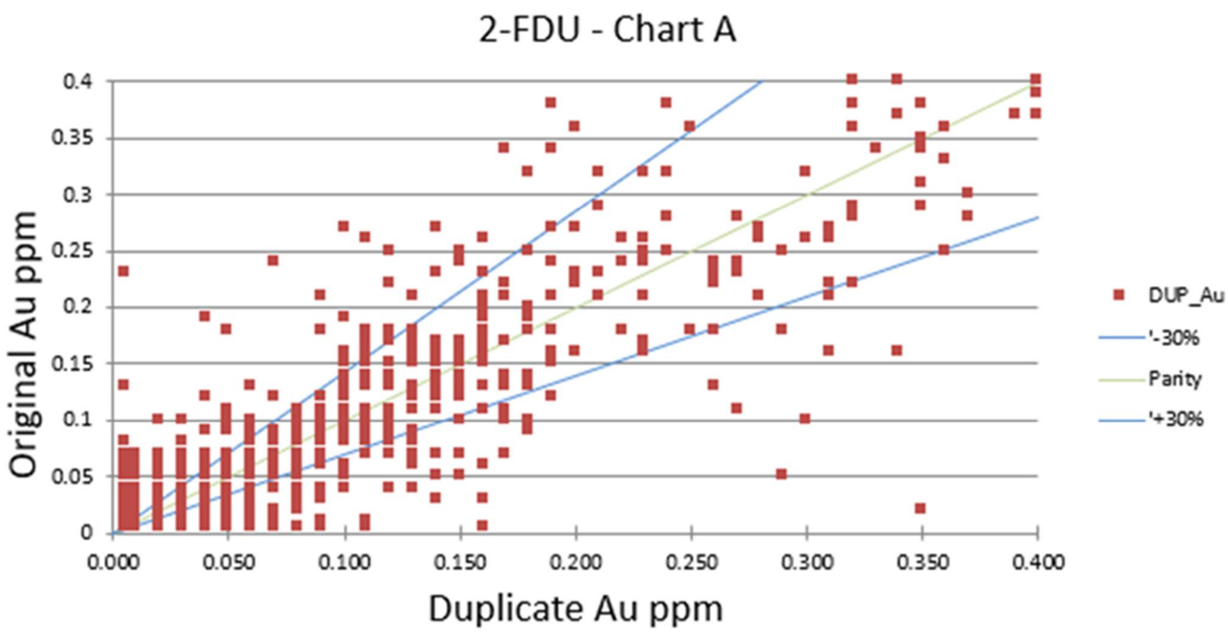


Figure 11-8: Field Duplicate Evaluation Chart A – 2021 - restricted to maximum 0.4 ppm Au

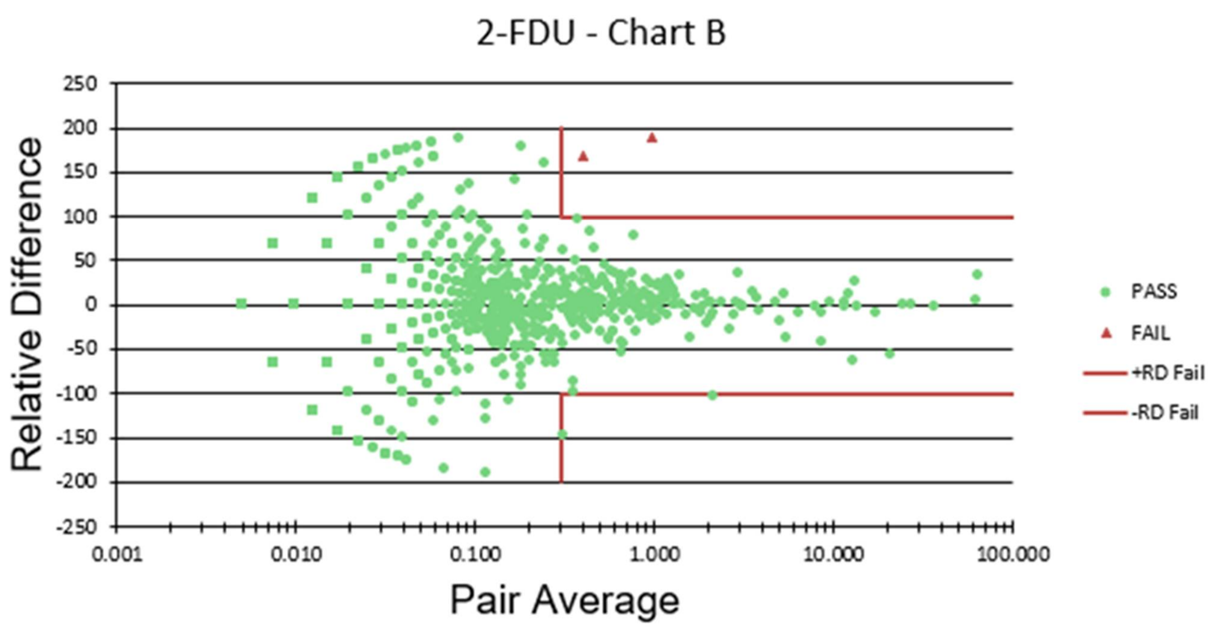


Figure 11-9: Field Duplicate Evaluation Chart B - 2021

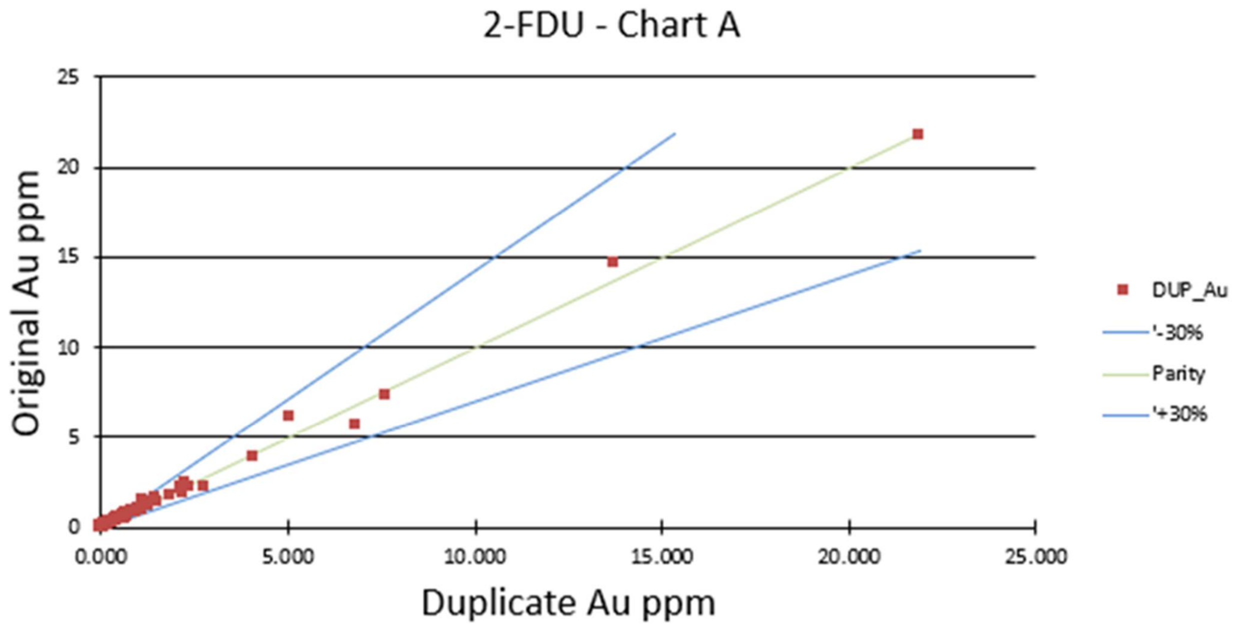


Figure 11-10: Field Duplicate Evaluation Chart A – 2022 - All data

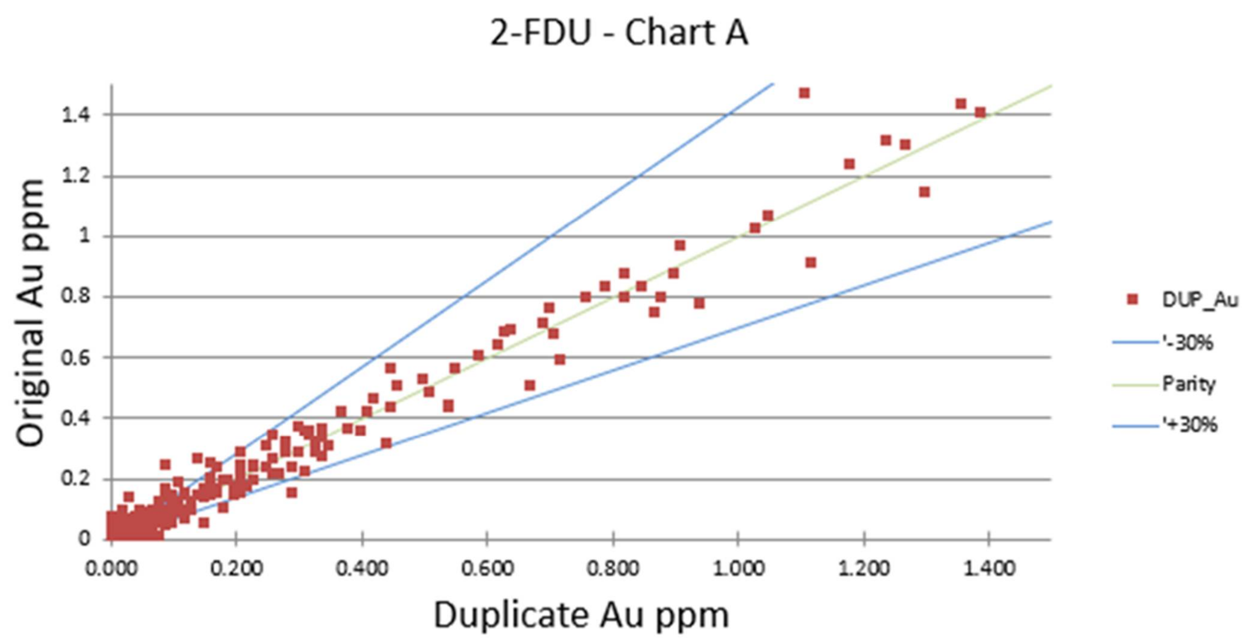


Figure 11-11: Field Duplicate Evaluation Chart A – 2022 - Restricted to 1.4 ppm Au (Max.)

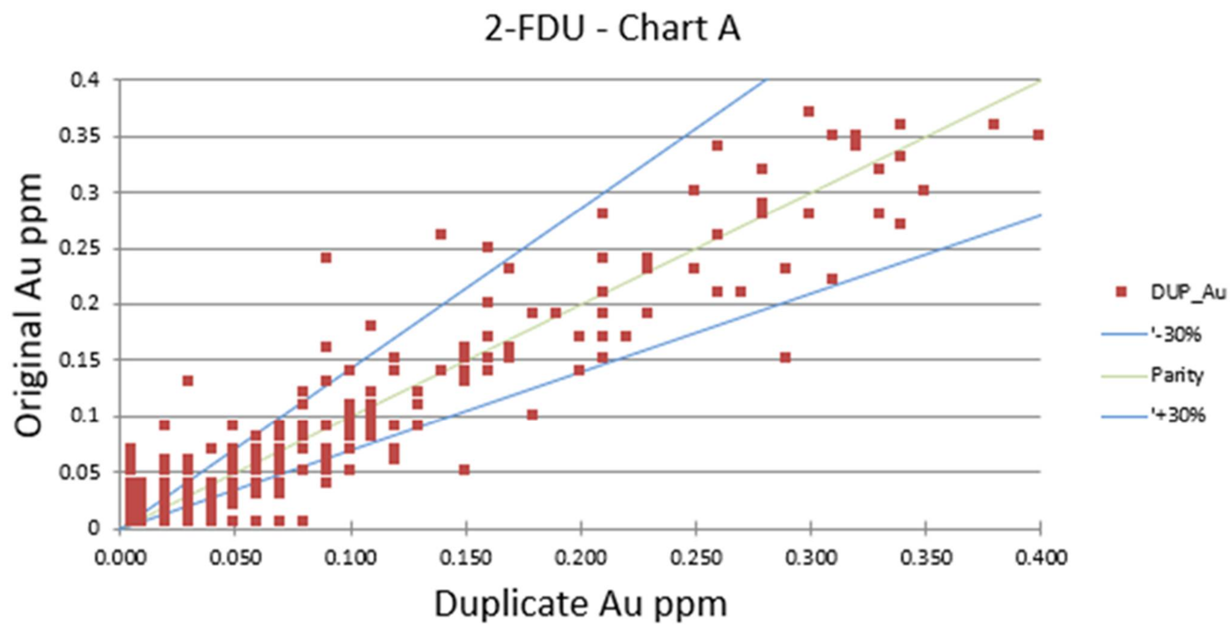


Figure 11-12: Field Duplicate Evaluation Chart A – 2022 - Restricted to 0.4 ppm Au (Max.)

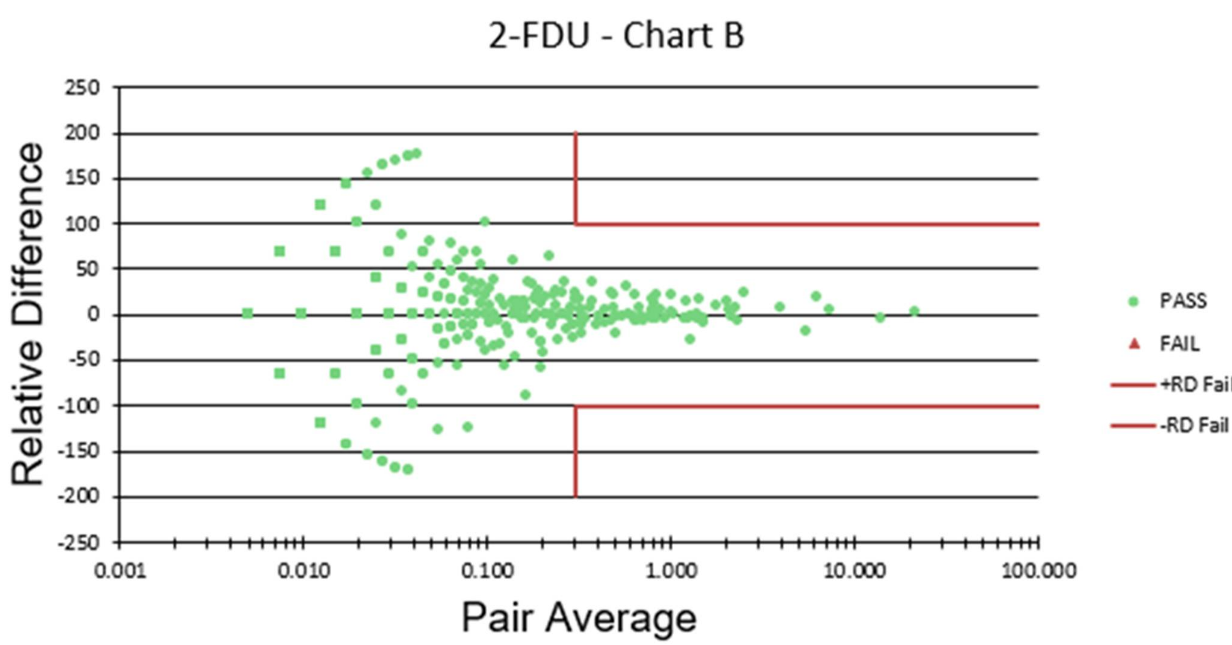


Figure 11-13: Field Duplicate Evaluation Chart B - 2022

In general, excluding a small number of anomalous or very high-grade results in the coarse material, the duplicate samples show a reasonable degree of correspondence with original samples. The coefficients of determination, i.e., R2 values, are listed in Table 11-5 and are typically within specified failure limits.

Table 11-5: R^2 Values of Duplicate Sample Pair Populations, Split by Drilling Campaign and Type (2010 to 2022)

Year	Drilling Type	Dup Type	R^2	No. Excluded (Anomalous Results)
2010	DD	Pulp	0.57*	3
	RC	Pulp	0.99	-
2014	DD	Pulp	0.41	-
	RC	Pulp	0.99	3
2017	DD	Field	0.98	-
	RC	Field	0.96	16
2018	DD	Field	0.90	3
	RC	Field	0.94	-
	RC-DD	Field	0.99	-
2019	DD	Field	0.99	-
	RC	Field	0.96	3
	RC-DD	Field	0.98	-
2020	RC	Field	0.98	-
	RC-DD	Field	0.92	-
2021	RC	Field	0.99	-
	RC-DD	Field	0.93	-
2022	RC	Field	0.99	-
	RC-DD	Field	0.99	-

Table 11-5 Note: *Only 10 duplicate pairs analysed

It is noted that the selected duplicate samples provide reasonable coverage in the context of the average mineralization domain grades, ranging from below detection limit into the tens-of-ppm in most drilling and sampling campaigns. Full duplicate charts are presented in Appendix A.

11.8.4 Certified Reference Materials (CRM)

CRM are samples that can be used to measure the accuracy of analytical procedures and are composed of material that has been thoroughly analysed and certified by several laboratories to accurately determine its grade within known error limits. The CRM used at the Lafigué Project were sourced from Geostats and covered a grade range of 0.63 to 5.85 ppm Au. Since 2019, three CRM (G913-9, G914-2 and G915-6) have primarily been used, covering a grade range of 0.67 to 4.91 ppm Au.

In total, the CRM insertion rate between 2010 and 2022 drilling was 5.2%, with an overall failure rate of <1% (outside ±3 SD). Based on a review of CRM failures, the majority can likely be attributed to inadvertent CRM sample swaps. It is noted that CRM G310-6 had some issues systematically under-reporting Au by approximately 0.05 ppm (8%) between 2017 and 2018, however in the context of none of the other CRM, including another CRM in the same grade range (G910-8) significantly and systematically under-reporting Au grade, and of the CRM in question having been retired in 2018, this issue is not considered material to the accuracy of assay results which form the basis of the 2022 MRE.

Additionally, gold was systematically under-reported by approximately 0.1 ppm (10%) for CRM G910-10 during the 2022 drilling programme up until 12 May 2022 (a total of 12 sample submissions), whereafter performance of the CRM abruptly improved to be approximately aligned with the certified value for this material (0.96 ppm). Endeavour considers it likely that the earlier under-performance of this CRM is attributed to a systematic mislabelling of a batch of these CRM with an alternative, lower grade CRM (G913-1 – 0.82 ppm Au) which was used on a separate exploration programme. This issue is not considered material to the accuracy of assay results which form the basis of the 2022 MRE, however it is recommended that sequences of inaccurate results for a given CRM are monitored more closely such that these may be investigated promptly in future.

A summary of CRM sample performance, split by drilling campaign and drilling type, is provided in Table 11-6 with all CRM control plots presented in the appendices of the May 2022 SRK MRE Report (Appendix A).

Table 11-6: Summary of CRM Performance, Split by Drilling Campaign and Type

Year	Drilling Type	CRM	Number of Submissions	Gold Grade (g/t)	Standard Deviation	Number of Failures	Failure Rate
2010	DD	G307-2	2	1.08	0.05	0	0.0%
		G311-2	2	4.93	0.18	0	0.0%
		G910-8	4	0.63	0.04	0	0.0%
		G998-8	3	5.85	0.39	0	0.0%
	RC	G998-8	20	5.85	0.39	1	5.0%
2014	DD	G307-2	38	1.08	0.05	0	0.0%
		G311-2	38	4.93	0.18	0	0.0%
		G910-8	38	0.63	0.04	0	0.0%
	RC	G307-2	106	1.08	0.05	1	0.9%
		G311-2	107	4.93	0.18	0	0.0%
		G910-8	108	0.63	0.04	0	0.0%
2017	DD	G300-8	12	1.07	0.06	0	0.0%
		G310-6	4	0.65	0.04	0	0.0%
		G316-2	3	1.04	0.04	0	0.0%
		G910-8	10	0.63	0.04	0	0.0%
		G913-9	12	4.91	0.17	0	0.0%
	RC	G300-8	103	1.07	0.06	1	1.0%
		G910-8	103	0.63	0.04	1	1.0%
		G913-9	105	4.91	0.17	0	0.0%

Table 11-6: Summary of CRM Performance, Split by Drilling Campaign and Type

Year	Drilling Type	CRM	Number of Submissions	Gold Grade (g/t)	Standard Deviation	Number of Failures	Failure Rate
2018	DD	G310-6	66	0.65	0.04	1	1.5%
		G316-2	36	1.04	0.04	2	5.6%
		G913-9	59	4.91	0.17	1	1.7%
	RC	G300-8	5	1.07	0.06	0	0.0%
		G310-6	126	0.65	0.04	1	0.8%
		G316-2	84	1.04	0.04	2	2.4%
		G910-8	5	0.63	0.04	0	0.0%
		G913-9	133	4.91	0.17	1	0.8%
		G914-2	59	2.48	0.08	0	0.0%
		G915-6	28	0.67	0.04	0	0.0%
	RC-DD	G310-6	52	0.65	0.04	0	0.0%
		G316-2	47	1.04	0.04	2	4.3%
		G913-9	44	4.91	0.17	0	0.0%
2019	DD	G913-9	19	4.91	0.17	0	0.0%
		G914-2	13	2.48	0.08	0	0.0%
		G915-6	14	0.67	0.04	0	0.0%
	RC	G913-9	587	4.91	0.17	1	0.2%
		G914-2	583	2.48	0.08	2	0.3%
		G915-6	598	0.67	0.04	1	0.2%
	RC-DD	G913-9	143	4.91	0.17	0	0.0%
		G914-2	154	2.48	0.08	0	0.0%
		G915-6	139	0.67	0.04	0	0.0%
2020	RC	G913-9	697	4.91	0.17	1	0.1%
		G914-2	679	2.48	0.08	0	0.0%
		G915-6	669	0.67	0.04	0	0.0%
	RC-DD	G913-9	742	4.91	0.17	2	0.0%
		G914-2	745	2.48	0.08	0	0.0%
		G915-6	742	0.67	0.04	2	0.3%
2021	RC	G913-3	43	2.36	0.18	0	0.0%
		G913-9	503	4.91	0.17	0	0.0%
		G914-2	693	2.48	0.08	0	0.0%
		G915-6	651	0.67	0.04		0.0%
	RC-DD	G913-3	70	2.36	0.18	0	0.0%
		G913-9	147	4.91	0.17	0	0.0%
		G914-2	398	2.48	0.08	0	0.0%

Table 11-6: Summary of CRM Performance, Split by Drilling Campaign and Type

Year	Drilling Type	CRM	Number of Submissions	Gold Grade (g/t)	Standard Deviation	Number of Failures	Failure Rate
		G915-6	405	0.67	0.04	0	0.0%
2022	RC	G913-3	375	2.36	0.18	0	0.0%
		G914-2	367	2.48	0.08	0	0.0%
		G915-6	389	0.67	0.04	1	0.3%
		G318-10	49	4.58	0.17	0	0.0%
		G302-3	12	2.33	0.12	0	0.0%
		G910-10*	12	0.97	0.04	2	16.7%
	RC-DD	G913-3	27	2.36	0.18	0	0.0%
		G914-2	24	2.48	0.08	0	0.0%
		G915-6	24	0.67	0.04	0	0.0%
		G318-10	3	4.58	0.17	0	0.0%

Table 11-6 note:*G910-10 samples prior to 12 May 2022 considered likely to be sample swaps/mislabelled – see explanation in text.

11.8.5 Umpire Samples

Pulp samples from the 2017 to 2019 drilling programs were sent to ALS Ouagadougou in 2018 and 2019 for Umpire analysis. Umpire samples were primarily selected from mineralized zones although some lower grade and barren samples were also submitted.

Umpire samples were analysed for gold by fire assay (ALS method code Au-AA26), with QAQC samples submitted as per Endeavour's standard operating procedures (Section 11.8.2 to 11.8.4). The results from the analysis of original and umpire assays received and reviewed in 2018, including samples from 2017 and 2018 drilling, provided a satisfactory coefficient of correlation of 89.3 % when high grade samples were excluded. These high-grade samples returned a much lower coefficient of correlation of 45.4 %, reflecting the higher variance associated with these high-grade intervals. Original and umpire assays received and reviewed in 2019, including samples from 2018 and 2019 drilling, returned a coefficient of correlation of 90.6 % for all samples.

Samples from the 2020 to 2022 drilling programmes were recently sent for Umpire analysis. Results for these analyses have not been received as of the 'Effective Date' of this report.

11.8.6 Failure Procedures

The Issuer's approach when a blank, duplicate or CRM failure has been encountered in any of the 2014 or later drilling programmes was initially to determine if the failure is genuine or a data entry issue. Data entry issues were identified by cross-checking the relevant original paper sample logs against the database. CRMs were additionally checked against photographs taken of each CRM sachet directly prior to insertion into the sample bag. Sample weights were checked to verify that the sample providing the failed values are, in fact, QA/QC samples.

Genuine failure procedures are summarised as:

- Duplicate Failure: The weight of the duplicate pairs is checked and should be similar. If a gold value in the duplicate pair exceeds 1 g/t and there is a large contrast between the original and duplicate assay grades, the samples are assumed to be in a zone of coarse (nuggety) gold and no re-assay is requested. If the sample grades are both lower and the difference in grade between samples is within normal failure ranges, a re-assay is requested.
- Blanks Failure: Always sent for re-assay
- CRM Failure: The weight of the failed CRM is checked. The weight of a CRM sample is always much less than a regular sample submission, e.g. (0.5 to 0.6) kg for a CRM versus over 2 kg for a regular sample. If a CRM failure occurs in a zone lacking gold mineralisation, the laboratory is informed of the failure, but a re-assay request is not made. If the failed CRM occurs in a zone of gold mineralisation, the laboratory is informed, and a re-assay request submitted.

When a re-assay of a failed quality control sample is required, all regular samples in the sequence between other QAQC samples (typically, ten samples before and ten samples after the failed QA/QC sample) are re-assayed. The re-assay sample is generated by sampling an additional 50 to 60 g from the original 250 g pulp sample. If the re-assay sequence includes CRMs, replacement CRMs are provided. If there is insufficient original pulp material for a re-assay, material from the coarse reject is sourced and a new pulp is prepared and provided for re-assay after pulverisation.

11.8.7 Data Management

The Fetekro data is stored and organized between the project server and Endeavour Exploration's (EEL) Central Database Management server in Canada. Supporting data is organized in a standard file format and folder structure across all EEL projects. The critical data is captured and stored in an industry standard, relational, Microsoft SQL based, Database Management System (DBMS). Assay and analytical QAQC data is handled exclusively by the Central Database Management team operating independently of and external to the projects level teams.

The database is easily queried to assess the state and accuracy of the contained data. All supporting data and original lab generated lab assay certificates are available for audit internally or by third party auditors.

Access to the supporting data, the SQL based databases and the DBMS are closely restricted to only authorized personnel. Strict Information Technology controls restrict access to these data sources. All data on both project level and Central Database Management servers in Canada have comprehensive backup systems implemented with all data backed up to a local NAS server as well as daily cloud-based backups.

11.9 Independent Audits

In 2017 Andre Vorster from CSA-Global visited the Project site to review sampling and QAQC protocols. Following implementation of a series of recommendations, Andre Vorster revisited the site for a follow-up review in 2019. In 2021 a Laboratory auditing specialist, John Coates, from SEMS International audited the primary Bureau Veritas laboratory and the ALS Ouagadougou Umpire lab.

11.10 Comments

The QP considers that the majority of sample preparation, analyses and security protocols employed at the Lafigué Project conform to industry best practice and that the QAQC sample results available for drilling since 2010 are generally of sufficient quality to support the 2022 Mineral Resource estimate.

11.11 Interpretation and Conclusions

Interpretations, conclusions and risks pertaining to Section 11, are summarised in Section 25 of this Report.

11.12 Recommendations

Recommendations for Section 11 are presented in Section 26 of this Report.

11.13 References

References cited in the preparation of Section 11, are as defined in Section 27 of this Report.

12. DATA VERIFICATION

12.1 Introduction

In accordance with Item 12 of NI Form 43-101F1 and the Qualified Person's responsibilities defined in Table 2-3, the following subsections summarise where relevant:

- the data verification procedures applied by the Qualified Person(s);
- any limitations on or failure to conduct such verification, and the reasons for any such limitations or failure; and,
- the Qualified Person's opinion on the adequacy of the data for the purposes used in this Report.

12.2 Geology and Resources

The following Section summarises the data verification process applied to Sections; 6, 7, 8, 9, 10 and 11 of this Report.

12.2.1 Site /Facility Visits

SRK Qualified Person, Dr Lucy Roberts, Principal Consultant (Resource Geology), and Dr James Davey, Consultant (Resource Geology), visited the site between 14 May and 16 May 2021. The visit involved a tour of the PE 58 Exploitation Permit area; verification of a selection of drillhole collar positions; a review of selected drillcore and RC chip samples; discussion on the geological and mineralisation interpretation; and a review of some quality assurance/quality control ('QA/QC') procedures employed by the Issuer.

Andre Vorster and Fabien Linares from CSA-Global visited the Fetekro project in May 2017. Andre Vorster visited the Project again in May 2019. CSA-Global undertook a comprehensive review of all sampling and QAQC procedures in the field, as well as at the sample management and storage facilities. Comprehensive reports with recommendations and guidelines for industry best practice were provided at the end of these visits.

12.2.2 Geological Data

12.2.2.1 Historical Data Validation

Upon acquisition of the Project in 2016, the Issuer implemented a SQL-based database management system ('DBMS'), where all historical data generated from the Fetekro Project was audited during the importation process. Errors screened during this process included:

- inconsistent collar coordinates;
- incorrect or missing down hole survey records;
- missing sample assay records; and
- missing or overlapping downhole interval records.

During the database upgrade process all original Assay certificates were located and reimported into the new DBMS.

No material issues with the historical drillhole data used for the 2022 MRE have been identified, however the absence of QAQC sampling during these periods (prior to 2010), as discussed in Section 14, is noted.

12.2.2.2 Database and Work Programme Verification

Since 2013 all data acquired across the Fetekro Project area is managed using the built-in data integrity requirements of an industry standard SQL-based DBMS, where database checks include identifying:

- inconsistent collar coordinates;
- incorrect or missing DTH survey records;
- missing assay records;
- missing data or overlapping interval errors; and
- incorrect 3D plotting of drillhole traces.

Any errors highlighted during this process are actioned by the Issuer's database management team as appropriate.

Prior to the exportation of a final database from the DBMS, an audit is undertaken by the central database team within the DBMS. Additional checks are completed by the database management team using the software-based auditing tools provided in the Geosoft Target, Surpac and Leapfrog analysis packages.

Although the database management procedures carried out by the Issuer have not been independently verified, a review of the exported database by SRK as part of the 2022 MRE process did not highlight any major issues. Adjustments made to the drillhole database for use in the MRE are summarised in Section 14. In addition to the above database verification procedures carried out by the Issuer's database management team, SRK cross-checked a selection of assay results in the drillhole database with their corresponding original laboratory assay certificates and identified no significant issues.

12.2.3 Database Management

In 2016 the Issuer's Exploration Central Database team officially implemented an industry standard, relational, Microsoft SQL based, Database Management System (DBMS) for the Fetekro project. Data from this system has been disseminated to external auditors, but no direct audit of the system has been undertaken by external auditors.

12.2.3.1 Collar and Survey Verification Surveys

A total of thirteen 2018 and 2019 drillhole collars were resurveyed by Kouamelan during 2020, with no material discrepancies found. Subsequently, all drillholes completed since 2020 have also had verification collar surveys to confirm their positions. Prior to undertaking the MRE, both collar and downhole surveys were checked visually in 3D in order to highlight any clear errors in the survey readings. Additionally, Qualified Person, Dr Lucy Roberts and Dr James Davey of SRK verified the position of five drillhole collar locations (from drilling campaigns ranging from 2014 to 2021) during their visit to site between 14 May and 16 May 2021.

12.2.3.2 QA/QC Procedures

QAQC procedures are generally aligned with industry best practice. The field level QAQC procedures were reviewed by Andre Vorster and Fabien Linares from CSA-Global in 2017. They were further verified by Andre Vorster from CSA-Global in 2019. Recommendations from the 2017 CSA review were focused on implementing standardised sampling procedures and contamination mitigation procedures, both of which were synthesised into a series of Standard Operating Procedures ('SOPs'). Recommendations from the 2019 review included further optimisation of the SOPs generated from the earlier 2017 review based on observations of what procedures might work best in practice. Additional recommendations were made for clearer allocation of QAQC oversight responsibilities to ensure key field staff understood who was responsible for contamination mitigation. Recommendations for improved sample recovery tracking were also introduced.

The QAQC procedures used for laboratory-generated analytical data evaluation were verified and partially developed by CSA-Global in 2017. CSA evaluated all analytical QAQC data generated from the Issuer's active projects in 2017 and provided recommendations and SOPs for evaluating QAQC sample results (this included various calculations, statistics and charts which could be programmed into the DBMS QAQC report generator for ongoing data review).

12.2.3.3 Twinned Hole Comparison

No twinned drillholes have been completed at Lafigué, however a limited number of pairs of drillholes each spaced within 4 m of each other, does allow some short-scale comparisons to be made. A statistical comparison of samples within the estimation domains in three examples of these pairs of drillholes (Table 12-1) indicate a relatively poor relationship between close-spaced drillholes, particularly where grades in one of the holes is elevated, as seen in D0597A. This is likely predominantly due to the inherent nugget effect and short-scale variability of gold mineralisation, however bias in the contrasting sampling procedures between drilling campaigns, such as between 2002 versus 2014, and between RC and DD holes cannot be precluded on the basis of these limited data. It is noted, however, that although Au grades are not always continuous over short distances, the available paired drillholes do broadly delineate the same package of mineralised rock and therefore support the interpreted reasonable continuity of the mineralised structure(s).

A visual comparison of Au grades within the three examples of close-spaced drillholes detailed in Figure 12-1. It is recommended that twinned drillholes are completed at several, representative locations across the deposit, including a comparison of RC and DD types.

Table 12-1: Summary statistical comparison of close-spaced drillholes at Lafigué

Hole ID	Year	Separation (m)	Weighted Average Grade		Main Mineralised Interval Thickness	
			Au (g/t)	Delta	m	Delta
LF14-039	2014	3.5	2.5	-39%	5.7	58%
LFRC02-50	2002		1.5		9.0	
D0597B	1997	3.5	3.6	44%	31.2	12%
D0597A	1997		5.2		35.1	
LFDD19-669	2019	2.5	4.31	-24%	4.3	65%
LFRC02-56	2002		3.29		7.0	

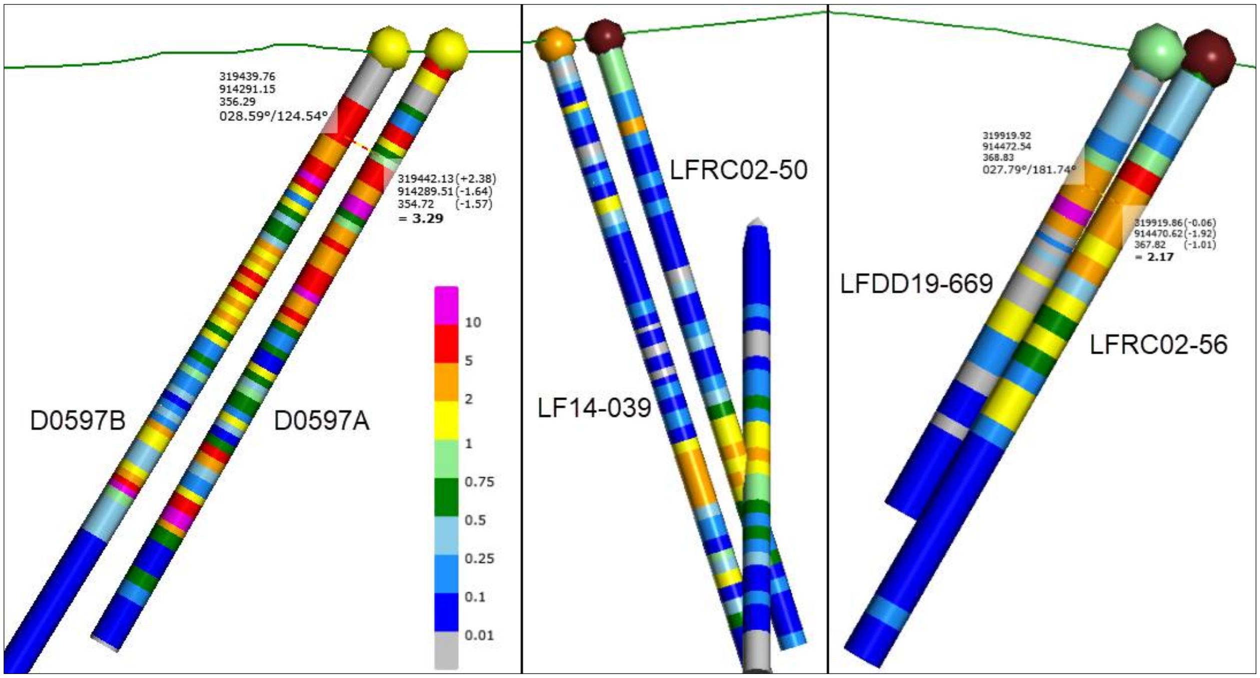


Figure 12-1: Cross-section views showing a visual comparison of Au grades in close-spaced drillholes

12.2.3.4 Paired Statistics

No paired statistics have been undertaken for drillhole data from Lafigué.

12.2.3.5 Witness Samples

For drilling completed since 2014 witness samples were retained for reference. For RC, AC and ARC drilling, reference samples were taken at the drill site. These Reference samples were sent to the secure sample storage facility at the Fetekro exploration camp for long term storage. Diamond drill core was cut in half (longitudinally) , generating one regular sample for submission to the laboratory for analysis and another which was retained in the core box for reference. Core boxes were sent to the secure core storage facility at the Fetekro exploration camp for long term storage.

12.2.4 Comments on Section 12.2

Through a series of independent site visits and implementation of SOPs aligned with industry best practice sampling and analytical procedures, the QP is satisfied that the data used in support of the current Mineral Resource estimate is sufficiently reliable for the declaration of Mineral Resources in accordance with CIM Definition Standards guidelines (CIM, 2014). Where possible, historical data generated from the Fetekro Project has been audited and some collar positions have been independently verified. Confidence in the accuracy of historical drillhole data could be improved through a twinned drillhole programme including QAQC sampling procedures aligned with industry best practice. The QP notes that other than database auditing procedures, pre-2010 drillhole data has not undergone a detailed verification process due to limitations with sample availability.

Further discussion is provided in Section 25.9.1.

12.3 Property Description and Location

The QP for Section 4 has worked with the respective 'Experts' as noted in Section 3, to ensure that the data presented is aligned with the requirements of a DFS/NI 43-101 Technical Report and is not misleading. Where there are concerns, these are stated.

12.4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The data verification process applied in compiling Section 5 is discussed more fully by thematic area below.

- Climate

  The QP has reviewed the data prepared by Knight Piesold and compared to other data sourced from the World bank/WMO/SODEXAM. Whilst rainfall data is largely in alignment, the evaporation data presented is not site specific. However, for evaporation any discrepancies between actual site data and the data presented by Knight Piesold, is unlikely to materially affect the development of the study or operations. Knight Piesold have considered the data and state that its appropriate for the feasibility study.

  There are concerns with respect to the quality of the wind rose data, but again the data is believed to be directionally fair (vector and scalar), and any errors are unlikely to materially impact the Project or mine. There are concerns with the use of the wind data for dust/noise dispersion modelling and to date, no noise or dust dispersion modelling has been undertaken.

- Population and Demographics

  The population and demographics data is now outdated and likely inaccurate. Notwithstanding this, the outputs are directionally fair and provide sufficient guidance to the Issuer. More recent data will be available once the results of the 2021 RGPH census survey are released.

- Road, Rail, Fuel Power and Communications Infrastructure

  The QP for this section has used a number of public domain sources, including data from intergovernmental organisations to compile this section. This data has been further corroborated with the Issuer's staff who have been in contact with the relevant stakeholders. The data presented is considered fair and appropriate for use in the feasibility study.

12.5 Mineral Processing and Metallurgical Testing

ALS Metallurgy in Perth were selected to conduct the testwork having demonstrated reliable laboratory procedures, expert support, good quality control procedures, accurate and timely reporting; and high levels of consistency in conducting previous testwork programmes.

Comminution testing is critical to the Project, typically being the highest capital and operating cost centre and also the most critical area of the plant in terms of ensuring nameplate throughput is met. The comminution testwork suite selected is widely regarded as one of the best approaches for ore characterisation in terms of comminution energy input required:

- The SMC test provides a cost-effective means of obtaining a range of other power-based comminution parameters from drill core. This is a precision test, using particles that are either cut from drill core using a diamond saw to achieve close size replication or else selected from sized, crushed material so that particle mass variation is controlled within a prescribed range.

- The particles are then broken at a number of prescribed impact energies. The high degree of control imposed on both the size of particles and the breakage energies used ensures that the test is largely free of the repeatability problems associated with tumbling mill-based tests.
- The data from the breakage tests conducted by ALS is analysed by JKTech for quality control and referencing to their extensive ore database to ensure standardised results that can be reliably used.
- The balance of standard Bond Index determinations produce results that are well understood in the industry and supported by large databases of reference examples and industrial practice.

ALS understands the value of data quality and integrity for engineering and mining companies. The ALS quality program consists of a series of checks and balances with monitoring at senior management levels. Their global information management system provides oversight and access to all processes. The ALS metallurgy and analytical facilities are accredited to ISO 9001- 2008 standards.

All of the assay samples generated during the course of the gravity concentration and gold leaching test program were submitted to the ALS analytical laboratory in Perth for analysis.

The following analytical techniques were employed:

- Gold in ores and leach residues:Fire assay/ICP-MS
- Gold in solution:Direct ICP-MS
- Multi-element assay in solids:Mixed acid digestion/ICP-OES

12.6 Mineral Reserves

SRK declares that it has taken all reasonable care to ensure that the information contained within Section 16 of the Lafigué Project DFS is, to the best of its knowledge, in accordance with the facts and contains no omission likely to affect its import. SRK has relied upon the accuracy and completeness of technical and financial information data furnished by or obtained from Endeavour. Endeavour has confirmed to SRK that, to its knowledge, the information provided by Endeavour (when provided) was complete and not incorrect or misleading in any material respect. SRK has no reason to believe that any material facts have been withheld.

This Lafigué DFS report includes technical information, which requires subsequent calculations to derive subtotals, totals and weighted averages. Such calculations may involve a degree of rounding and consequently introduce an error. Where such errors occur, SRK does not consider them to be material.

In accordance with 'Item 12 of NI Form 43-101F1 and the Qualified Persons' (QP) responsibilities defined in Table 2-3, the following subsections summarise where relevant:

- the data verification procedures applied by the Qualified Person(s);
- any limitations on or failure to conduct such verification, and the reasons for any such limitations or failure; and,
- the Qualified Person's opinion on the adequacy of the data for the purposes used in this Report.

12.6.1 Geological Resource Model

The Mineral Resource estimate for the Lafigué deposit has been classified in accordance with the CIM Definition Standards and includes Indicated and Inferred Mineral Resources. The Geological Resource model was re-reported to ensure the model matches in terms of grade and tonnes to what was stipulated in the model handover notes.

12.6.2 Mining Depletion

To date, the Lafigué Deposit has not been mined on a commercial scale but has been subject to substantial artisanal mining. The survey was of sufficient resolution to resolve the main open pit working areas which were typically on the scale of 10s of metres at surface, and less than 10 m deep. The QP is satisfied that the appropriate steps were followed to as accurate as posable deplete the Mineral Resource and that suitable modifying factors are applied to account for losses and dilution as a result of the artisanal mining.

12.6.3 Modifying Factors

The May 2022 Mineral Resource block model was re-blocked to the selective mining unit ('SMU'), size 5 m x 5 m x 2.5 m along the X-direction, Y-direction, and Z-direction respectively, to create regularised mine planning models. The re-blocking process is seen as an industry best practice and one of the most elegant ways of applying proper dilution to the resource models containing too small a block size to be mined selectively.

On the basis that the Lafigué deposit has not been mined historically, an additional 5% dilution was added on a block-by-block basis. The additional factor is applied in addition to the modifying factors incurred during the regularisation process. The QP is satisfied that sufficient modifying factors are applied to account for losses and dilution as a result of mining.

12.6.4 Mineral Reserves Estimates

The QP confirms that the Mineral Reserve statement presented in Section 15 has been derived from the Mineral Resource with an 'Effective Date' of 15 May 2022 authored by SRK (31113_Lafigué_MRE_May_2022).

12.7 Mining Methods

The data verification process applied in ensuring that the data used and presented herein is valid and suitable for use in the DFS is discussed in Section 12.

12.7.1 Hydrogeological Review

A review of the hydrogeological assessment report completed by Endeavour Mining in 2021, and associated data provided within the appendices was conducted, as well as supporting data regarding the design climatology that forms the basis of the hydrology data as presented by Knight Piesold (KP, 2021). The review of the Hydrological data excludes independent verification by means of re-calculation.

The QP considers that the pit hydrogeological characterisation and dewatering design has been undertaken to a 'pre-feasibility study level' of development. However, whilst there are hydrological risks, these are not considered significant relative to the geotechnical risks. Notwithstanding this, further work is required to better define the geological structural model and the associated hydrogeological conditions, and this work should be done during detailed design/FEED and prior to start of mining.

12.7.2 Geotechnical Review

A review of the Bastion Geotechnical Engineering (BG) feasibility study report (Bastion, 2021) and associated data provided within the appendices has been done. The review of the Geotechnical data excludes independent verification by means of re-calculation.

The QP considers that BG has implemented a diligent review of the pre-existing geotechnical data, identifying gaps and defining confidence levels within the various models feeding into the Geotechnical Model and updating the slope stability analyses. The QP is aligned with the conclusion and recommendations of BG, with the slope design being reasonable and adequate to support the Ore Reserves for the Lafigué Project for this level of study.

12.8 Recovery Methods

The QP for Section 17 has verified the data used and considers that the data as applied in the development of the Plant scope is reasonable, and in-line with the requirements of a DFS. Refer to Section 13 for comments on the data verification of metallurgical inputs to the Plant DFS design.

12.9 Project Infrastructure

12.9.1 Seismic Assessment

The data used and presented in the preparation of the seismic hazard assessment is considered fair and appropriate for this level of study.

12.9.2 Engineering Geotechnical

The location of the engineering geotechnical testwork samples has been reviewed and are aligned with the proposed Site infrastructure features. Further, the testwork results and interpretation thereof are considered appropriate and fair, and suitable for use in the DFS without any reservations.

12.9.3 Airstrip

The wind data utilised for the airstrip design was sourced from a meteorological station located 80 km from PE 58. The prevailing wind direction is south-southwest to north-northeast for the Bouake station. The Lafigué airstrip alignment runs parallel to the prevailing wind direction. Based on a review of satellite imagery, the airstrip direction corresponds with other airstrips in proximity to PE 58. The airstrip alignment has been approved by Nationale de l'Aviation Civile de Côte d'Ivoire (ANAC) and is therefore considered suitable for application.

12.9.4 Climate

The design climatology (KP, 2021b) was undertaken in 2020 and further reviewed in 2021 by KP, resulting in no revisions. The closest source of daily precipitation to the project was the Dabakala meteorological station (25 km northeast of the project site), which provided daily precipitation records from 1922 to 2000 and was utilised for storm term storm events, monthly and annual analysis and water balance scenarios. 89% of the dataset was suitable for the climatology assessment, which provides a good level of confidence in the climate assessment outcomes.

A search of public domain pan evaporation data in the vicinity of the Site did not identify a reliable source. Average monthly pan evaporation values were found for Zuenoula and Ferké, approximately 175 km to the southwest and 165 km to the northwest of the site. The dataset is sufficient for the study; however, installing an automated weather station at Site will allow the data to be calibrated to the Site during the project/operations.

12.9.5 Mine Services Area (MSA)

For the MSA specifically, Endeavour went through a number of mining contractor tenders for the design and costing of the MSA infrastructure. The scope of facilities and approach to costing is reasonable, and appropriate for the feasibility study.

12.9.6 Emulsion Plant and Explosives Storage

The QP has reviewed the benchmark data from a similar Endeavour CI operation and compared against the layout and costs for the proposed Lafigué emulsion and explosive facilities. The QP considers the facility size, capacity and costs used in the DFS, appropriate to support mine operations over the LOM.

12.9.7 Power

The QP for power supply, is of the opinion that the information contained within this document in relation to grid connected power supply is not misleading and adequate for the purpose of use (the DFS).

Further, on the basis of having worked in West Africa for more than 30 years and within the last five years on four mining projects in CI, not including two further projects currently under construction in CI, the author believes he is well positioned to ensure the validity of the data used and the results presented.

Estimates have been gathered for use within this report by ECG Engineering Pty Ltd (ECG) from contractors and suppliers currently operating in CI.

12.9.8 Water

KP has reviewed the following data sets and used appropriately in the development of the water balance model, earthworks quantities and engineering layouts. The data presented is considered appropriate for its intended use.

- climatology data used (previously commented on in Section 12.9.4)
- the mine plans (Schedule 12 and 13);
- LIDAR survey; and
- Engineering geotechnical testwork.

12.9.9 Waste Management

Tailings design parameters were adopted based on laboratory testing results, and are considered appropriate providing that the sample taken are representative of the LoM plan. Testing of operational tailings samples should be completed routinely during operations to verify the original testing results. Tailings geochemistry testing was completed on the bulk samples received for the physical tailings testwork. These are considered appropriate for a DFS level design.

12.9.10 Balance of Infrastructure

Lycopodium have reviewed the base technical data prior to use and consider the data used in the development and design of the balance infrastructure reasonable, and in-line with the requirements of a DFS.

12.10 Market Studies and Contracts

12.10.1 Markets

General market data was obtained from reputable sources and is a fair and accurate representation of forecast and historical data. Forecast data is based on consensus market data and thus is a reasonable benchmark for commodity prices.

For local fuel pricing, historical data from Endeavour's supply chain function, for an existing CI operation has been reviewed and is considered appropriate for use in the feasibility study.

12.10.2 Contracts

For the outsourced service level contracts indicated in Table 19-11, the QP offers no opinion on the CAPEX and OPEX costs presented in Section 21 and used in Section 22, on the basis that this is the responsibility of the various QPs.

The QP for this section has reviewed the status of the tender and budget quotations used in the DFS and confirms that there are no key contracts currently in place[80] and the only tender received, is not yet fixed and firm.

This section does not address the agreements noted in Section 4.

12.11 Environmental Studies, Permitting and Social or Community Impact

The data presented in Section 20 was sourced from the ESIA and supporting specialist reports developed by Cabinet ENVAL. According to the ESIA secondary and primary data collection was undertaken from 2019 to early 2021. The reports provided include photographic evidence of field observations to support the findings. Other credible sources such as the IUCN databases, were also consulted and referenced accordingly.

Endeavour/SML appointed Digby Wells Environmental (DWE), an international environmental and social consultancy with expertise in good international industry practice, to conduct a gap analysis and verification of the data contained in the ESIA. Several consultations were held with Enval to verify data and some gaps/discrepancies were identified. These have been addressed in a forward work programme recommended by Digby Wells Environmental to fill these gaps/discrepancies. Should this forward work programme be implemented, Digby Wells believes that limitations to the existing knowledge of the environmental and social aspects related to the Lafigué Project will be addressed.

A detailed assessment of the gaps/discrepancies identified is provided in Section 25.17, followed by the forward work programme ('Recommendations') proposed in Section 26.17.

12.12 Capital and Operating Costs

12.12.1 Capital Costs

The capital cost estimate for the Lafigue Project has been reviewed and benchmarked against historical projects and is considered appropriate for a DFS level of accuracy.

12.12.2 Operating Costs

The operating cost estimate for the Lafigue Project has been reviewed and benchmarked against historical projects and is considered appropriate for a DFS level of accuracy.

[80] Recruitment contracts may or may not carry over from construction phase.

12.13 Economic Analysis

The QP has reviewed as far as practical, the CAPEX and OPEX assumptions and data applied in the financial model. Where there are issues that the reader should be aware of, these are stated.

12.14 Adjacent Properties

Information presented herein is based on data published in the public domain only. The Qualified Person has been unable to independently verify the information presented and makes no inferences regarding its relevance to the Lafigué Project, nor does the Qualified Person infer that such information is indicative of mineralization within the Lafigué Project.

12.15 Other Relevant Data and Information

12.15.1 Human Resources

In accordance with the framework of information developed/presented, the QP for Section 24.1 has reviewed the data available in the context in which it was provided, and is the opinion that it is appropriate for use in the DFS. No comment is offered with respect to what other information should have been presented.

12.15.2 Project Schedule

The project implementation schedule developed for the DFS has been reviewed and benchmarked against similar recent projects and is considered realistic and appropriate for a DFS.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Overview

Metallurgical and comminution testwork has been undertaken for the Lafigué Project (previously name the Fetekro Project) on representative samples sourced from the expected minable pit based initially on the July 2019 mineral resource estimate, with subsequent additional testing of the significant down dip resource extension from the 2020 drilling programme.

Each testwork programme achieved similar metallurgical and comminution outcomes across the range of samples, with both weathered oxide and fresh primary ores showing high proportions of gravity gold and high gold leach extractions from the gravity tail, with low to moderate reagent requirements. The fresh ores were extremely competent with high breakage energy requirements for the coarse particles and moderately high fine grinding energy demands.

The 2020 drilling programme almost doubled the preliminary 2019 mineral resource estimate. The 2020 resource extension had not been finalised or the potential pit outline delineated when the testwork samples were selected as infill drilling was still in progress. As a result, a few samples are outside the defined pit, but the mineralisation style is very consistent in the main ore zone, so these are still regarded as being representative. The resource remains open at depth and exploration is ongoing with potential targets having been demarcated.

13.2 Historical Testwork Development

The first scouting metallurgical testwork programme in 2018 indicated that the gold was free milling with very high gravity/leach extractions. Subsequent to this, two sets of metallurgical and comminution testwork programmes were undertaken:

- The 2019 testwork programme was based on the successful outcomes from the scouting work, testing a wide range of metallurgical and comminution variability and composite samples. Sampling was based on the September 2019 resource definition.
- The 2021 programme of testing focussed on samples selected from the 2020 resource extension drilling programme. The 2021 programme aimed to confirm comminution parameters and metallurgical performance established in the previous work.
- The 2021 testwork focussed on metallurgical and comminution variability since all indications from the 2018 and 2019 programmes suggested that the ore was relatively homogeneous with high gravity recoverable gold content and a free milling gravity tail. This programme of testing was focussed on sampling from the 2020 resource extension drilling.

Additionally, the comminution circuit selected during the Lafigué Pre-feasibility Study (PFS) included high pressure grinding rolls (HPGR). Independent testwork to determine HPGR characteristic parameters for process and engineering design and selection of the associated equipment for the comminution circuit was conducted in parallel with the 2021 testwork programme.

Endeavour's site geologists selected all samples for the metallurgical testwork programmes to ensure representivity of the gold and sulphide mineralised zones, as well as the different host lithologies. Lycopodium Minerals Pty Ltd (Lycopodium) conducted a site visit to review the sample selections and understand the basis for their selection in the context of the ore mineralisation, available core, and geological setting.

The 2019 and 2021 testwork programmes were prepared to a level commensurate with that required for a Feasibility Study and in accordance with the 'Canadian Institute of Mining, Metallurgy and Petroleum' (CIM) Best Practice guidelines[81]. The programmes incorporate testwork to confirm the ore's amenability to treatment in the plant flowsheet selected for the PFS/DFS and to define gold recoveries and operating cost inputs for economic evaluation of the resource and Mining Reserve definition. All the necessary engineering design parameters required to facilitate detailed process plant design were determined in the metallurgical and HPGR testing programmes.

The testwork programmes were carried out by ALS (ALS Metallurgy, Perth, Western Australia) under the direction of Lycopodium. Analysis of the SMC comminution test results was completed by JKTech (JK Tech Pty Ltd, Queensland). HPGR testing was conducted by Köppern in Germany. Thickening testwork was carried out by appropriate equipment vendors, Outotec and GBL (GBL Process Pty Ltd, Perth, West Australia) on slurry samples prepared by ALS.

The testwork programmes have been reported by the ALS in the following laboratory testwork reports:

- ALS Report No. A19052, September 2018 (ALS, 2018) - Scouting Testwork.
- ALS Report No. A20279, February 2020 (ALS, 2020) - Detailed Testwork.
- ALS Report No. A21932, July 2021 (ALS 2021b) - Detailed Testwork.

The historical testwork results have been interpreted and with detailed reporting by Lycopodium in the following report:

- Lycopodium Report No. 2094-GREP-003, Fetekro Gold Project Definitive Metallurgical Investigation, October 2020.

This section summarises the metallurgical testwork on which the DFS process design has been based, further information is presented in Section 6 of the DFS report (Lycopodium, 2022).

13.3 Lafigué Site Characteristics

13.3.1 Geological Setting

Since mid-2017, Endeavour has carried out an extensive drilling programme on the northern boundary of the Fetekro exploration permit, focussing primarily on the Lafigué target where a large, mineralised vein system was defined over an area 2.5 km long by 0.6 km wide.

The Lafigué deposit consists of a network of mineralised high-grade shear zones. Mineralisation is mainly hosted by a network of quartz veins. Visible gold can be observed in these veins. The lodes generally occur on lithological or structural discontinuities, typically at the edges of the granodiorite intrusive or re-opening of early quartz-carbonate veins. The mineralised lodes show thicknesses up to 40 m.

Gold mineralisation is associated with a series of stacked gently S-dipping mineralised lenses of hydrothermally altered lithologies in a brittle-ductile shear zone slightly dipping to the south or south-southeast. Grades of more than 30 g/t Au are common locally. The bulk of the gold mineralisation comprises discrete high-grade intercepts.

[81] https://mrmr.cim.org/en/best-practices/mineral-processing/

The extended 2020 Lafigué resource added further mineralised material extending the resource width but mainly down dip, with the deposit remaining open at depth and along strike. Pit optimisation combined the previous ore zones into a single main pit with two minor satellite pits (2.7% and 0.3% of the ore tonnes). More detail on the resource model, mineral reserve estimation and mine scheduling can be found in Sections 14, 15 and 16 of this Report. As indicated in Table 13-1, the majority of the in-pit ore (94.4%) is associated with fresh rock, and this hosts over 95% of the contained gold.

Table 13-1: DFS Design Pit Inventory by Weathering

Weathered State	Total (kt)	Waste (kt)	Ore (kt)	Grade (g/t Au)
Oxide	20 920	20 178	742	1.18
Transitional	32 230	30 260	1969	1.36
Fresh	424 419	378 569	45 850	1.72
Total	**477 568**	**429 007**	**48 561**	**1.70**

As shown in plan and typical cross sections, Figure 13-1 to Figure 13-4, extension drilling confirmed the previous geological interpretation of the mineralisation and intersected additional gold zones sub-parallel to the main gold-bearing structure. Drillhole intercepts are calculated using a minimum composite grade of 0.5 g/t Au, a minimum composite length of 2 m, a cut-off grade of 0.5 g/t Au, and a maximum internal dilution length of 1 m.

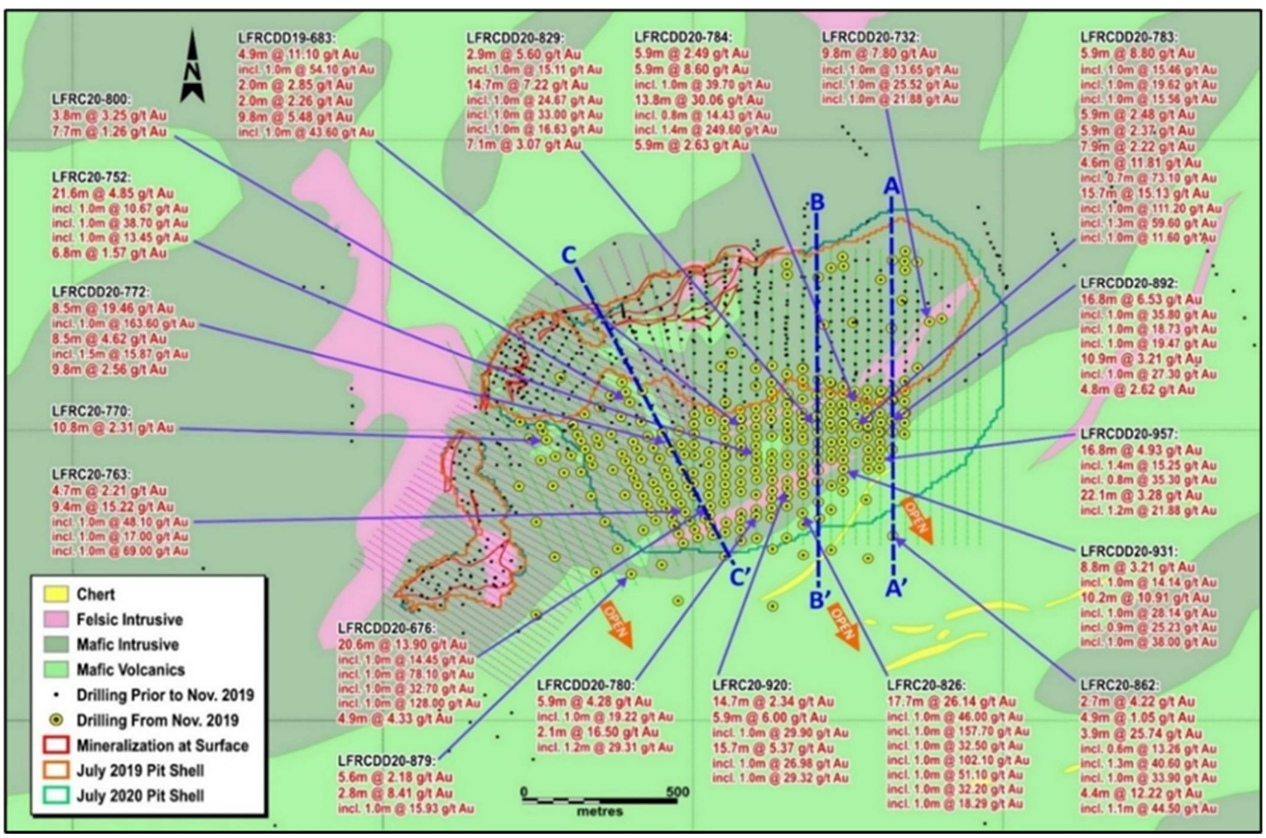


Figure 13-1: Lafigué Plan View Showing Geological Interpretation, New Drilling and Proposed Pit Extension

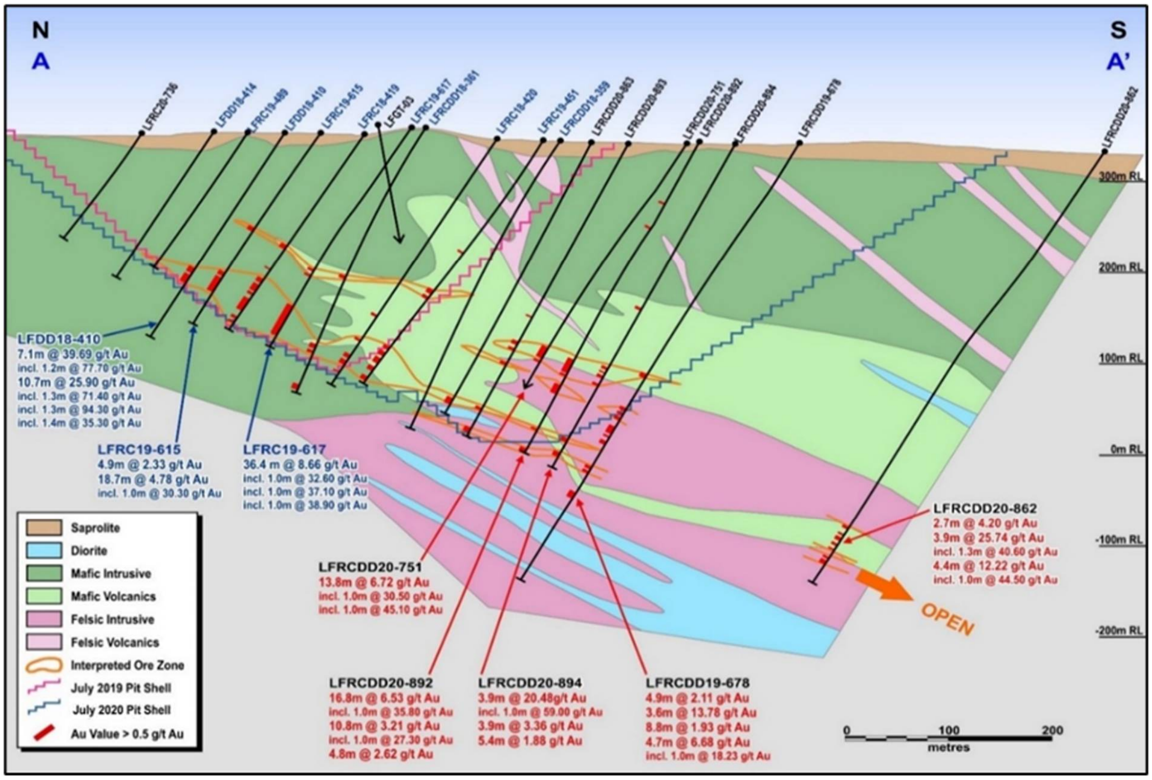


Figure 13-2: Section A-A', 320645E Lafigué Main

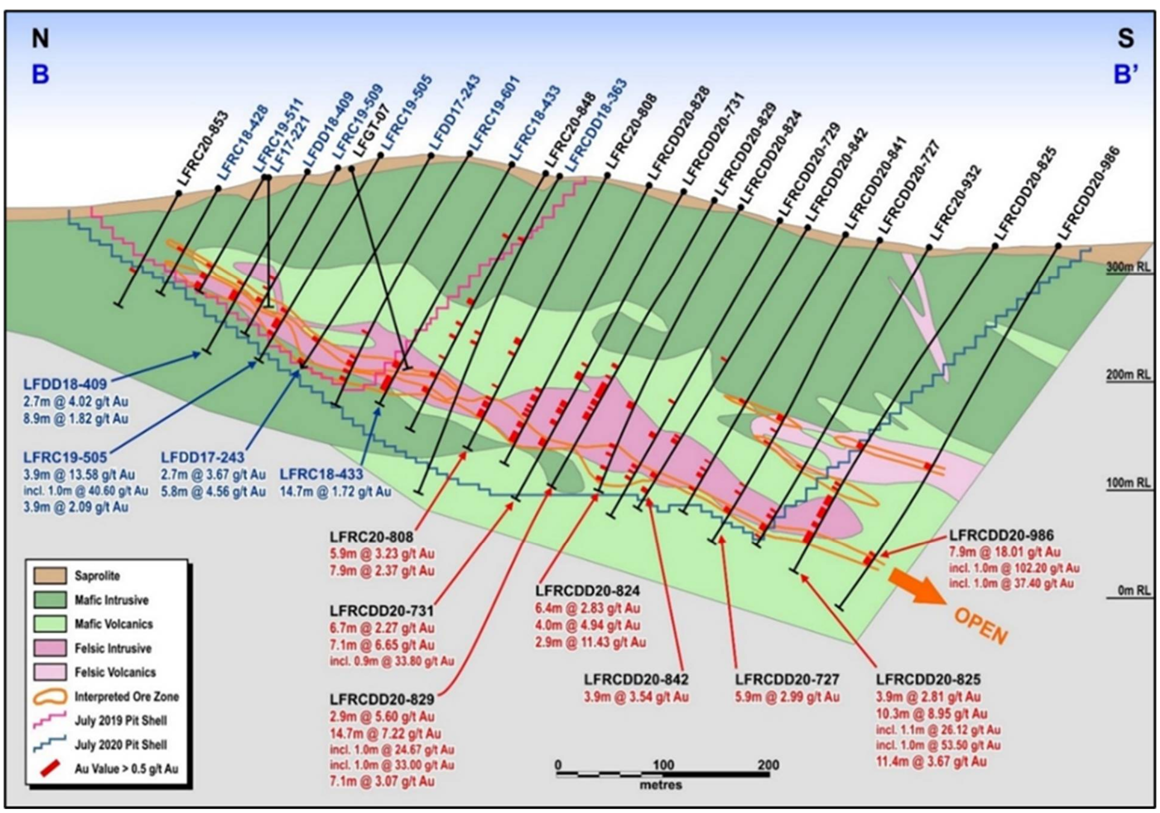


Figure 13-3: Section B-B', 320405E Lafigué Main

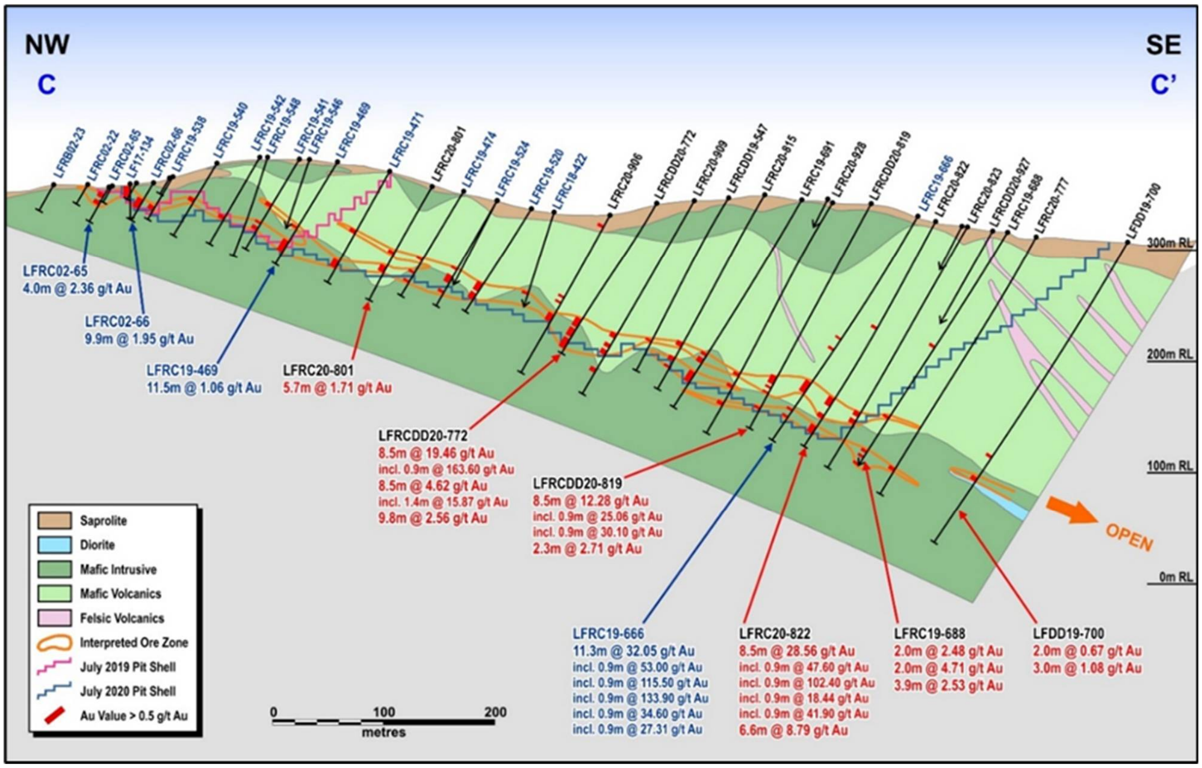


Figure 13-4: Section C-C', 319875E Lafigué Main

13.3.2 Site Water

Whilst metallurgical testwork was conducted using Perth tap water, the quality of water on site is expected to be largely similar and would not materially affect the metallurgical testwork results presented herein. Process water make-up will be primarily sourced from site water run-off. Sampling during the wet season was somewhat affected by recent artisanal mining activity, but apart from high TSS values, testing confirmed that water quality drawn from the settlement ponds would likely be very good. Qualities from local bores were also good with low TDS values and slightly alkaline pH.

13.4 Lafigué Testwork Sample Selection

13.4.1 Sample Selection Summary

Three testwork phases were conducted in parallel with exploration advances over time. All samples tested were selected from the main orebody assuming that mineralisation in the small satellite pits would be similar, being extensions of the same original orebody.

In total, 85 metallurgical variability sample intercepts were selected for metallurgical testing with 20 additional samples for comminution testing. The majority of the samples selected were from the fresh, unweathered ore since this constitutes the bulk of the resource (Table 13-1). Ten of the metallurgical samples were selected from the near surface weathered oxide zone to confirm gold recoveries for this material, but mainly for characterisation of the ore and slurry. No specific transitional ore samples were selected, but some of the oxide and fresh intercepts displayed transitional character. Samples were selected to represent mineable widths and included appropriate dilution. A large number of additional fresh ore intercepts were selected to make up three specific lithology composites for independent HPGR testing.

13.4.1.1 Ore Domains and Host Lithologies

Initial sampling focused on coverage of mineralised intercepts by depth and geographical location and generated a broad range of gold grades. Sampling for the third and final testwork programme attempted to represent mineralised domains and lithological hosts that had been defined. In practice, however, the domains outlined were simply geometrical delineations to facilitate the resource estimation, with no relevance to mineralogy or grade differences.

The major host lithologies defined are mafic volcanic (VMAF) country rock with felsic (IFEL) and mafic (IMAF) intrusives. Approximate percentages of the in-pit ore reserves by host lithology are 19%, 34% and 37% respectively. Testing of lithology composites indicated some minor physical differences between the hosts such as the IFEL examples being typically more abrasive and requiring slightly higher comminution energy.

Differences between host lithologies are of little consequence for processing, however, given the nature of the gold mineralisation. Much of the gold mineralisation occurs at or near the contact between lithologies, such that ore parcels would generally contain a mix of adjoining lithologies. Samples that are composites across several metres of mineralised material received the classification of the dominant lithology, but could contain a number of hosts.

A further feature of the contact zones is increased alteration. The resultant changes to the rock structure, often associated with intense silicification result in the most competent rock types, and it is these features that motivated selection of a HPGR comminution flowsheet to improve energy efficiency, and not the average host lithology characteristics.

Pyrrhotite and pyrite are ubiquitous throughout the deposit and occur in all lithological facies. Sulphides are generally finally disseminated in the foliation planes in the VMAF and IMAF lithologies (ductile deformation). In the IFEL, sulphides are generally associated with the quartz veinlets and occur with a patchier form due to the brittle deformation style, but they can be disseminated closer to the contact zones.

Sphalerite and galena have been rarely observed in the core (<10 occurrences) and it was only considered possible (and worthwhile) targeting areas with high pyrrhotite or pyrite for sampling, but no other specific sulphide content.

Although pyrrhotite association was considered a likely cause of slower gold leaching characteristics in occasional samples, most of the pyrite and pyrrhotite is not gold containing but occurs in similar zones.

Gold extraction testing did not identify any differences between host lithologies with rare observed variability being highly localised and likely to be caused by less common mineralisation features occurring on a microscopic scale. This mineralisation was not observable by field geologists on review of specific samples and is unlikely to be identified by grade control sampling and assays. Fortunately, the gold mineralisation is generally consistent across the resource with large fractions of readily liberated, coarse gravity gold and free milling gravity tails resulting in high (>97%) overall gold extractions.

The demonstrated ore characteristics and relative homogeneity of the resource allow all samples selected to be considered equally representative while discounting ore lithologies and considering only differences in weathering.

13.4.2 2018 and 2019 Testwork Sample Selection

13.4.2.1 2018 Scouting Programme

A scouting metallurgical testwork programme (October 2018) was undertaken to determine basic metallurgical performance and preliminary gold recoveries for the Lafigué ores. Additional programmes of BLEG/LeachWELL testing on a large number of exploration drill intercepts had already indicated good agreement with fire assays confirming that the gold is essentially all free milling.

For the scouting testwork ten metallurgical variability samples and two samples for comminution testing were selected from the Lafigué prospect by the site geologists. The samples provided examples of the oxide and fresh weathered states and different host lithologies for comminution testing.

Testwork sample details are summarised in Table 13-2.

Table 13-2: 2018 Testwork Sample Details

Composite Type	Composite ID	Hole ID	Depth (m)	Facies	Site Assay Avg. Grade (g/t Au)
Variability	VC#1	LFDD17-232	60 to 67.7	Fresh LG	1.32
	VC#2	LFDD17-232	150.64 to 157.5	Fresh HG	19.7
	VC#3	LFDD17-244	56.40 to 73.4	Fresh LG	0.77
	VC#4	LFDD17-244	76.20 to 98.00	Fresh HG	5.13
	VC#5	LFDD17-246	19.00 to 31.00	Fresh LG	3.14
	VC#6	LFDD17-247	0.00 to 12.75	Oxide	0.81
	VC#7	LFDD18-250	52.70 to 59.40	Fresh LG	0.69
	VC#8	LFDD18-251	79.00 to 91.00	Fresh LG	1.27
	VC#9	LFDD18-401	0.00 to 12.45	Oxide	1.54
	VC#10	LFDD18-402	0.00 to 8.65	Oxide	2.91
Comminution	CC#1	LFDD17-244	21.55 to 41.0	Mafic with Quartz vein	
	CC#2	LFDD17-232	112.90 to 135.2	Felsic	

13.4.2.2 2019 Testwork Programme Outline

The 2019 testwork programme tested a wider range of variability samples to provide improved coverage of the orebody and mineralisation styles. More in depth testing to establish engineering parameters for processing plant design was also included in the programme. Historical testing had demonstrated a high degree of consistency between samples in terms of metallurgical performance, so the testwork programme could be planned with a high degree of confidence.

Lycopodium determined the sample quantities required for the planned testwork programme including comminution testing, master composite optimisation testing and variability testwork.

Endeavour's exploration geologists selected appropriate core samples from the various ore lithologies, weathered states and mineralisation styles based on core inspection and downhole assay data. The mineralised intercepts selected were reviewed by Lycopodium and adjusted to cover appropriate mineable widths, include likely dilution and be more representative of expected life of mine (LOM) average grades. Specific samples with inclusion of pyrrhotite and pyrite were selected to check if these species impacted gold extraction.

Comminution samples aimed to mainly cover the typical lithological distribution and alteration around the major contact zones where the gold mineralisation typically occurs (Figure 13-2 to Figure 13-4.) Some lower grade intercepts were used for comminution samples (since the broader gold mineralised intercepts served more purpose in the metallurgical suite of samples), but similar alteration of the host rocks was evident in these cases. Examples of individual host lithologies, mafic volcanics and intrusives as well as the felsic intrusive and adjoining country rock types were also sampled for comminution testing.

The overall sample selection process aimed to ensure representivity across all major ore lithologies, such that the testwork programme results could be used for process selection and economic evaluation of the resource.

13.4.2.3 Sample Selection for 2019 Programme

Thirty-six samples including seven oxide/transition examples were selected by the site geologists for metallurgical testwork along with twelve samples for comminution testing. Samples selected were contiguous lengths of either quarter or half diamond core with drilling diameters varying with hole depth. Met comp #17 was made up by combining two near contiguous intercepts to increase mass, reducing the total number of metallurgical test composites to 35.

The samples covered the range of gold mineralisation styles with high grade coarse gold intercepts as well as disseminated low grade examples. Examples of the oxide and fresh weathered states and different host lithologies were included. The drillholes sampled provided good geographical coverage of the defined ore resource.

The known high gravity gold content was expected to result in a high degree of assay variability. Sample assays prior to composite make-up were the averages for the intercepts based on the recorded assays for the alternate corresponding core halves. As a result, these were occasionally quite different to the actual variability composite assays.

The comminution samples are listed in Table 13-3 and the metallurgical test samples in Table 13-4. No oxide samples were suitable for comminution testing, as the material is too fine for breakage or work index testing. Comm comp #3 was a low mass intercept selected as a BWI variability sample. Meterage representing a single lithology was limited in the mineralised zones being at or near the contact between lithological hosts. Corresponding test borehole locations are represented graphically in Figure 13-5.

Table 13-3: Comminution Test Samples

Sample ID	Hole ID	From (m)	To (m)	Mass (kg)	Lithology
Comm #1	LFDD18-250	25	52.7	27.3	IMAF with intervals VMAF + QZ
Comm #2	LFDD18-400	57.3	80.25	21.6	VMAF
Comm #3	LFDD18-400	87.4	97.9	10.45	IMAF
Comm #4	LFDD18-400	118.9	125.95	20.3	VMAF
Comm #5	LFDD18-403	110.2	140.95	29.85	VMAF + IFEL +QZVN
Comm #6	LFDD18-407	168.5	189.8	24.5	contact VMAF/IMAF
Comm #7	LFDD18-410	35.3	70.5	32.1	sheared Gabbro interleaved with QV
Comm #8	LFRCDD18-358	198.4	220	25.1	IMAF + VMAF + QZ + IFEL
Comm #9	LFRCDD18-358	279.2	308.95	38.3	VMAF + IFEL + IMAF
Comm #10	LFRCDD19-547	159.45	176.8	21.9	IMAF + VMAF intervals
Comm #11	LFRCDD19-667	66	77	44.3	IFEL
Comm #12	LFRCDD19-667	110	117	28.7	IMAF

Table 13-4: Metallurgical Test Samples

Comp ID	Hole ID	From (m)	To (m)	Mass (kg)	Au (g/t)
Met#1	LFDD17-233	152	162.6	9.44	10.3
Met#2	LFDD17-243	199	205.4	6.60	1.8
Met#3	LFDD17-243	224	233.25	8.94	3.0
Met#4	LFDD18-252	82.15	90	6.82	1.4
Met#5	LFDD18-400	80.25	87.4	6.60	0.9
Met#6	LFDD18-400	97.9	106.95	8.70	1.2
Met#7	LFDD18-404	100.8	107.65	6.40	3.8
Met#8	LFDD18-404	111.2	115.7	4.28	0.9
Met#9	LFDD18-405	164	169.7	5.84	3.9
Met#10	LFDD18-409	115.1	126.85	11.48	3.1
Met#11	LFDD18-409	126.9	138.05	10.94	2.2
Met#12 [HG	LFDD18-410	173.8	182.1	7.54	29.8
Met#13 [HG]	LFDD18-410	182.1	193	10.26	24.4
Met#14	LFDD18-412	342.6	351	8.46	1.0
Met#15	LFRCDD18-359	300.4	304.75	4.56	7.2
Met#16	LFRCDD18-361	258.8	270.6	12.50	3.8
Met#17	LFRCDD18-361	270.6	272.7	8.46	1.1
	LFRCDD18-361	277.7	282.95		
Met#18	LFRCDD18-361	305.4	315.4	11.52	1.7

Table 13-4: Metallurgical Test Samples

Comp ID	Hole ID	From (m)	To (m)	Mass (kg)	Au (g/t)
Met#19	LFRCDD18-362	266.6	273.2	7.48	0.8
Met#20	LFRCDD19-441	245	252.72	6.52	5.2
Met#21	LFRCDD19-459	181.4	190	9.90	3.2
Met#22 [Oxide]	LFDD19-670	0	7.3	17.30	22.2
Met#23 [Oxide]	LFDD19-670	7.3	10.4	12.58	2.5
Met#24 [Oxide]	LFDD19-671	0.9	7.7	16.22	1.5
Met#25 [Oxide]	LFDD19-668	5.45	11.3	17.22	2.9
Met#26 [Oxide]	LFDD19-669	3.4	6.7	15.12	0.9
Met#27 [Oxide]	LFDD19-669	6.7	12.3	14.50	3.5
Met#28 [Oxide]	LFDD19-669	13.75	17.2	16.52	0.6
Met#29	LFRCDD19-667	62	66	15.78	0.8
Met#30	LFRCDD19-667	77	83.45	26.42	0.9
Met#31	LFRCDD19-667	83.45	90	26.04	1.9
Met#32	LFRCDD19-667	90	95	19.96	1.8
Met#33	LFRCDD19-667	95	99	15.56	8.9
Met#34	LFRCDD19-667	99	103.5	16.98	1.9
Met#35	LFRCDD19-667	103.5	109.25	22.78	3.7

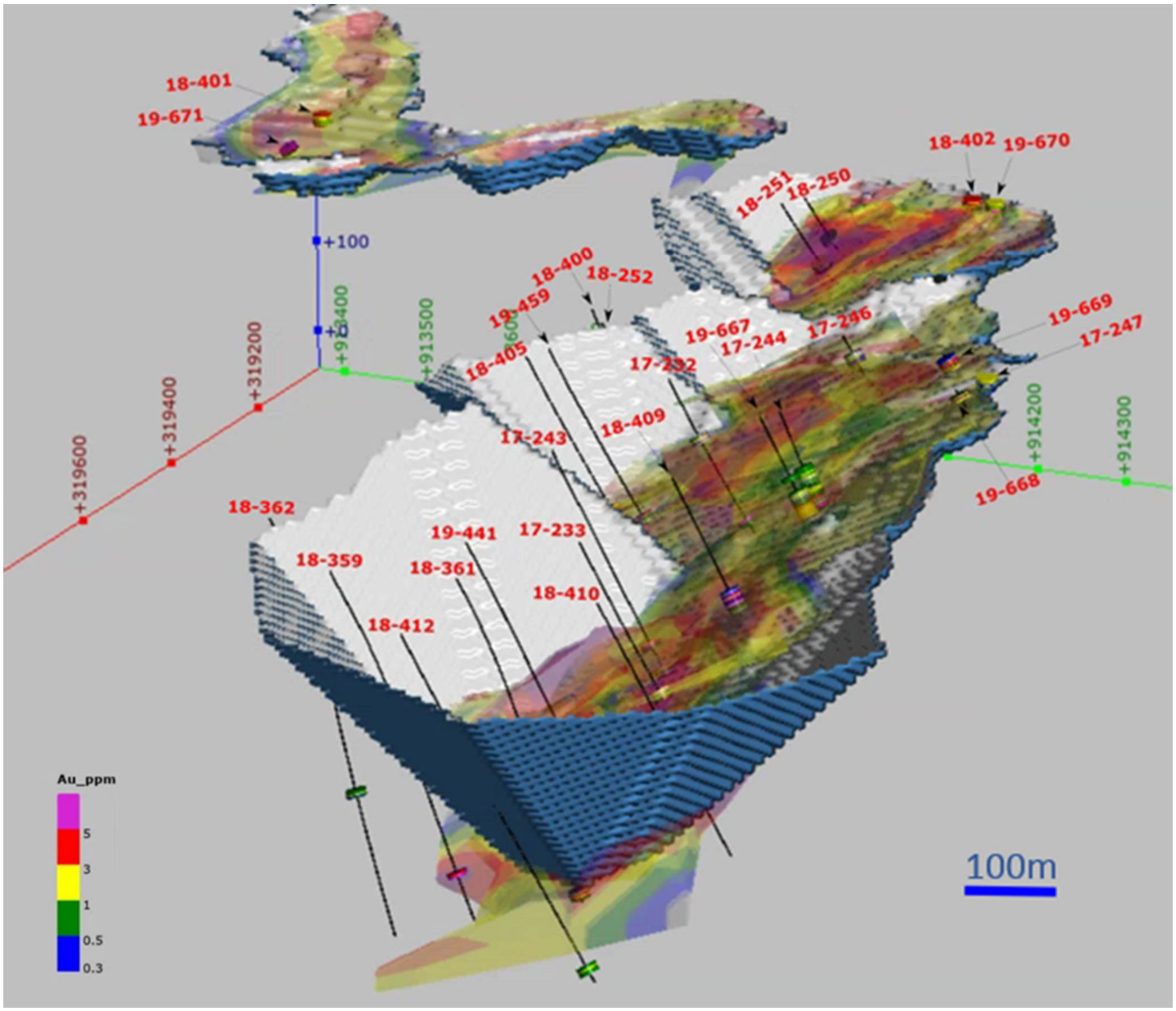


Figure 13-5: 2019 Lafigué Prospect Oblique View from the Northeast

Figure 13-5 note: Model shows metallurgical sample hole locations, preliminary pit shell and mineralisation

13.4.2.4 2019 Master Composite Make-up

Having observed the consistent metallurgical response from the variability intercepts in the preliminary testing phase and similarly high gold extractions from the various BLEG testing programmes, it was decided that all samples from each weathered state could be considered equally representative and combined to form the master composites. A high-grade composite was set aside to allow the combined fresh intervals to better align with the average resource grade.

Equal sub-sample masses from the intercept composites were used to make up Lafigué fresh and oxide master composite with expected Au grades of 2.75 g/t and 4.9 g/t respectively based on the averaged site assays.

A few intercept composites were not included in the fresh master composite having too low a mass as supplied. These were used for variability testing only.

The Lafigué high grade composite had an expected Au grade of 27.1 g/t, based on the averaged site assays.

Differences between the composite gold assays and expected values were anticipated given the high assay variability associated with the 'spotty' gold mineralisation, in addition to the errors inherent in averaging the assays across the intercepts. The expected gold assay values were estimated only to provide an indication of the order of magnitude for sample selection purposes.

13.4.3 2021 Testwork Sample Selection

13.4.3.1 2021 Testwork Programme Outline

The purpose of the 2021 Lafigué metallurgical testwork programme was to confirm the suitability of the processing route selected during the PFS for the additional indicated resource tonnes defined by the further exploration and in-fill drilling. A significant flowsheet change from the scoping phase was to include HPGR comminution. This required specific testing to characterise the engineering design parameters for this unit operation. New sets of comminution and variability composite samples were sourced from drill cores within the Lafigué resource extension, with additional drilling for HPGR test composites. The 2021 testwork programmes built upon and supplemented the previously conducted testwork programmes.

The 2021 testwork programme included comminution testing based on six lithology samples, a detailed metallurgical programme based on 40 variability samples and a master composite sample. Two additional composite samples were subsequently made up for a specific investigation. Köppern defined and conducted HPGR testing on a further three lithology samples (IFEL, IMAF and VMAF) and a mixed sample comprising equal fractions of these three samples.

The HPGR testing required 3 tonnes of sample (1 t per lithology composite). The narrow ore zones and small core diameters (¼ NQ core) available at depth implied that a large number of intercepts were required for each composite to provide adequate mass. The large number of intercepts in each case ensured good geographical coverage of the resource for this sampling. All samples were sourced from the 2020 drilling for the resource extension.

13.4.3.2 2021 Sample Selection Approach

Endeavour's original sampling intent was to source samples representative of each of the mineralised domains defined across the full deposit with coverage of the weathered states and grade ranges in each domain. Twenty-two geological domains were defined for the resource block model. However, geology advice was that these domains were geometrical representations to facilitate the resource estimation only, and that from a statistical point of view there are no significant differences between the 22 domains. Testwork samples were therefore categorised based on host lithology and represented a range of mineralisation styles and gold grades. The major host lithologies defined are VMAF (background country rock mafic volcanics, frequently altered near the lithology contacts) with IFEL (felsic intrusive, mainly granodiorite) and IMAF (mafic intrusive, typically fine-grained gabbro and granite). Fine quartz veining is common in the mineralised zones.

All samples were fresh unweathered rock from the Lafigué Main resource, since the orebody extension was at depth with the gold mineralisation occurring almost exclusively at the base of the proposed pit. No further oxidised resources were identified at this stage.

Sample selection was from available cores to provide intervals spanning likely mining widths to represent the spatial distribution, the gold grade ranges and the lithological hosts. Other factors considered were spatial distribution within the domain, approximated pit shell boundaries and material availability. One to two metres of barren material was generally added to the top and bottom of each interval to reflect typical waste dilution. Sulphide mineral association was coincidental as only rare sulphide assays were determined for the cores.

Quartz (QTZ) composites were added as a lithological classification for sampling purposes to reflect mixed material made of quartz and host rock (VMAF and/or IMAF, IFEL more rarely). The quartz zones are quite discontinuous and are considered more an ore type than a lithological unit.

The samples for the 2021 testwork programme included six comminution samples and forty metallurgical variability samples as selected by Endeavour's geologists. Selected metallurgical variability samples were used to make-up the master composite sample for comprehensive metallurgical and physical testing. The comminution samples are listed in Table 13-5, with the metallurgical test samples in Table 13-6.

13.4.3.3 2021 Composite Sample Make-up

Endeavour directed that the master composite should be made up from equal contributions from each of the 40 variability samples, excluding the slow leaching samples 12, 15, 20 and 27 (these samples had elevated sulphide assays). Separate testing was conducted on these slow leaching variability samples to assess the options for improving leach kinetics and gold recoveries. This master composite sample was subsequently termed the Average Grade Master Composite.

Subsequent review of the variability testing results indicated that further samples demonstrated slow leaching characteristics, but this had been masked by high gravity recoveries. No clear explanation for the slow leaching was evident, but it was assumed that pyrrhotite (Po) association was likely to be a dominant factor in this behaviour. Two further composites were made up to help better understand the slow leaching behaviour and possible mitigation measures. These two composites were named the P and O composites.

The slow leaching samples (five variability samples per composite) were differentiated by oxygen uptake ahead of leaching. The P composite set of variability samples displayed low levels of dissolved oxygen ahead of cyanide leaching while the O composite variability samples set displayed normal dissolved oxygen levels. The P composite comprised equal masses of samples MET 6, 9, 10, 12, 15 while the O composite comprised equal masses of samples MET 1, 2, 4, 18, 29. The slow leaching examples were distributed across all the lithological hosts. Note that the slow leaching behaviour observed rarely affected overall gold extraction significantly and is mostly of concern in terms of maintaining 'ideal' carbon adsorption and leach solution profiles.

Table 13-5: Comminution Test Samples

Composite ID	Drill Hole ID	From (m)	To (m)	Mass (kg)	Lithology
COM 1	LFRCDD20-896	322.9	352.4	32.9	IFEL
COM 2	LFRCDD19-678	357.5	383.0	28.4	IFEL
COM 3	LFRCDD20-764	175.1	184.4	25.7	IMAF
COM 4	LFRCDD19-683	203.2	208.8	30.7	IMAF
COM 5	LFRCDD20-720	212.0	231.0	33.9	VMAF
COM 6	LFRCDD20-720	212.0	231.0	33.9	VMAF

Lithologies in Table 13-6 have the added gold grade range designation (high, medium, low) to indicate the spread of grade coverage.

Table 13-6: Metallurgical Test Samples

Composite ID	Drill Hole ID	From (m)	To (m)	Mass (kg)	Lithology
MET 1	LFRCDD20-731	262.0	271.2	9.7	IFEL_HG
MET 2	LFRCDD20-965	368.1	380.0	11.2	IFEL_HG
MET 3	LFRCDD20-726	286.0	294.9	8.9	IFEL_LG
MET 4	LFRCDD20-735	319.4	327.4	10.0	IFEL_LG
MET 5	LFRCDD20-896	365.9	375.0	9.0	IFEL_LG
MET 6	LFRCDD20-892	296.0	305.0	10.3	IFEL_LG
MET 7	LFRCDD20-828	230.0	239.0	10.7	IFEL_MG
MET 8	LFRCDD20-901	320.3	328.8	10.4	IFEL_MG
MET 9	LFRCDD20-745	303.0	313.5	12.2	IFEL_MG
MET 10	LFRCDD20-956	377.0	389.0	12.3	IFEL_MG
MET 11	LFRCDD19-677	227.0	235.0	9.4	IMAF_HG
MET 12	LFRCDD20-722	130.0	139.0	11.0	IMAF_HG
MET 13	LFRCDD20-724	247.0	252.0	9.2	IMAF_HG
MET 14	LFRCDD19-711	133.3	142.2	10.3	IMAF_LG
MET 15	LFRCDD20-768	115.6	120.9	11.4	IMAF_LG
MET 16	LFRCDD20-724	234.0	243.0	10.1	IMAF_LG
MET 17	LFRCDD20-819	205.0	212.6	10.8	IMAF_LG
MET 18	LFRCDD19-676	222.0	228.0	10.7	IMAF_MG
MET 19	LFRCDD20-948	255.8	266.0	14.4	IMAF_MG
MET 20	LFRCDD20-772	145.0	153.7	11.0	IMAF_MG
MET 20	LFRCDD20-772	145.0	153.7	11.0	IMAF_MG
MET 21	LFRCDD20-720	255.0	263.2	9.7	QTZ_HG
MET 22	LFRCDD19-681	271.7	280.4	10.5	QTZ_HG
MET 23	LFRCDD20-784	349.6	359.1	11.0	QTZ_HG
MET 24	LFRCDD20-931	326.1	337.5	10.6	QTZ_HG
MET 25	LFRCDD20-820	314.5	321.0	11.1	QTZ_HG
MET 26	LFRCDD20-837	250.2	256.3	10.0	QTZ_HG
MET 27	LFRCDD20-940	169.7	181.0	9.9	QTZ_LG
MET 28	LFRCDD19-699	256.5	265.6	10.2	QTZ_LG
MET 29	LFRCDD20-886	219.3	224.4	10.4	QTZ_LG
MET 30	LFRCDD20-890	228.6	239.7	11.1	QTZ_MG
MET 31	LFRCDD20-873	265.2	275.6	10.3	VMAF_HG
MET 32	LFRCDD20-783	343.0	352.5	11.6	VMAF_HG
MET 33	LFRCDD20-950	326.0	336.5	11.3	VMAF_HG
MET 34	LFRCDD20-824	280.6	285.7	9.5	VMAF_HG

Table 13-6: Metallurgical Test Samples

Composite ID	Drill Hole ID	From (m)	To (m)	Mass (kg)	Lithology
MET 35	LFRCDD20-866	318.9	329.5	10.7	VMAF_LG
MET 36	LFRCDD20-871	237.8	247.9	10.4	VMAF_LG
MET 37	LFRCDD20-725	225.6	233.9	9.1	VMAF_MG
MET 38	LFRCDD20-825	314.0	324.0	10.7	VMAF_MG
MET 39	LFDD18-403	198.0	203.7	11.0	VMAF_MG
MET 40	LFRCDD20-972	311.4	322.7	9.9	VMAF_MG

All testwork programmes were undertaken at the ALS Metallurgical Laboratory, Perth, Western Australia (ALS) under the supervision of Lycopodium Minerals Pty Ltd (Lycopodium). Comminution SMC test results were analysed by JKTech Pty Ltd, Queensland (JKTech) and High-Pressure Grinding Roll (HPGR) testing was performed by Köppern Aufbereitungstechnik GmbH & Co. KG, Freiburg, Germany (Köppern). Thickening testwork was carried out by Metso Outotec Australia Ltd, Perth, Western Australia (MO Group).

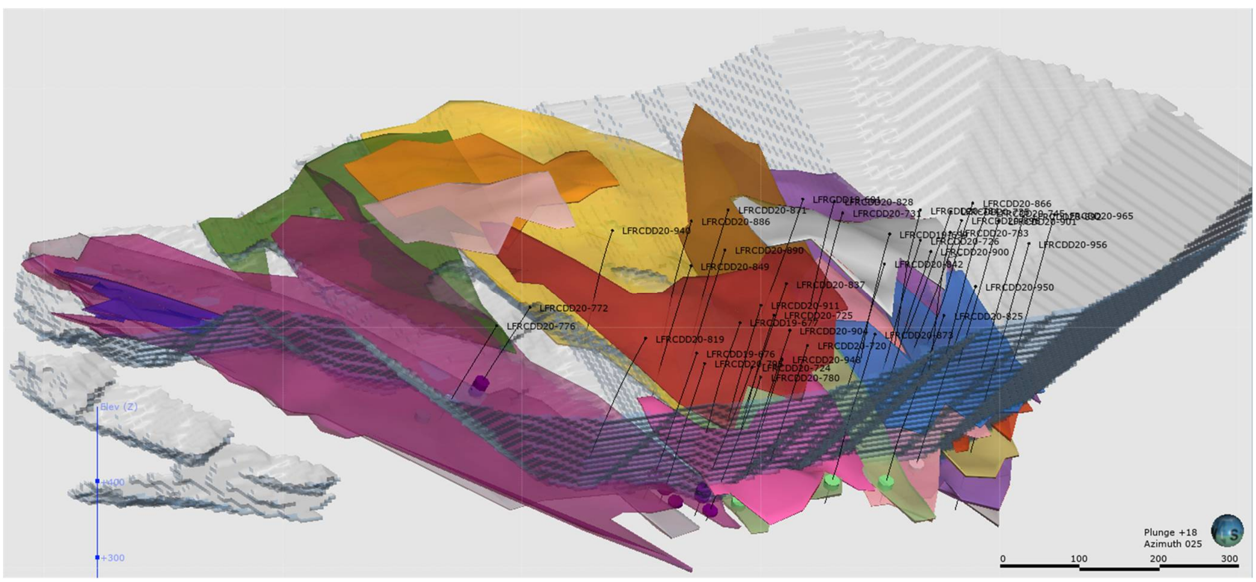


Figure 13-6: 2021 Metallurgical Sample Locations Against the Pit Outline Showing Ore Domains

Detailed plots of metallurgical sample locations by domain as well as comminution sample intercept locations are included the DFS Report (Lycopodium, 2022).

13.5 Comminution Testwork

The purpose of the comminution testwork programme was to determine the ore physical characteristics to allow modelling of the grinding energy required for size reduction to facilitate a crushing and milling circuit design appropriate for the plant throughput and feed type.

SMC, abrasion index (Ai) and Bond rod and ball work index (RWi and BWi) comminution tests were completed on the fresh ore samples. No oxide samples were selected for comminution testing as the oxide ore is too fine for breakage or work index testing. No core of suitable size for UCS or crushing work index testing was available, but the former is really only suitable for geotechnical data and the variability is generally too high for processing equipment selection. Crushing work indices are calculated from the SMC parameters with good reliability.

The SMC test results generate the drop weight index (DWi) which is a measure of the energy required for coarse particle breakage. This is inversely related to the Axb parameter which is similarly derived from the SMC breakage tests based on the product size distributions from fracture tests at various energy levels. A harder ore will experience less breakage from the same applied energy and hence will have a lower Axb value.

The following comminution tests were typically completed for the comminution test samples selected:

- SMC tests (tested using the -22.4 mm to +19.0 mm size fraction).
- Bond Abrasion Index (Ai) determination.
- Bond Ball Mill Work Index (BWi) determination (@ 106 µm closing screen to yield a P_{80} of approximately 75 µm).
- Bond Rod Mill Work Index (RWi) determination (@ 1180 µm closing screen to yield a P_{80} of approximately 750 µm).

13.5.1 Comminution Testwork Results Summary

Comminution testwork was conducted as part of each of the three testwork programmes and the above standard tests were run on the HPGR composite sample. The overall comminution data obtained, indicates that the samples tested are very competent, with the 85th percentile Axb value (in terms of hardness) being 26.0 which is at the 93rd percentile of the JK database. Even the median value for the dataset is 30.7 which is at the 83rd percentile. There are a few softer outliers, particularly the mafic sample from the 2018 programme, but removing these points has little effect on the above outcomes.

The Bond ball milling work indices are moderately high, with an 85th percentile value of 16.9 kWh/t. Here the median value is 15.8 kWh/t. There are a number of lower work index samples (<14 kWh/t) that are likely away from the altered ore zones and are more country rock examples i.e., waste dilution. One significant outlier is the 19.6 kWh/t maximum BWi. Removing this from the data set would reduce the 85th percentile value to 16.0 kWh/t. This outlier was likely a highly altered sample in the lithological contact zone

Quartz veining contributes to higher work indices and also significantly higher abrasion indices. The abrasion indices are quite variable. Values range from 0.026 to 0.398 with the higher values associated with lower SGs, indicative of higher quartz content. The felsic samples all had increased quartz content, but this sample set is too small to know if this is a characteristic of this lithology.

The abrasion value selected for design is the average in this case (Ai = 0.163), since this affects only the operating cost consumables estimates. The grinding media and lining materials will be optimised during operations to find the best balance between wear life and costs.

13.5.2 Scouting Comminution Testwork (2018 Programme)

The scouting testwork considered only two comminution samples. Comminution testwork results are summarised in Table 13-7.

The low Axb (and relatively high DWi) value for Composite 2 indicates a competent ore with a high breakage energy requirement. Composite 2 is significantly more competent and has a higher ball milling work index and abrasion index than Composite 1. Composite 1 is from a shallower depth and may have been more weathered/transitional in nature.

Table 13-7: 2018 Testwork - Comminution Results

Comminution Composite	Depth (m)	Comminution Testwork Results							
		DWi kWh/m³	SG	SMC Parameters				Ai	BWi kWh/t
				A	b	Axb	ta		
CC 1 (Mafic/QV)	21.55 to 41.0	4.7	2.66	63.2	0.9	56.9	0.55	0.089	13.9
CC 2 (Felsic)	112.90 to 135.2	8.2	2.63	80	0.4	32.0	0.32	0.349	16.8

13.5.3 2019 Testwork Programme

The 2019 testwork programme built on the scouting testwork and tested 12 comminution samples representative of the 2019 resource. Composite 3 was a transition sample and was too fine for SMC testwork, so only the BWi was determined. The comminution test results for the 2019 samples are summarised in Table 13-8. IMAF and VMAF are intrusive and volcanic mafic lithologies. IFEL is the felsic intrusive. QZ and QV indicate the presence of quartz veining. As discussed in Section 13.4.2, these samples were selected to represent mineable widths in the ore zones, which typically occur at the contact between lithologies. Selected samples of individual lithologies are typically representative of the hanging wall and foot wall dilution of the ore that will occur, although there is also some gold mineralisation away from the contacts.

As shown in Table 13-8, the comminution results are reasonably consistent. The Axb values are low and the DWi values are high (both design points are >90th percentile in the JKTech database) indicating that the fresh ore is very competent with a high breakage energy requirement. The RWi and BWi are not as extreme (with the exception of Comp 5), but values are still moderately high. The abrasion index indicates low to moderate abrasivity for these ores.

Table 13-8: 2019 Testwork - Comminution Composite Test Results

Sample Designation		SMC Parameters				Ai	RWi kWh/t	BWi kWh/t
		SG	DWi kWh/m³	Axb	ta			
Comp 1	IMAF+VMAF/QZ	2.87	7.9	36.4	0.3	0.0409	-	13.6
Comp 2	VMAF	2.82	9.3	30.7	0.3	0.0831	-	15.8
Comp 3	IMAF (Transition)					-	-	16.2
Comp 4	VMAF	2.88	11.0	25.9	0.2	0.0717	-	16.6
Comp 5	VMAF+IFEL+QV	2.78	9.5	29.3	0.3	0.228	-	19.6
Comp 6	VMAF/IMAF contact	2.79	8.9	31.2	0.3	0.228	-	15.7
Comp 7	Sheared Gabbro+QV	2.75	10.4	26.1	0.3	0.2746	18.8	17.2
Comp 8	IMAF+VMAF+QZ+IFEL	2.81	8.1	34.7	0.3	0.0259	-	14.4
Comp 9	VMAF+IFEL+IMAF	2.84	8.0	35.7	0.3	0.142	20.7	16.0
Comp 10	IMAF+VMAF	2.97	10.1	29.7	0.3	0.0584	-	13.5
Comp 11	IFEL	2.72	10.6	26.0	0.3	0.3976	18.4	16.4
Comp 12	IMAF	2.83	6.9	41.2	0.4	0.035	15.7	12.4

The comminution testwork results were provided to Orway Mineral Consultants (OMC) for comminution circuit selection and equipment sizing. The selection of the comminution circuit is detailed in Section 13.5.5 and includes the key design criteria used in modelling of the comminution circuit, the equipment selected for the circuit and the comminution power and consumables estimates for use in the operating cost estimate.

13.5.4 2021 Testwork Programme

The 2021 comminution programme tested the six specific lithology samples selected and includes additional results from the Köppern HPGR testing on a further three lithology samples, and a mixed sample comprising equal fractions of these three samples. Summary comminution test results for this testwork are provided in Table 13-9 following.

The 2021 testing provided a similar range of results to the previous testwork with low Axb values. The 85th percentile value (hardness, not number) for this set is 24.8, which is above the 95th percentile of the JK database. The RWi is classified as hard and BWi (85th percentile is 16.7 kWh/t) is classified as moderately hard to hard for this data set. The comminution test samples display moderately abrasive to abrasive characteristics.

The comminution test results show that the felsic intrusive and volcanic mafic rock samples exhibit hard comminution properties, whereas the intrusive mafic rock type shows comminution properties varying from moderately hard to hard. In terms of abrasiveness, the intrusive felsic rock type displays abrasive characteristics and mafic rock type shows moderately abrasive characteristics. There is little evidence of hardness increasing with depth below the transitional zone.

Table 13-9: Comminution Composite Summary Results

Sample Designation		SMC Parameters				Ai	RWi (kWh/t)	BWi (kWh/t)
		SG	DWi (kWh/m³)	Axb	ta			
2021 Comminution Composites								
COM 1	IFEL	2.71	8.5	32.0	0.31	0.371	18.3	16.2
COM 2	IFEL	2.71	11.0	24.6	0.24	0.322	17.6	17.4
COM 3	IMAF	2.92	9.8	29.7	0.26	0.120	19.0	13.9
COM 4	IMAF	2.82	6.7	42.2	0.39	0.040	16.7	13.4
COM 5	VMAF	2.83	9.8	28.8	0.26	0.113	19.1	14.7
COM 6	VMAF	2.79	11.3	24.8	0.23	0.102	19.6	16.4
2021 Köppern HPGR Test Composites								
Sample 1	IMAF	2.90	-	-	-	-	-	13.1
Sample 2	VMAF	2.85	-	-	-	-	-	13.9
Sample 3	IFEL	2.74	-	-	-	-	-	14.5
Comp.	MIX 1	2.82	9.3	29.8	0.28	-	-	13.9
Comp.	MIX 2	2.82	8.2	34.0	0.32	-	-	13.9

13.5.5 Comminution Circuit Selection

Value engineering assessments conducted for the various comminution circuit options determined that over the life of mine, the HPGR - ball mill comminution circuit was more economically attractive than a SAG - ball mill alternative given the poor SAG energy efficiency with the competent ore. The increased HPGR circuit complexity was highlighted as a potential operability issue, but Endeavour is committed to maximising energy efficiencies where possible.

13.5.5.1 Comminution Testwork Interpretation

OMC undertook the Lafigué comminution equipment selection based on the 85th percentile ore characteristic hardness and a target grind size P_{80} range of (89 to 106) µm. The selected comminution circuit comprises primary and closed-circuit secondary crushing, an HPGR and a ball mill both operating in closed circuit. In addition to having the lowest comminution energy requirement, this circuit is less affected by variability in coarse ore competency with very consistent throughput and ball mill feed particle size distribution (PSD) being generated by the HPGR.

A summary of the key ore properties used for the comminution circuit evaluation/design is presented in Table 13-10.

Table 13-10: Ore Properties for Comminution Circuit Design

Parameter	Unit	Value	Source
CWi	kWh/t	28.4	Calculated
RWi	kWh/t	19.5	85th Percentile
BWi	kWh/t	16.9	85th Percentile
Abrasion Index	g	0.163	Average
Axb		26.0	85th Percentile
SG	t/m³	2.80	Average

OMC modelled the HPGR based on testwork results from Köppern and selected the HPGR based on product screening at 4 mm and the associated recirculating load for a design throughput of 4.0 Mt/a (db). OMC reported that the circulating load indicated by the HPGR testwork was higher than expected based on benchmarking for similar ores. It was noted that the testwork was undertaken at a low specific pressing force of 2.4 N/mm² and low single pass energy input of 1.2 kWh/t.

The ball mill was selected for treating a -4 mm feed from the HPGR and grinding to an 80% passing (P_{80}) size of 106 µm, but Endeavour requested a more conservative margin be applied to the mill selection resulting in a product grind P_{80} of 89 µm at the design conditions.

A summary of the OMC recommended equipment selections and operating power required is presented in Table 13-11.

Table 13-11: Recommended Comminution Circuit Equipment Summary

Equipment	Unit	Source
Primary Crusher		Jaw
Size		C160
Installed Power	kW	250
Gross Power	kWh/t	0.2
Secondary Crusher		Cone
Size		2 x HP6
Installed Power	kW	2 x 450
Gross Power	kWh/t	0.9
HPGR		
Size Diameter x Width		1.76 x 1.38
Installed Power	kW	2700
Gross Power	kWh/t	2.8
Ball Mill		
Size, Diameter x EGL		6.4 x 10.7
Installed Power	kW	7700
Gross Power	kWh/t	12.6
Total Gross Power	**kWh/t**	**16.5**

13.5.5.2 Recommended HPGR Design Basis

Overall, the testwork results mass balanced well, so the tested circulating load is expected to be representative of a full-scale machine operating with a pressing force of 2.5 N/mm^2, and a feed moisture content of 5% for the Lafigué ore. However, the selection of the operating conditions do not reflect typical industrial full scale circuit operating practice, and as a result required scaling for the equipment specification. A higher operating pressure, lower feed moisture and median operating speed will be expected for the equipment offered by the vendors. The testwork results were scaled accordingly on the basis of the known trends in operating data from OMC's database, assuming that the total power input remained the same as the testwork.

Table 13-12: HPGR Design Parameters

Parameter	Units	Test 13 Conditions	Scaled Design Basis
Moisture Content	%	5	2.5
Specific Pressure	N/mm^2	2.4	3.5
Specific Energy	kWh/t	1.2	1.4
Circulating Load	% New Feed	132	100
Total HPGR Feed	% New Feed	232	200
Total Power Input	kWh/t	2.8	2.8

OMC advised that for any future programmes (further testing is not required at Lafigué), the HPGR testwork should be conducted at conditions better aligned with typical operating practice. Under typical conditions, a higher input energy per pass would be anticipated along with a lower circulating load. The overall energy input (circulating load x energy per pass) is expected to be similar for equivalent competency ore. Having scaled the data for this ore as described, the circulating load is more in line with industry expectations and the equipment offered will not be significantly oversized. The associated increase in specific energy required has little impact, since the machines typically have 100% margin in their installed power to manage extremely hard particles without stalling.

13.6 Head Assay and Mineralogical Investigation

13.6.1 Head Assay and Gravity/Leach Extraction Testwork (2018 Scouting Programme)

The scouting programme undertaken in 2018 was the first metallurgical investigation on this resource with no prior knowledge other than the exploration assay investigations. The basic gravity/leach extraction test results significantly expand on the head assay data to characterise the gold mineralisation and the metallurgical processing implications for the Lafigué ores. This work served to inform the future programmes, so the full testwork results are summarised in this section. It is important to note that these were preliminary tests only and results are indicative, but not optimised.

The ten variability samples covered three major ore types from the prospect (based on the understanding at the time), oxide (three samples), low grade fresh (five samples) and high grade fresh (two samples). Gold and silver head assays for the variability samples are summarised in Table 13-13. The high gravity gold content resulted in a reasonable degree of assay variability, or it could be viewed as the variability in the triplicate gold head assays being confirmatory of the presence of significant coarse gold. It was noted that the assay data were frequently higher than the calculated heads following gold extraction, which was unusual with spotty gold, but this may have been due to the relatively small number of samples selected, with some individual high-grade assays skewing the likelihood of a positive bias.

Silver grades were generally very low (below the 2 ppm ICP detection limit) with a few exceptions where an isolated higher assay occurred. The low silver assays suggested this would be insignificant in terms of resource value, so more accurate assaying was not pursued.

Gravity/extraction testwork was conducted with the results summarised in Table 13-13 and Figure 13-7. All leach tests were conducted with a gravity stage prior to cyanidation of the gravity tail. Gold extractions were high, with typically high gravity gold recoveries, and very high overall gold extractions. Silver extractions were not monitored due to the low head grade and metal value.

For the variability samples at a P_{80} grind size of 75 µm, gravity gold recovery varied between (20 and 85%), with overall gold extractions greater than 96% for all samples. After removal of the gravity gold, leach kinetics were generally fast with the bulk of gold leached within four to eight hours. Indicative silver extractions were typically >70% Ag based on head and tail assays. Reagent requirements were low for the fresh samples and low to moderate for the oxide samples.

Three master composites (oxide, HG fresh and LG fresh) were formed from selected variability samples. Detailed head assays showed few deleterious elements for gold leaching with low levels of base metals, sulphides, and arsenic.

Preliminary evaluation of the effect of grind size on gold extraction was conducted on the master composites with the results summarised in Table 13-13 and Figure 13-7. Generally gold extraction increased as grind size decreased, however extractions were only slightly higher at a grind size of P_{80} 75 µm than at P_{80} 106 µm. For the tests undertaken, the optimum economic grind appeared to be a P_{80} of 106 µm, with similar gold recoveries and lower operating costs when compared with a P_{80} of 75 µm.

Table 13-13: 2018 Testwork – Head Assay and Gravity/Leach Results

Test No.	Sample ID	Facies	Grind Size P80 µm	Head Assay Triplicate Au g/t	Head Assay Avg Au g/t	Calc. Head Au g/t	Gold Residue Au g/t	Gold % GRG	Overall % Extraction @ Time hours 2	4	8	12	24	36	Reagent Cons. kg/t NaCN	Reagent Cons. kg/t Lime	Silver Head Assay Ag g/t	Silver Res. Grade Ag g/t	Silver Ind. Ext'n* % Ag
BF1171	VC 1	Fresh LG	75	0.64 / 0.66 / 0.53	0.61	0.88	0.03	43	86	91	91	93	96	96.6	0.25	0.25	0.6	<0.3	75
BF1172	VC 2	Fresh HG	75	7.35 / 7.54 / 6.23	7.04	6.51	0.03	85	98	99	99	100	100	99.5	0.15	0.25	8.1	<0.3	98
BF1173	VC 3	Fresh LG	75	0.86 / 1.30 / 1.22	1.13	0.95	0.02	47	89	96	98	99	99	97.9	0.22	0.25	0.6	0.2	67
BF1174	VC 4	Fresh HG	75	3.63 / 3.66 / 5.53	4.27	3.50	0.04	45	90	95	97	98	98	98.9	0.22	0.25	1.8	<0.3	92
BF1175	VC 5	Fresh LG	75	2.24 / 2.13 / 1.69	2.02	1.52	0.02	33	90	95	96	97	99	98.7	0.40	1.25	0.3	<0.3	50
BF1176	VC 6	Oxide	75	0.58 / 0.57 / 0.65	0.60	0.67	0.02	21	91	93	94	95	96	97.0	0.44	2.20	1.2	<0.3	88
BF1177	VC 7	Fresh LG	75	1.63 / 1.17 / 1.10	1.30	1.03	0.02	37	94	97	98	98	98	98.1	0.38	0.30	<0.3	<0.3	0
BF1178	VC 8	Fresh LG	75	1.29 / 1.11 / 3.02	1.81	1.23	0.04	28	92	95	95	95	96	96.8	0.39	0.30	0.3	<0.3	50
BF1179	VC 9	Oxide	75	2.03 / 2.12 / 3.90	2.68	2.01	0.03	44	83	89	94	95	96	98.5	0.58	1.80	22	3.0	86
BF1180	VC 10	Oxide	75	3.80 / 5.17 / 3.32	4.10	3.74	0.05	49	89	93	96	96	96	98.7	0.51	3.00	3.9	<0.3	96
BF1231	MC 1	Oxide	150	1.73 / 1.26 / 1.92	1.64	2.47	0.44	34	64	70	76	77	81	82.2	0.44	3.10	8	0.3	96
BF1232	MC 1	Oxide	106			2.03	0.04	38	82	88	93	95	96	98.0	0.40	3.05		0.3	96
BF1233	MC 1	Oxide	75			2.25	0.04	58	90	93	96	98	98	98.2	0.40	3.15		<0.3	98
BF1234	MC 2	HG Fresh	150	5.87 / 4.03 / 4.36	4.75	3.91	0.07	61	86	93	96	96	97	98.2	0.16	0.35	<2	<0.3	85
BF1235	MC 2	HG Fresh	106			5.06	0.11	51	84	90	92	94	96	97.8	0.22	0.35		<0.3	85
BF1236	MC 2	HG Fresh	75			5.76	0.05	66	91	96	98	98	99	99.1	0.17	0.35		<0.3	85
BF1237	MC 3	LG Fresh	150	1.54 / 1.86 / 1.90	1.77	1.82	0.12	28	75	83	88	89	91	93.4	0.39	0.75	<2	<0.3	85
BF1238	MC 3	LG Fresh	106			1.07	0.03	31	85	91	93	93	97	97.2	0.36	0.80		<0.3	85
BF1239	MC 3	LG Fresh	75			1.62	0.04	37	88	93	94	96	98	97.5	0.39	0.75		<0.3	85

*Indicative silver extraction only - assays outside of detection limits and metallurgical balance incomplete.

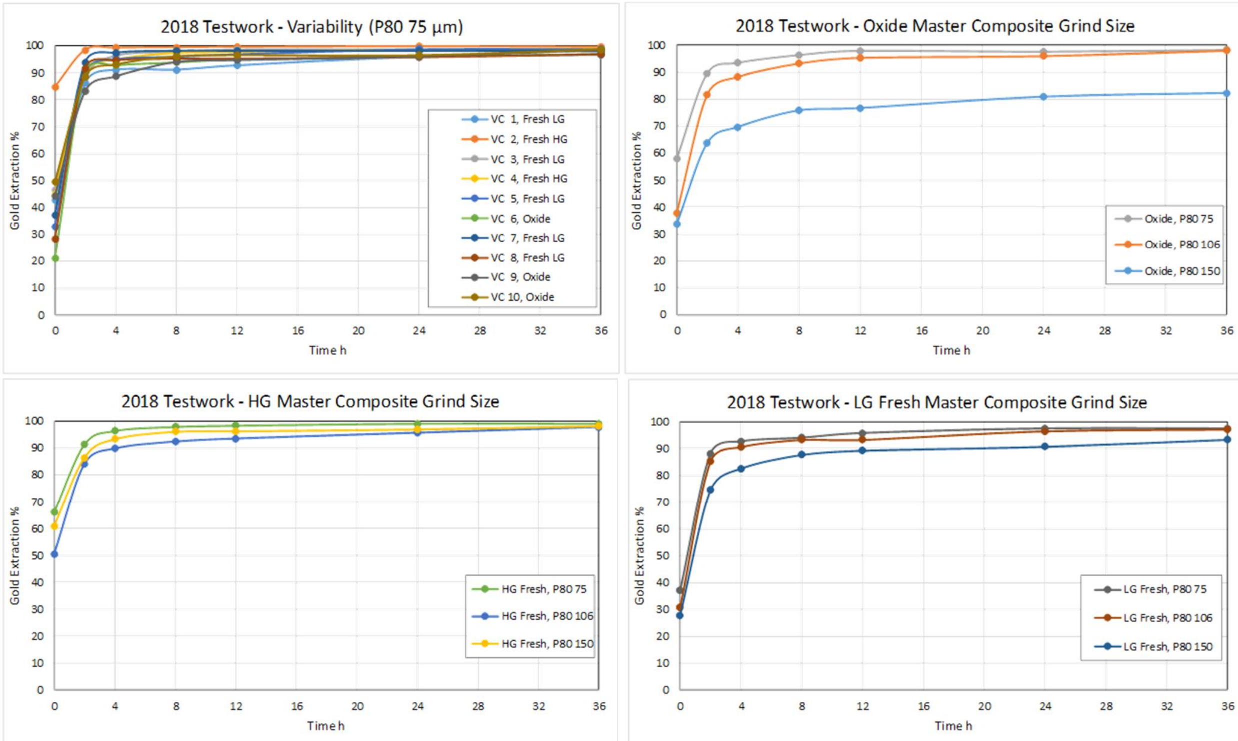


Figure 13-7: 2018 Testwork - Variability Gold Extractions and Effect of Grind Size on Master Composite Gold Extractions

13.6.2 2019 Sample Assays and Mineralogy

13.6.2.1 Formation of Ore Type Master Composites

The consistent metallurgical response from the variability samples in the 2018 scouting testwork testing phase showed that all samples from each weathered state could be considered equally representative and combined to form the master composites. No significant differences between the low- and high-grade fresh samples were noted, apart from higher gravity recoveries with the high-grade samples. Once the gravity gold was removed, leach performance and tails grades were similar.

The fresh master composite was the main composite for the optimisation testwork given that the fresh ore makes up the majority of the resource (85% for the 2019 resource) and is the driver for the process flowsheet design. The fresh master composite was made up of equal masses of 20 selected variability composites, with an expected composite gold grade of 2.75 g/t, based on the averaged exploration core assays. Several low mass variability composites were not included in the fresh master composite and were set aside for variability testing only. Two high grade composites were excluded from the fresh master composite in order to achieve a more typical average gold grade.

A high-grade fresh master composite was made up of equal masses from the two excluded high grade variability composites, with an expected composite gold grade of 27.1 g/t, based on the averaged exploration core assays.

An oxide master composite was made up of equal masses from the seven oxide variability composites, with an expected composite gold grade of 2.75 g/t, based on the averaged site assays.

The remaining portions of the variability composites and unused variability composites were independently assayed and set aside for later variability testwork.

13.6.2.2 Master Composite Head Assay

Multi-element head assays were determined for each of the main composite samples and a mineralogical investigation was conducted on the fresh ore composites.

Triplicate gold analyses were performed by standard fire-assay on the master composite with the balance of the elements being determined by ICP scan or standard assay techniques. Master composite key elements are summarised in Table 13-14.

The variance in the triplicate gold assays suggests the presence of significant coarse gold or highly localised fine gold concentrations. This presence of coarse gold was confirmed in subsequent gravity testwork.

The fresh and oxide master composite average gold grades are above the reserve averages, but within the expected annual gold grade ranges. Grade variability will be a feature of these ores. The high-grade fresh composite did not have the expected elevated sulphide and other metal grades typically associated with high-grade gold mineralisation; this was set aside to allow further investigation of the impacts of these elements, but in their absence, this composite was treated as an additional variability sample.

Silver assay head grades are below the 2 g/t detection limit for XRF. Silver extraction rates were not generally monitored in the testwork given the low head grades.

There are no deleterious elements for gold leaching present with low levels of base metals, antimony (Sb) and tellurium (Te) in the master composite. Even sulphide assays are low considering that pyrite is quite prevalent in the mineralised zones and pyrrhotite association with the gold mineralisation has been noted fairly regularly. Mercury (Hg) and arsenic (As) levels are low and should not present an environmental risk or occupational health issue in the elution or electrowinning circuits. The Te and Hg assays are elevated in the high-grade composite, suggesting a degree of gold association, since sulphides were almost absent from this sample. This is an isolated occurrence, and all assays would be significantly diluted in a normal ore parcel.

Organic carbon levels were low for all composites (<0.03% C_{org}) and preg-robbing due to the presence of organic carbon is not expected to occur.

Table 13-14: 2019 Testwork Master Composite Head Assay

Element	Master Composite		
	Fresh	High-Grade Fresh	Oxide
Au 1 (g/t)	2.53	56.5	3.57
Au 2 (g/t)	1.78	22.4	2.41
Au 3 (g/t)	1.83	19.6	3.19
Avg Au (g/t)	2.05	32.8	3.06
Ag (ppm)	<2	<2	<2
As (ppm)	<10	<10	<10
Corg (%)	0.03	<0.03	0.06
Cu (ppm)	66	44	90
Fe (%)	4.98	6.58	8.36

Table 13-14: 2019 Testwork Master Composite Head Assay

Element	Master Composite		
	Fresh	High-Grade Fresh	Oxide
Hg (ppm)	<0.1	0.5	<0.1
S (%)	0.42	<0.02	0.04
S^{2-} (%)	0.26	<0.02	0.02
Sb (ppm)	0.2	0.2	0.2
SiO_2 (%)	56.6	56.8	61.8
Te (ppm)	0.4	4.8	0.4
Zn (ppm)	116	114	112
True SG	2.79	2.81	2.79

13.6.2.3 Mineralogical Analysis – Fresh and High-Grade Fresh Master Composites

Fresh and high-grade fresh master composite sub-samples were ground to a P_{80} of 75 µm and separated into a gravity concentrate and gravity tail fractions, using centrifugal concentration and hand-panning before being prepared for mineralogical investigation. The gravity concentrates were submitted for quantitative mineralogical analysis by QEMSCAN (quantitative evaluation of minerals by scanning electron microscopy) while XRD (X-ray diffraction) was used to identify the minerals in the gravity tails.

As shown in Figure 13-8, minor amounts of pyrrhotite and pyrite were detected in the fresh master composite gravity concentrate fraction. These sulphides occur as moderately well liberated particles. Hardly any sulphide minerals (pyrrhotite, pyrite, etc.) were detected in the gravity concentrate fraction of the high-grade fresh composite.

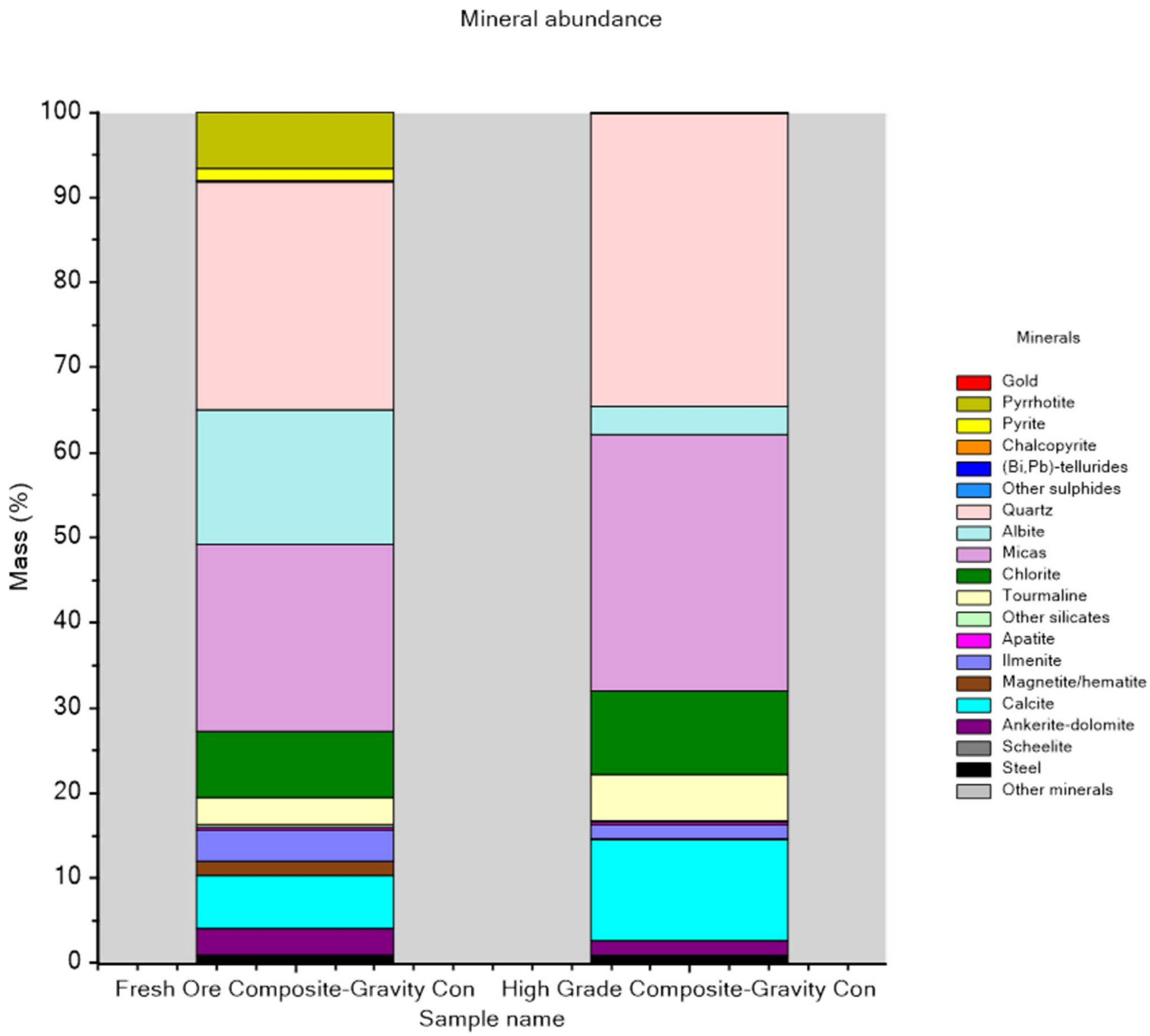


Figure 13-8: 2019 Testwork Gravity Concentrate Mineralisation

Silicates and carbonates make-up most of the remaining sample mass for each gravity concentrate fraction. The gravity tail fractions are similarly mainly silicates and carbonate minerals. Biotite, quartz, chlorite, muscovite, and albite are the main silicate minerals. Calcite is the main carbonate mineral.

Optical microscopy and QEMSCAN analysis were used to detect gold grains in the concentrates. Some coarse free/liberated grains were found in both the concentrates (Figure 13-9), but a number of the grains were partially or totally encapsulated. Eleven grains were detected in the master composite, and 56 grains in the high-grade composite. The largest free gold grain is approximately 1.5 mm in size (detected in the high-grade composite) and the remaining grains ranged in size from (2 to 500) µm.

The smaller grains in the master composite were exclusively pyrite/pyrrhotite hosted (frequently encapsulated), but those in the high-grade composite were mainly associated with the bismuth tellurides (Bi, Pb Te). Examples are presented in Figure 13-10.

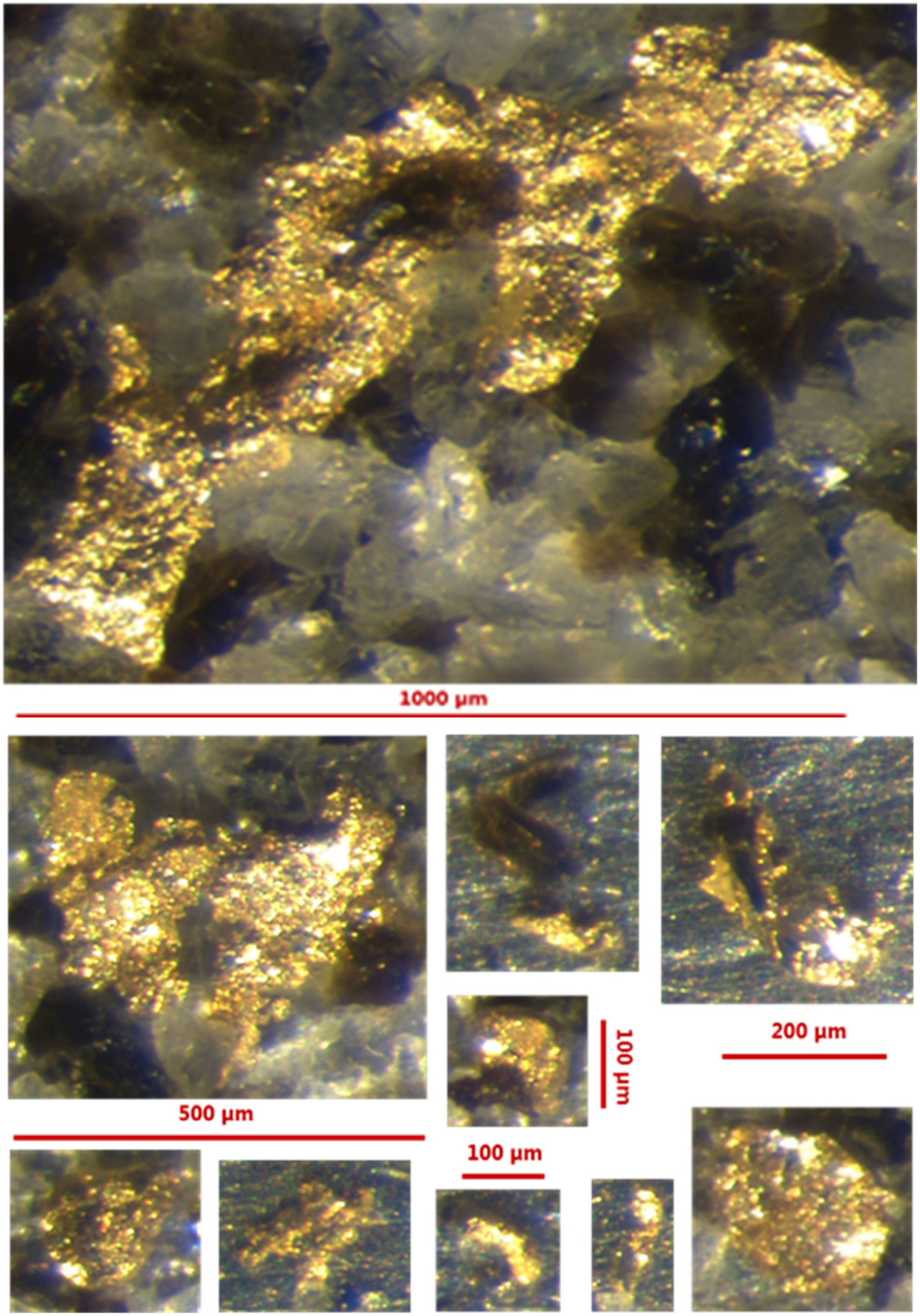


Figure 13-9: 2019 Testwork Coarse Gold Mineralisation

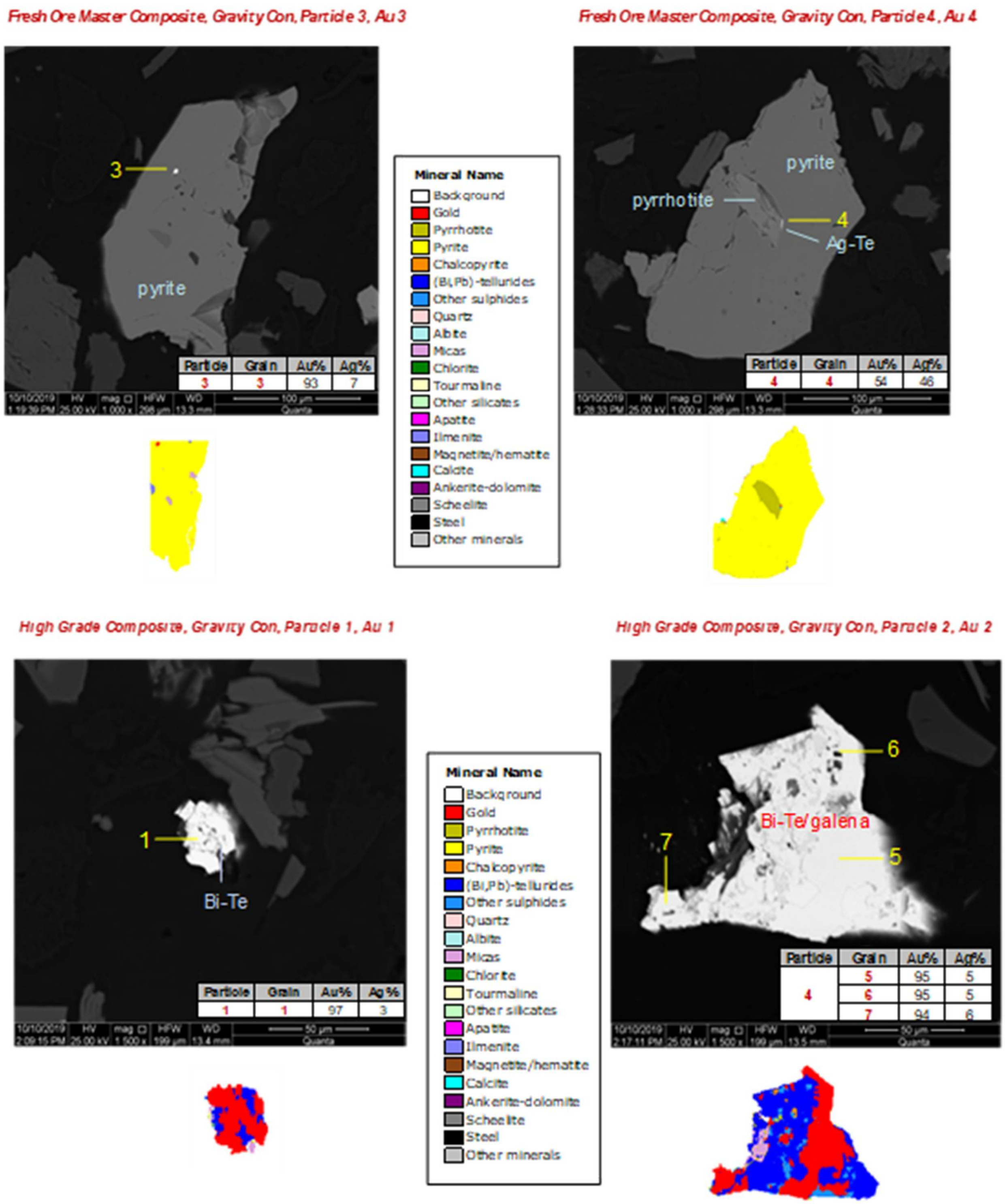


Figure 13-10: 2019 Testwork Locked Gold Mineralisation

13.6.3 2021 Variability Sample Head Analyses

Gold analyses were performed using standard fire-assays on sub-samples split from the 40 variability samples and three composite samples. The balance of the elements was determined by ICP scan or specific element standard assay techniques. Key results for the composite samples are summarised in Table 13-15.

- The triplicate gold assays of the 40 variability samples and the average master composite typically show significant variability (even in some low-grade samples) confirming the presence of gold as coarse particles, which agrees with the conclusion from previous testing. Subsequent gravity concentration testwork showed that much of the gold (typically 70%, based on average master composite sample) presented as gravity recoverable gold.
- To highlight the variability in individual sub-sample assays, triplicate gold assays of the 40 metallurgical variability samples (used to make-up the master composites), are shown in Figure 13-11. It is noteworthy that some samples had moderately consistent assays suggesting more disseminated fine mineralisation, but the impact of coarse-grained gold on assay variability is apparent for many of the samples.
- Silver head grades are just above the 2 ppm ICP detection limit for the 'O' and master composites. These higher grades result from the occasional variability sample with higher Ag assays (up to 6 ppm), but silver values remain generally below detection levels. Silver extraction rates were not generally monitored in the testwork given the low head grades.
- Iron and sulphur concentrations appear to be more elevated in samples that were found to exhibit lower overall gold leach extractions.
- Elements deleterious to gold leaching such as base metals, antimony, arsenic, and tellurium are rarely present in the Lafigué ore and, if so, concentrations are negligible. Isolated higher Te and Hg assays are evident in some variability samples. The higher tellurium assays (up to 7.6 ppm) tend to align with higher gold and silver values. Mercury levels in the variability set remain generally low (<0.1 ppm) and appear more randomly distributed, although some high values (up to 2.6 ppm) align with higher gold assays. If moderate mercury levels are present in the ore, this could present an occupational health risk in the elution or electrowinning circuits. Mercury levels in the leach solutions and on the carbon should be monitored, with any emissions being controlled using industrially available mercury abatement systems. Given the rarity of the occurrences and likely low degree of solubilisation that may occur, monitoring only is recommended.
- Organic carbon levels were low for all composites (<0.03% Corg) and preg-robbing due to the presence of organic carbon is not expected to occur.

Full assays are included in Appendix II of the ALS Testwork Report A21932 (ALS, 2021b), and further discussion of the sample assays is covered in Section 13.6.3.

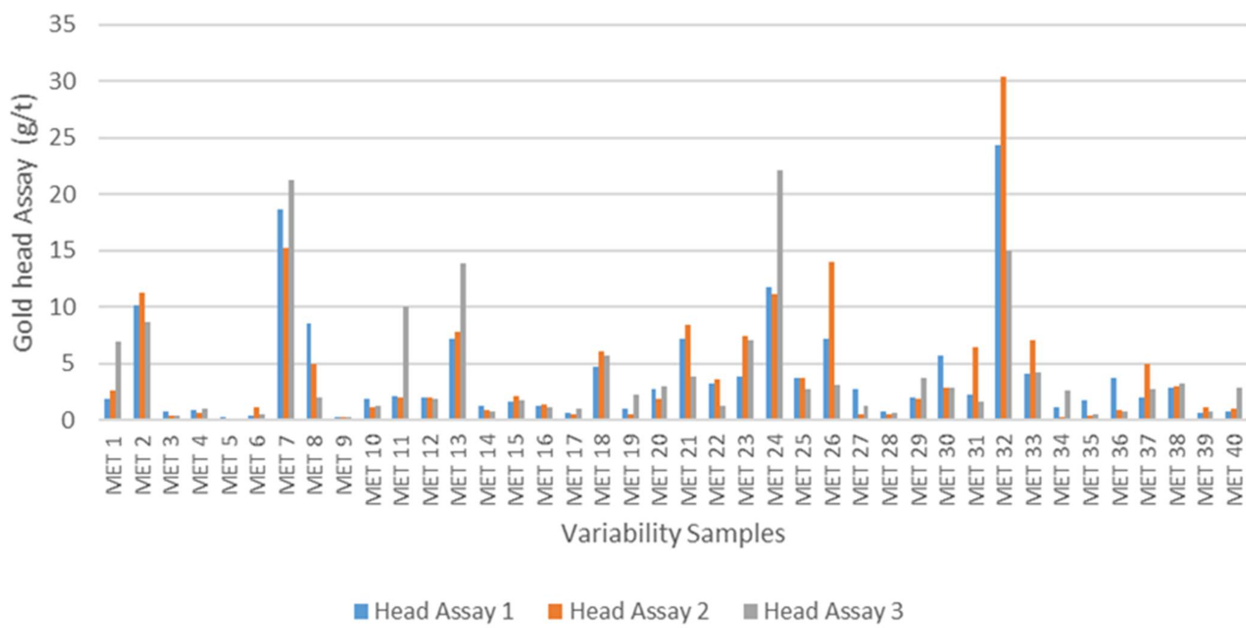


Figure 13-11: Variability Composite Gold Assays

Table 13-15: Key Elemental Head Assays for the Lafigué Composites

Composite Samples	Au (g/t)	Au (g/t)	Au (g/t)	Ave. Au (g/t)	Ag (g/t)	Cu (ppm)	Fe (%)	Hg (ppm)	S^{2-} (%)	Sb (ppm)	Te (ppm)	Zn (ppm)
Ave. Grade Master Composite	1.07	1.78	2.20	1.68	3	57	5.27	0.58	0.18	0.5	1.9	97
O Composite	3.54	3.71	-	3.63	3	59	4.90	0.24	0.24	0.1	1.4	105
P Composite	1.19	3.32	-	2.26	<2	64	6.05	0.08	0.60	0.2	0.6	78

13.6.4 2021 Mineralogical Analysis by QEMSCAN and XRD

The gravity concentrate and gravity tailings fractions of the 'O' and 'P' composites samples were submitted for quantitative mineralogical analysis by QEMSCAN and XRD. Each sample was separated into gravity concentrate and gravity tail fractions using a Knelson concentrator and hand-panning before being prepared for mineralogical investigation.

Each gravity concentrate fraction was mixed with high purity graphite to ensure particle separation and discourage density segregation. The sample-graphite mixtures were then set into moulds using an epoxy resin, producing a representative sub-sample of randomly orientated particles. The resin blocks were cut back to expose a fresh surface, ground and fine-polished ahead of being presented to the QEMSCAN for analysis.

Powder XRD was used to analyse the gravity tails samples to quantify the minerals identified.

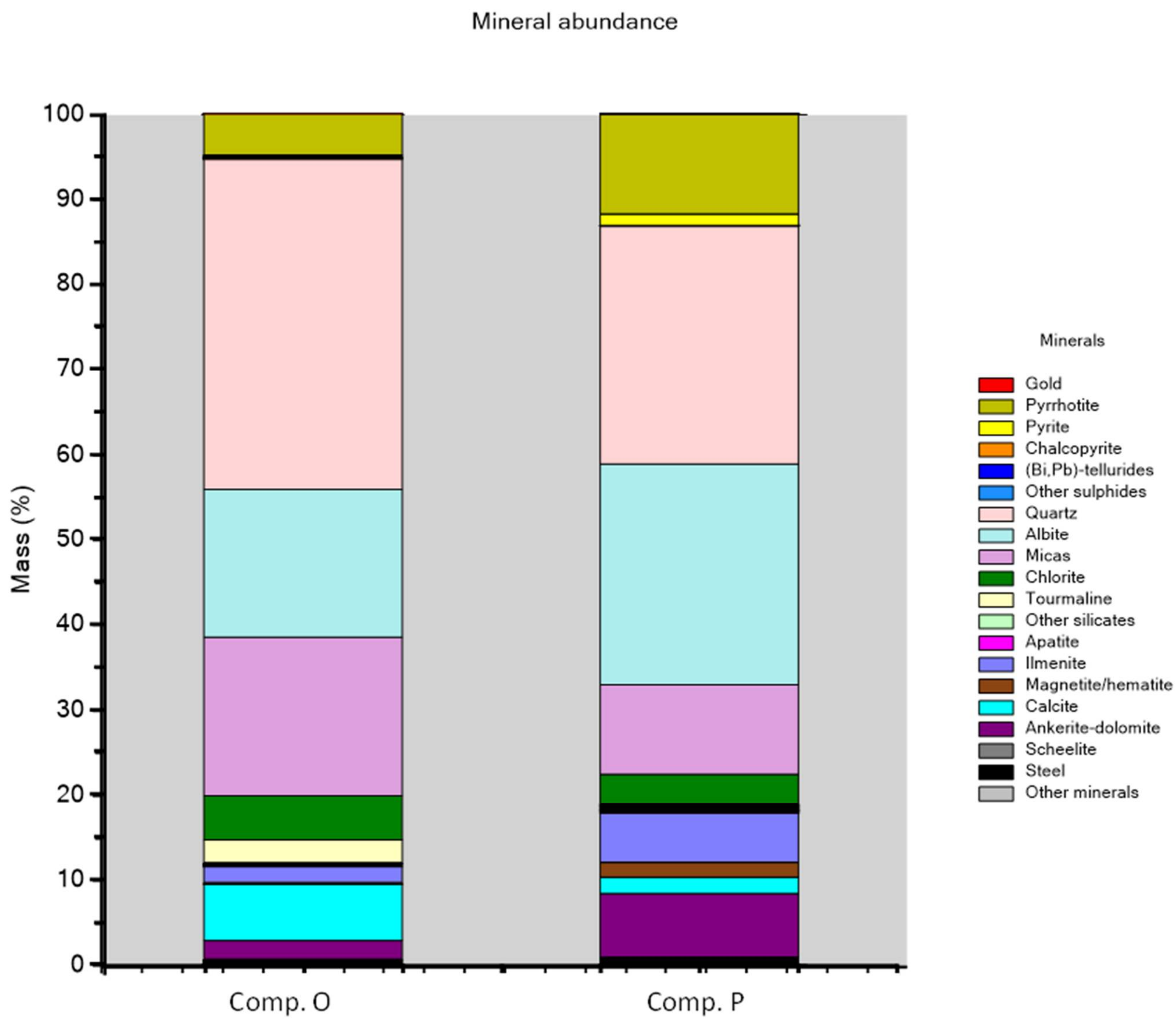


Figure 13-12: P and O Composite Gravity Concentrate Mineralisation

Minor amounts of pyrrhotite (4.82% w/w) and pyrite (0.39% w/w) were detected in the 'O' composite gravity concentrate fraction as shown in Figure 13-12. The gravity concentrate from the 'P' composite showed a higher concentration of pyrrhotite (11.8% w/w) and pyrite (1.29% w/w). These sulphides occur as moderately well liberated particles.

Of interest is the difference between these composites in terms of the alteration: with the absence of tourmaline and relatively little calcite in comp. P. Equally contrasting are the larger amounts of magnetite/hematite and ilmenite in comp. P. It is noteworthy that the key constituents of comp. P are from significantly shallower depths, (115 to 135 m), compared to over 200 m for most of the 2021 testwork samples. This suggests that the mineralisation style may differ slightly away from the footwall contact where most of the gold mineralisation occurs. Comparing these examples with the 2019 master composite (Figure 13-8) shows that this appears to be an average of comp. P and comp. O being made up from a selection of fresh samples from the shallower part of the ore body. This would include examples of mineralisation away from the footwall and extending into the deeper orebody as indicated in Figure 13-2.

Silicates and carbonates make up most of the remaining sample mass for each gravity concentrate fraction. The gravity tail fractions are mainly silicates and carbonate minerals. Biotite, quartz, chlorite, muscovite, and albite are the main silicates. Calcite and ankerite-dolomite are the main carbonate minerals.

Optical microscopy and QEMSCAN analysis were used to detect gold grains in the concentrates. Five coarse grains, ranging in size from (50 to 400) um, were observed in both the concentrates (Figure 13-13 and Figure 13-14) under optical microscopy.

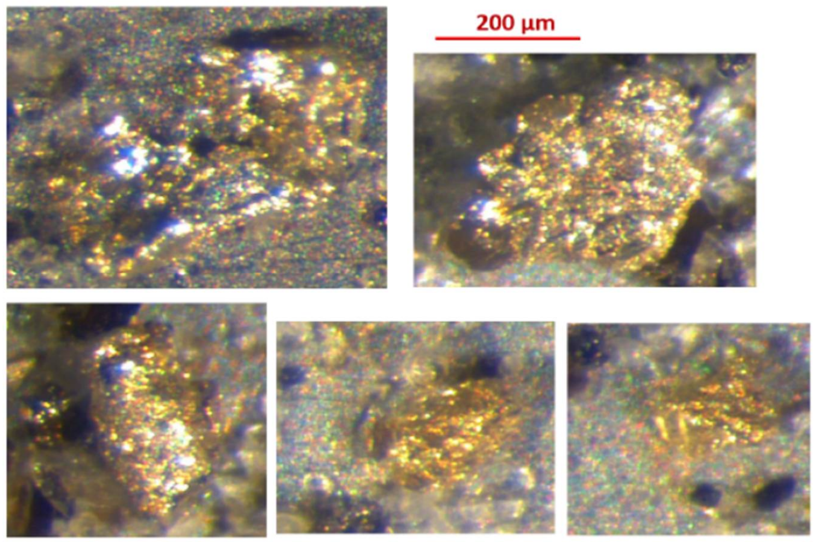


Figure 13-13: Coarse Gold Mineralisation - Optical Microscopy O Composite

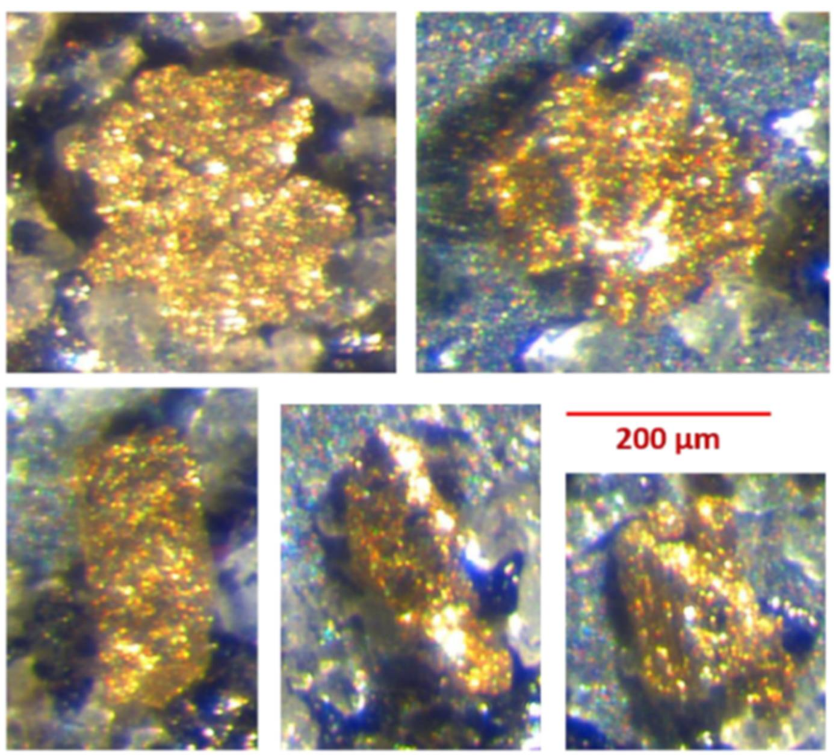


Figure 13-14: Coarse Gold Mineralisation - Optical Microscopy P Composite

QEMSCAN detected one liberated gold grain with a 77% Au + 23% Ag composition and one pyrrhotite enclosed gold grain, with a 88% Au + 12% Ag composition in the P composite sample. None was detected in the O composite sample. These are illustrated in Figure 13-15.

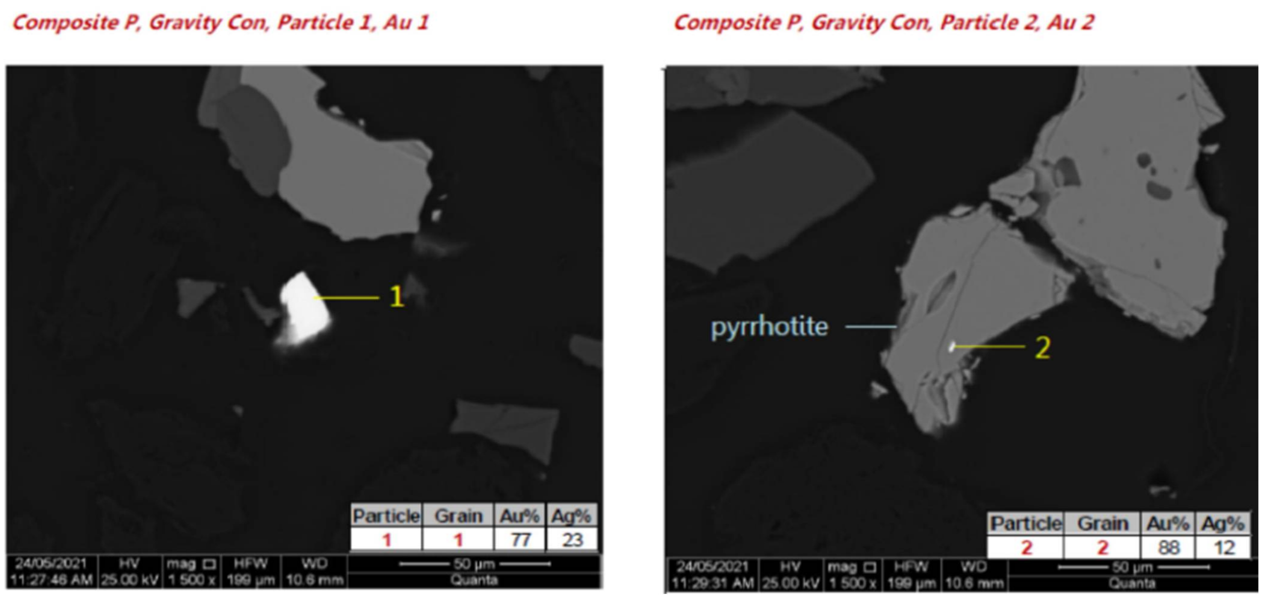


Figure 13-15: Gold Mineralisation - QEMSCAN P Composite

The full mineralogical investigation report is appended to the ALS Testwork Report A21932

13.7 2019 Master Composite Metallurgical Testing

The metallurgical testwork was carried out on fresh and oxide master composites which were considered representative of each of the weathered states. Previous testwork on Lafigué variability samples along with an extensive BLEG programme had demonstrated consistent metallurgical performance with high gold extractions in all cases, so 'sighter' leach tests on the individual samples making up the composite were deemed unnecessary.

The programme aimed to determine optimum processing conditions for the master composite samples. These conditions would then be applied to the individual variability composites making up the master composites as well as additional variability samples selected to provide examples of specific mineralisation types or less prevalent host rock mineralisation. This approach is cost effective when the metallurgical characteristics of the ore are reasonably well understood, reducing the individual sample masses required for testing and the number of tests required.

All metallurgical testwork was undertaken by ALS Metallurgy using Perth tap water.

13.7.1 Master Composite Grind Optimisation Testing

Testwork to determine the optimum grind size for gold extraction, was carried out on the fresh and oxide master composites. Grind/extraction tests, with gravity prior to leaching were completed at grind sizes of P_{80} 125, 106, 90 and 75 µm.

After milling to the target grind size, gravity gold was recovered by centrifugal concentration with mercury amalgamation of the gravity concentrate. Amalgam tails and gravity tails were then combined for leaching. Intermediate solution samples were taken during the leach tests to monitor gold dissolution kinetics.

The grind/gravity/cyanidation gold and silver extraction results are summarised in Table 13-16 and Figure 13-16. The fresh composite results typically indicated:

- Gravity gold content is high (60 to 70 %) with typically significant variation between the assayed head and calculated head gold grades, despite each sample being split from the same blended composite sample. The large differences in gravity gold content (still evident in leach tails assay variability) made it difficult to compare the test results precisely. Gravity recovery appeared to be slightly lower for the coarser grind (125 µm) suggesting that more free gold was liberated at the finer grinds.
- The fresh composite appeared to be relatively insensitive to grind, with high leach extractions across the size range tested. Some improvement in gold extraction with increasing fineness of grind was evident, with lower residue grades and faster leach kinetics.
- Reagent consumption was low and similar across the grind size range tested.
- Silver head grades for this test series were very low (0.6 g/t Ag) and there was no discernible trend in extraction with grind size. Silver extractions averaged 75% with residue grades of approximately 0.15 g/t Ag. Silver leach kinetics were slightly faster at the higher cyanide concentration, but final silver extractions were similar in both sets of testwork.
- Silver extraction was not tracked in subsequent testing due to the generally low head grades and low economic value to the project.
- Gold extraction was essentially complete within 8 to 12 hours, but minor variance in solution assays continued through to 36 hours. There is potentially a small fraction of partially locked or silver/telluride/pyrrhotite associated gold that would be slower leaching. This fraction typically accounts for <0.5% additional extraction.
- Leach kinetics were slightly slower for some tests with lower cyanide additions, but no clear trend with grind size was evident. Variability between tests is evident along with the impact of gravity gold and possibly other mineralisation differences affecting the results more than the grind size despite being split from the same well mixed sample.

Table 13-16: 2019 Testwork Master Composite Gravity Leach Grind Optimisation Tests

Master Composite	Testwork	Test No.	Grind Size P_{80} µm	Initial NaCN % w/v	Leach Solids %w/w	Sparge Gas	Assay Head Au g/t	Calc Head Au g/t	Leach Residue Au g/t	% GRG	Overall % Gold Extraction @ Time hrs						Reagent Cons (kg/t)	
											2	4	8	12	24	36	NaCN	Lime
Fresh	Grind Opt	JS4483	125	0.02	40	O_2	2.05	2.04	0.06	61	83	93	95	96	96.7	97.3	0.22	0.25
		JS4484	106	0.02	40	O_2	2.05	2.51	0.03	68	88	95	97	98	98.0	98.8	0.18	0.25
		JS4485	90	0.02	40	O_2	2.05	1.66	0.03	58	83	94	95	96	97.0	98.2	0.12	0.35
		JS4486	75	0.02	40	O_2	2.05	1.87	0.02	64	83	96	98	98	99.0	98.9	0.20	0.30
	Grind Opt Repeat	JS4483R	125	0.10	40	O_2	2.05	2.74	0.05	60	90	94	97	97	98.0	98.2	0.28	0.30
		JS4484R	106	0.10	40	O_2	2.05	2.71	0.05	70	93	97	98	98	99.1	98.3	0.31	0.25
		JS4485R	90	0.10	40	O_2	2.05	2.62	0.05	69	94	97	98	98	97.9	98.1	0.28	0.26
		JS4486R	75	0.10	40	O_2	2.05	2.33	0.02	70	96	98	98	99	98.9	99.1	0.31	0.27
Oxide	Grind Opt	JS4489R	106	0.10	40	O_2	3.06	3.54	0.04	74	94	97	99	99	98.9	98.9	0.20	3.35
		JS4490R	75	0.10	40	O_2	3.06	4.62	0.03	81	97	99	99	99	99.6	99.4	0.18	3.15
							Ag g/t	Ag g/t	Ag g/t		Overall % Silver Extraction @ Time hrs							
Fresh	Grind Opt	JS4483	125	0.02	40	O_2	<2	0.6	0.15	-	29	48	62	71	75.8	75.7	0.22	0.25
		JS4484	106	0.02	40	O_2	<2	0.6	0.15	-	34	48	62	67	71.4	75.7	0.18	0.25
		JS4485	90	0.02	40	O_2	<2	0.6	0.15	-	31	46	60	65	74.5	74.5	0.12	0.35
		JS4486	75	0.02	40	O_2	<2	0.6	0.15	-	32	48	63	68	73.3	73.2	0.20	0.30
	Grind Opt Repeat	JS4483R	125	0.10	40	O_2	<2	0.7	0.15	-	46	68	73	73	72.9	77.0	0.28	0.30
		JS4484R	106	0.10	40	O_2	<2	0.6	0.15	-	43	58	68	73	73.5	73.4	0.31	0.25
		JS4485R	90	0.10	40	O_2	<2	0.7	0.15	-	46	59	67	75	78.9	78.9	0.28	0.26
		JS4486R	75	0.10	40	O_2	<2	0.5	0.15	-	50	61	72	72	72.1	72.0	0.31	0.27

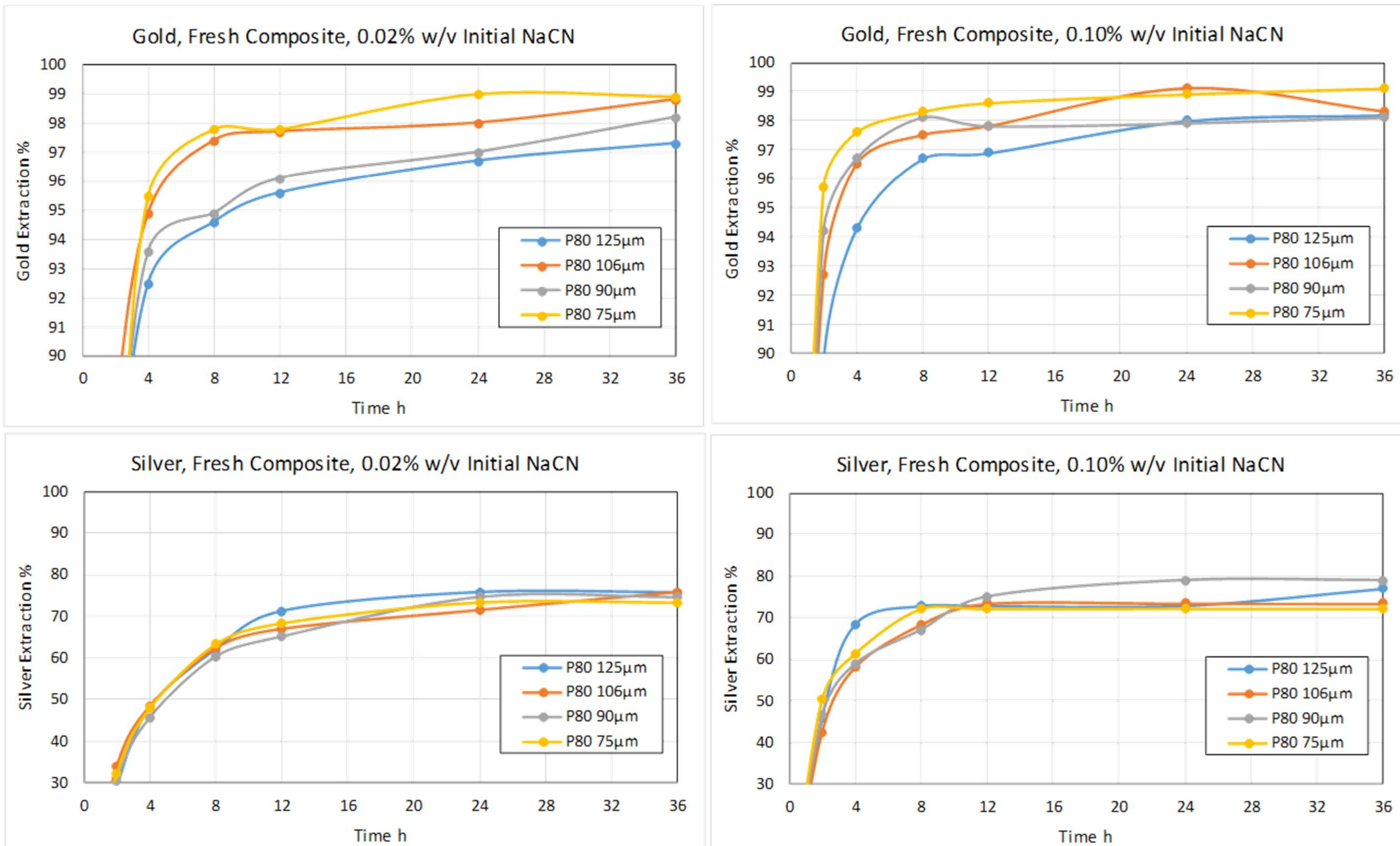


Figure 13-16: 2019 Testwork - Effect of Grind Size on Gold and Silver Extraction for Fresh Master Composite

As the oxide ores are a minor component of the feed blend and are not grind determining, testwork on the oxide composite was completed at a grind P_{80} of 75 and 106 µm only. Leach kinetics were faster than the fresh ore, with slightly higher overall gold extraction. This composite is considered insensitive to grind size on the basis of these test results.

The oxide tests were similarly repeated at a higher cyanide concentration with similar results to the original set. One notable difference was the significantly higher lime demand for the oxide ore compared to the fresh ore. This is a common feature of oxide ore processing where even small amounts of clay/iron oxides interact with the lime to make oxyhydroxides independently of pH modification requirements.

13.7.2 Grind Optimisation Economic Analysis

The optimum grind economic analysis for the fresh ore was completed using the testwork results and appropriate gold, consumable, and power prices. The evaluation compared gold revenue against operating expenditure for the grind sizes tested. The net revenue (gold revenue less operating cost) was calculated for each grind size. The marginal change in operating cost, gold revenue and net revenue was compared using a grind size of P_{80} 75 µm as the basis.

The original economic analysis (in 2019) selected a grind size of P_{80} 106 µm as the optimum, so this grind was used for the balance of the 2019 testing. The basis for this economic analysis was a gold price of USD 1500/oz per the resource optimisation constraint and a power unit cost of USD 0.13/kWh, as advised by Endeavour for the Cote d'Ivoire grid supply.

A 2022 review of the results highlighted that the P_{80} 90 µm test was compromised with low head grades in both test series and that the current study grid power cost (4Q21 basis) is USD 0.112/kWh making more energy intensive grinding more attractive. Fitting a plausible trend to the gold recovery data indicates a slightly finer optimum grind. The trend offsets are, however, within the assay accuracy at the low residual gold levels achieved. Since the 2019 analysis, the gold price has increased (although, the study basis remained constant), further favouring a finer grind. The benefit of hindsight from further testing also suggests there may be a degree of risk mitigation in finer grinding and suggests a range between a P_{80} of 90 µm and 106 µm would be an appropriate target. A grind size range is more manageable for the operation as this allows steadier grinding circuit control with fewer step changes to meet a more tightly specified target.

13.7.3 Bulk Gravity Gold Separation and Preparation of Bulk Gravity Tails

Gravity gold recovery (GRG) testing for individual test samples was generally conducted using the laboratory Knelson concentrator fed with 1 kg sub samples of each composite ground to the nominated P_{80} grind size. The fixed mass recovery based on the gold captured in the grooves is relatively too great a fraction of the feed mass tested compared with plant practice. In order to obtain a more representative gravity concentrate sample (typically <0.1% of the feed mass), bulk gravity separation testwork was performed on the fresh and oxide master composites, as well as the high-grade fresh composite. In addition to demonstrating maximum GRG recoveries, these tests generated a set of gravity tailings samples with a more consistent head grade for downstream leach optimisation testwork.

The bulk gravity tests were conducted on 30 kg sub-samples of each ore composite at a P_{80} grind size of 106 µm using the laboratory Knelson concentrator for gravity separation. The gravity concentrate was upgraded using a Superpanner to be ~0.2% of feed mass for intensive cyanidation with 5.0% w/v starting NaCN and LeachWELL® addition at pH 12. This approach better aligns with mass recovery in plant practice and avoids overstating the gravity recoverable gold content. The intensive leach solution was submitted for gold assay, whilst the intensive leach residue was recombined with the bulk gravity tailings.

The bulk gravity tailings were thoroughly homogenised and split into sub-samples for the downstream testwork programme. The gold recoveries from this stage (Table 13-17) were incorporated into the metallurgical balances for the subsequent gravity tails leach test series.

Table 13-17: 2019 Testwork Master Composite Bulk Gravity Test Work

Sample ID	Test No.	Head Assay, Au (g/t)	Calc'd Head, Au (g/t)	Gravity Tails Assay, Au (g/t)	% GRG Au Extr'n
Fresh Ore Master Comp.	JS4491	2.53/1.78/1.83	3.03	0.73/0.59/0.71	77.7
Fresh Ore High-Grade Comp.	JS4492	56.5/22.4/19.6	15.7*	Not assayed*	91.5
Oxide Ore Master Comp.	JS4493	3.57/2.41/3.19	4.66	1.17/1.23/1.27	73.8

*Gravity tails sample not assayed as expected to be high grade; sample calculated head determined from concentrate assay and subsequent leach testwork solution and tails assays.

It is noteworthy that for both master composites the calculated heads following gravity recovery were higher than all the triplicate head assays. Also, tails assays following gravity gold recovery still showed a reasonable degree of variability indicating that the 'nugget' style gold mineralisation persists in the finer gold mineralisation.

13.7.4 Master Composite Leach Optimisation Testwork

Gold extraction versus cyanide concentration, slurry density and air/oxygen tests were conducted on the fresh and oxide bulk gravity tailings at the selected P_{80} grind size of 106 µm. Optimisation testwork focussed on the fresh ore composite as it represents the majority of the resource. The optimisation test series results are included in Table 13-18.

The leach optimisation results are discussed in more detail in the following sections, with graphical presentation of the key results for comparative purposes.

13.7.4.1 Cyanide Concentration Test Series

As cyanide usage was low in all previous tests and the initial grind series with low starting cyanide showed that gold extraction was similar, the following cyanide test series was completed on the fresh and oxide master composite bulk gravity tails:

- Initial cyanide strength of 400 ppm NaCN maintained above 200 ppm at sampling intervals.
- Initial cyanide strength of 200 ppm NaCN maintained above 200 ppm at sampling intervals.
- Initial cyanide strength of 200 ppm NaCN maintained above 150 ppm at sampling intervals.

All tests were conducted at 40% solids with oxygen sparging with results graphed in Figure 13-17. The baseline 106 µm tests from the grind optimisation series are included (1000 ppm starting NaCN and 200 ppm starting NaCN) for comparison.

The tests indicated that gold extraction was not sensitive to cyanide concentration over the range tested. Overall gold extractions were between (98 and 99)%, irrespective of initial cyanide concentration. For the fresh ore, cyanide consumption decreased from 0.31 kg/t to 0.07 kg/t as the initial cyanide concentration was decreased, showing that increased NaCN concentrations drive the consumption higher with more side reactions occurring.

Based on the results from the test series, a slightly conservative initial cyanide concentration of 250 ppm (0.025% w/v) NaCN was selected as the optimum addition rate. This higher value more readily ensures that 0.02% w/v NaCN will be maintained beyond start-up.

Table 13-18: 2019 Testwork - Master Composite Leach Optimisation

Master Composite	Testwork Series	Test No.	Grind Size P_{80} µm	NaCN Conc Initial % w/v	NaCN Conc Maint. % w/v	Leach Solids % w/w	Sparge Gas	Assay Head Au g/t	Calc Head Au g/t	Residue Au g/t	GRG %	Overall % Gold Extraction @ Time Hours 2	4	8	12	24	36	Reagent Cons kg/t NaCN	Lime
Fresh	Grind Opt Repeat	JS4483R	125	0.10	0.05	40%	O_2	2.05	2.74	0.05	60	90	94	97	97	98.0	98.2	0.28	0.3
		JS4484R	106	0.10	0.05	40%	O_2	2.05	2.71	0.05	70	93	97	98	98	99.1	98.3	0.31	0.25
		JS4485R	90	0.10	0.05	40%	O_2	2.05	2.62	0.05	69	94	97	98	98	97.9	98.1	0.28	0.26
		JS4486R	75	0.10	0.05	40%	O_2	2.05	2.33	0.02	70	96	98	98	99	98.9	99.1	0.31	0.27
	NaCN Conc Opt.	JS4493	106	0.04	0.02	40%	O_2	2.05	2.98	0.04	79	94	97	98	98	98.4	98.7	0.14	0.25
		JS4494	106	0.02	0.02	40%	O_2	2.05	3.00	0.06	79	91	95	97	97	98.0	98.0	0.11	0.25
		JS4495	106	0.02	0.015	40%	O_2	2.05	2.97	0.05	79	91	95	97	98	98.5	98.3	0.07	0.25
	Slurry Density Opt	JS4496	106	0.10	0.05	50%	O_2	2.05	2.99	0.04	79	95	97	98	98	98.4	98.7	0.2	0.2
		JS4497	106	0.10	0.05	55%	O_2	2.05	2.99	0.04	79	95	98	98	98	98.8	98.7	0.19	0.15
		JS4498	106	0.10	0.05	60%	O_2	2.05	3.03	0.06	78	94	96	97	97	97.5	98.0	0.21	0.13
	Air vs. O2	JS4499	106	0.10	0.05	40%	Air	2.05	2.99	0.05	79	97	97	98	98	98.3	98.3	0.29	0.25
	Bulk Leach	JS4501	106	0.025	0.02	55%	Air	2.05	3.00	0.06	78	88	93	96	97	98.0		0.08	0.23
	No Gravity	JS4588	106	0.025	0.02	55%	Air	2.05	1.50	0.10	0	18	29	51	66	86.2	93.3	0.09	0.3

Table 13-18: 2019 Testwork - Master Composite Leach Optimisation

Master Composite	Testwork Series	Test No.	Grind Size P_{80} µm	NaCN Conc Initial % w/v	NaCN Conc Maint. % w/v	Leach Solids % w/w	Sparge Gas	Assay Head Au g/t	Calc Head Au g/t	Residue Au g/t	GRG %	Overall % Gold Extraction @ Time Hours 2	4	8	12	24	36	Reagent Cons kg/t NaCN	Lime
Oxide	Grind Opt Repeat	JS4489R	106	0.10	0.05	40%	O_2	3.06	3.54	0.04	74	94	97	99	99	98.9	98.9	0.2	3.35
		JS4490R	75	0.10	0.05	40%	O_2	3.06	4.62	0.03	81	97	99	99	99	99.6	99.4	0.18	3.15
	NaCN Conc Opt.	JS4505	106	0.04	0.02	40%	O_2	3.06	4.73	0.04	73	95	97	99	98.7	99.2	99.2	0.10	2.90
		JS4506	106	0.02	0.02	40%	O_2	3.06	4.75	0.05	72	92	96	97	97.9	98.8	98.9	0.10	2.70
		JS4507	106	0.02	0.015	40%	O_2	3.06	4.71	0.05	73	93	96	98	98.4	98.2	98.9	0.07	2.70
	Slurry Density Opt	JS4508	106	0.10	0.05	45%	O_2	3.06	4.70	0.04	73	95	97	98	98.5	98.8	99.1	0.24	3.00
		JS4509	106	0.10	0.05	35%	O_2	3.06	4.71	0.03	73	96	98	99	99.0	99.4	99.4	0.32	2.70
		JS4510	106	0.10	0.05	32%	O_2	3.06	4.84	0.04	71	94	98	99	98.6	99.2	99.2	0.43	3.00
	Air vs. O2	JS4511	106	0.10	0.05	40%	Air	3.06	4.72	0.06	73	92	96	97	98	98.4	98.7	0.30	3.15
	Bulk Leach	JS4513	106	0.025	0.02	40%	Air	3.06	4.78	0.13	72	92	94	95	96	97.3		0.25	2.46
High Grade Fresh	Air vs. O2	JS4502	106	0.025	0.02	55%	Air	32.8	15.70	0.05	92	98	99	99	100	99.7		0.05	0.20
		JS4503	106	0.025	0.02	55%	O_2	32.8	15.70	0.06	92	99	99	99	99	99.6		0.01	0.20

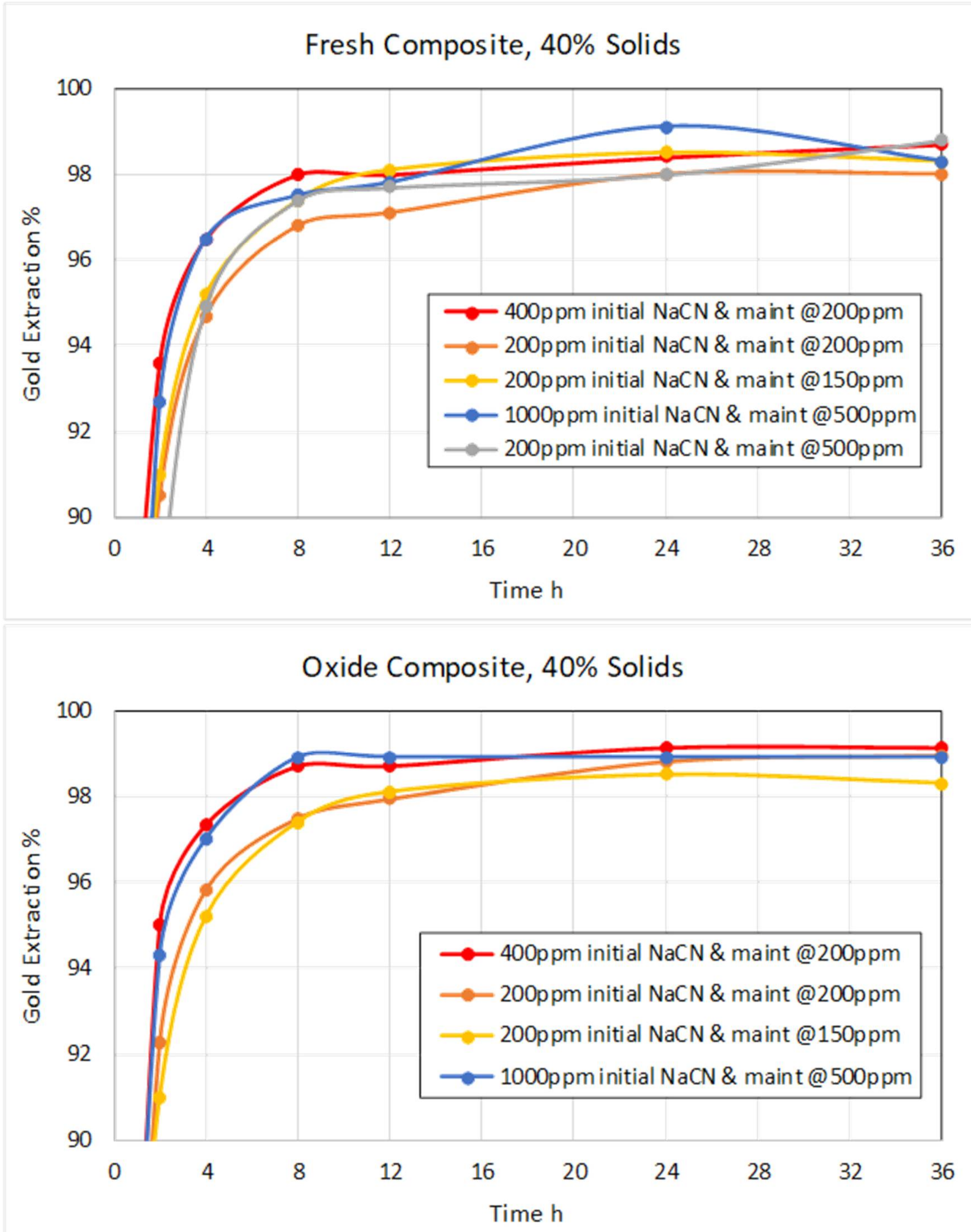


Figure 13-17: 2019 Testwork - Master Composite Cyanide Optimisation

13.7.4.2 Slurry Density Test Series

The impact of leach slurry density on gold leach kinetics and overall extraction for the fresh and oxide bulk gravity tails was tested. Slurry densities trialled were:

- Fresh - 40% (from grind optimisation series), 50%, 55% and 60% w/w solids.
- Oxide - 32%, 35%, 40% (from grind optimisation series) and 45% w/w solids.

Higher densities were tested for the fresh ore since it was planned to thicken the leach feed. The oxide ore was tested over a lower density range in case there were slurry viscosity issues as slurry density increased. All tests were conducted at 0.1% w/v initial NaCN concentration with oxygen sparging. Leach kinetic curves for the density tests are presented in Figure 13-18. The slurry density test details are presented in Table 13-18.

For the fresh ore, overall gold extractions were very similar suggesting that slurry density had little impact on leaching efficiency, although the test conducted at 60% w/w solids displayed slightly slower gold leach kinetics than for the lower slurry density tests. The 40% w/w solids test sample used the result from the grind optimisation series and went slightly against trend suggesting that the different approach to gravity gold recovery (amalgamation vs. intensive cyanidation) may have had a minor impact on the tails leach performance.

For the oxide ore, leach slurry density also had very little impact on overall gold extractions or leach kinetics. However, based on the scouting rheology testing and high oxide viscosity result (Section 13.12.3), it may be advisable to maintain a lower oxide leach density for some of the oxide ore types. Based on the subsequent process flowsheet for the operation, oxide ores will be treated as low blend fractions of the feed, so potentially viscous slurries are unlikely to be encountered.

Based on the density test series, a fresh ore slurry density of 55% w/w solids was selected as the preferred leach operating point. Low oxide/transitional feed blend fractions are not expected to affect this operating density.

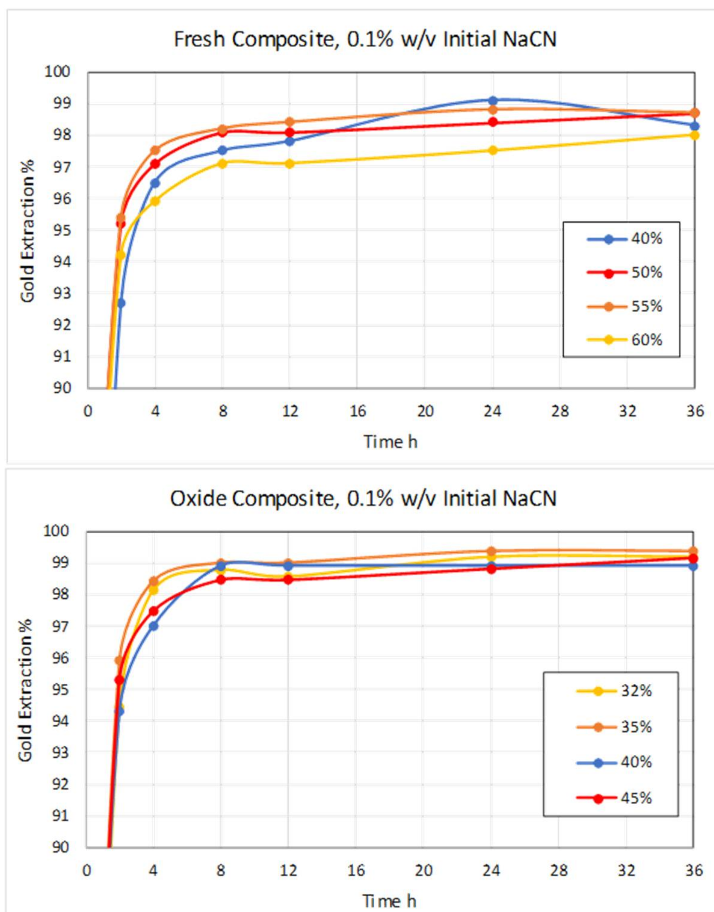


Figure 13-18: 2019 Testwork - Master Composite Slurry Density Optimisation

Figure 13-18 note: legend based on solids on a per cent by weight basis

13.7.4.3 Air vs. Oxygen Test Series

All cyanidation tests to this point were conducted with oxygen sparging. To evaluate the benefits of oxygen (O_2) sparging, air sparging tests were carried out at 40% w/w solids and an initial 0.1% NaCN addition, to allow comparison with the grind optimisation testwork results. Results are presented in Figure 13-19 and indicate that oxygen sparging marginally improved the initial leach kinetics but gold extraction was essentially similar and complete after 24-hours of leaching in both cases. The impact was more noticeable for the oxide composite, which also required significantly more lime for pH modification, suggesting more reactive components were present and/or possible clay buffering was occurring. The air vs. oxygen test results are included in Table 13-18.

It was decided that air sparging of the leach slurry would be satisfactory following this testing. This enhances process simplicity without the additional capital and ongoing operating costs associated with an oxygen generation plant. Subsequent oxygen uptake rate tests (Section 13.12.2) confirmed negligible oxygen demand for these ores.

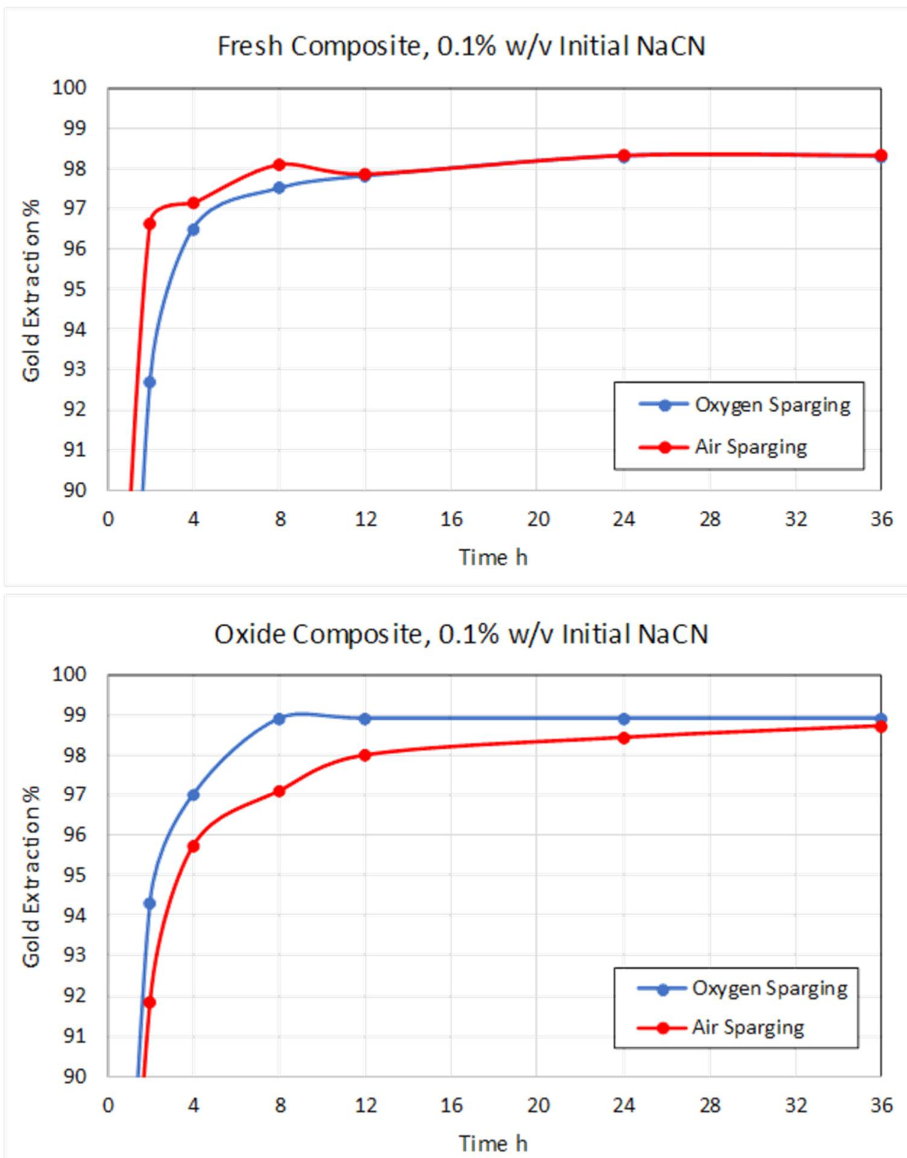


Figure 13-19: 2019 Testwork - Master Composite Oxygen vs. Air

13.7.4.4 Gravity Tails Leach Optimisation Conclusions

Gravity tails leaching kinetics were fast for all scenarios tested with little further gold extraction occurring after 12 hours. A residence time of 24-hours should provide adequate margin for maximising recoveries in practice.

- Cyanide usage was low in all tests, but consumption does increase with increasing solution NaCN concentration. No impact on leach kinetics or overall extraction with decreasing cyanide addition was evident in these tests. Operation at minimum cyanide concentration is attractive to minimise operating costs, but common practice is to maintain a minimum concentration of 150 ppm of free cyanide (NaCN) to ensure maximum gold leach extractions.
- Fresh ore operation at slurry densities of 55% w/w solids showed no decrease in leach kinetics or recovery suggesting relatively low slurry viscosities and high mass transfer rates. The oxide composite tested showed similar tolerance for increased slurry density up to 45% w/w solids. Operation on fresh ore feeds blended with low (20 to 30)% oxide/transition fractions is not expected to affect operating density significantly.
- Oxygen uptake rates by the slurries appeared low with negligible improvement in leach kinetics or gold extraction with oxygen sparging in place of air.

Recommended leaching conditions selected were thus:

- A cyanide concentration of 0.025% w/v NaCN (allowed to decay to 0.02%).
- A leach feed slurry density of 55% w/w solids for fresh ore. Blended ore leach % solids may be dependent on the slurry viscosity, but likely to be unchanged.
- Air sparged leaching with a residence time of 24-hours.

13.7.5 Master Composite Bulk Leach Testwork

Demonstration tests using the optimised leach conditions were run as bulk leaches on the fresh and oxide bulk gravity tails to prepare leach tails samples for physical characterisation testwork to determine engineering design parameters.

The oxide bulk leaching conditions were similar to the fresh sample, except a lower leach slurry density of 40% w/w solids was used. In practice, when oxide is blended with low viscosity fresh ore, higher leach densities may be suitable.

The bulk leach testing results are summarised in Table 13-18 and Figure 13-20. For both the fresh and oxide samples, the bulk leach kinetics were slower than those for the individual optimisation tests and the final gold extraction (24-hours) was up to 1% lower.

None of the optimisation tests were conducted under the same combination of conditions as the bulk leaches, so the results are compared to tests with either the same per cent solids, same cyanide addition or with air sparging. The comparisons indicate that:

- For the fresh ore, a higher cyanide concentration (as per the air sparging and density optimisation tests) would have improved leach kinetics and final gold extraction (Figure 13-20).
- For the oxide ore, a higher cyanide concentration or oxygen sparging (as per the air sparging and cyanide optimisation tests) would have improved leach kinetics and final gold extraction.

Increasing the leach cyanide concentration slightly would provide additional free cyanide to drive the leach reaction without increasing the required cyanide addition significantly. With increasing leach slurry density, the cyanide mass added to achieve the same solution concentration is significantly reduced, as shown in Table 13-19 following.

Table 13-19: Cyanide Addition Rate as a function of Slurry Solids Concentration

Leach Slurry Percent Solids w/w	40%	55%	55%	55%
Leach Slurry Cyanide Concentration, ppm NaCN	250	250	300	350
Required Leach Cyanide Addition, kg NaCN/t ore	0.38	0.20	0.25	0.29

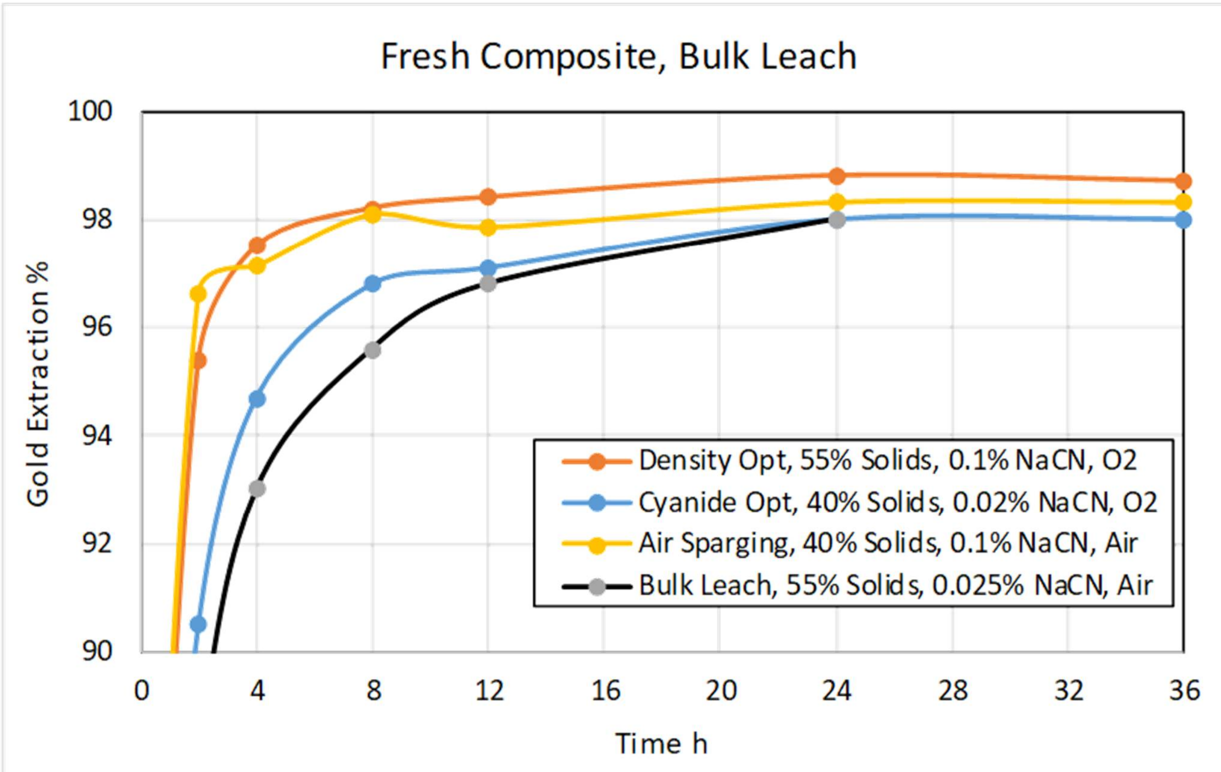
Alternatively, provision of additional residence time implies additional capital investment, but there is very little ongoing operating cost. If this base mitigation measure is implemented, more flexible process changes such as increasing the grind fineness or cyanide concentration can be applied as required.

13.7.5.1 Bulk Leach Tails Testing

The bulk leach tails were utilised for downstream testwork including sequential triple contact carbon loading testwork, viscosity testwork, cyanide detoxification testwork, and vendor thickener testwork on the fresh master composite.

A blended slurry composite (40% oxide/60% fresh) was made-up from the fresh and oxide bulk leach tails. This composite was utilised for downstream viscosity testwork, cyanide detoxification testwork and vendor thickener testwork to provide data for the highest oxide blend envisaged at this stage. Subsequent adoption of the HPGR comminution circuit placed a further constraint on maximum oxide blend allowed.

Bulk solution assays for the fresh and blended leach tails samples are presented in Table 13-39, being the feed to the cyanide detoxification testwork.



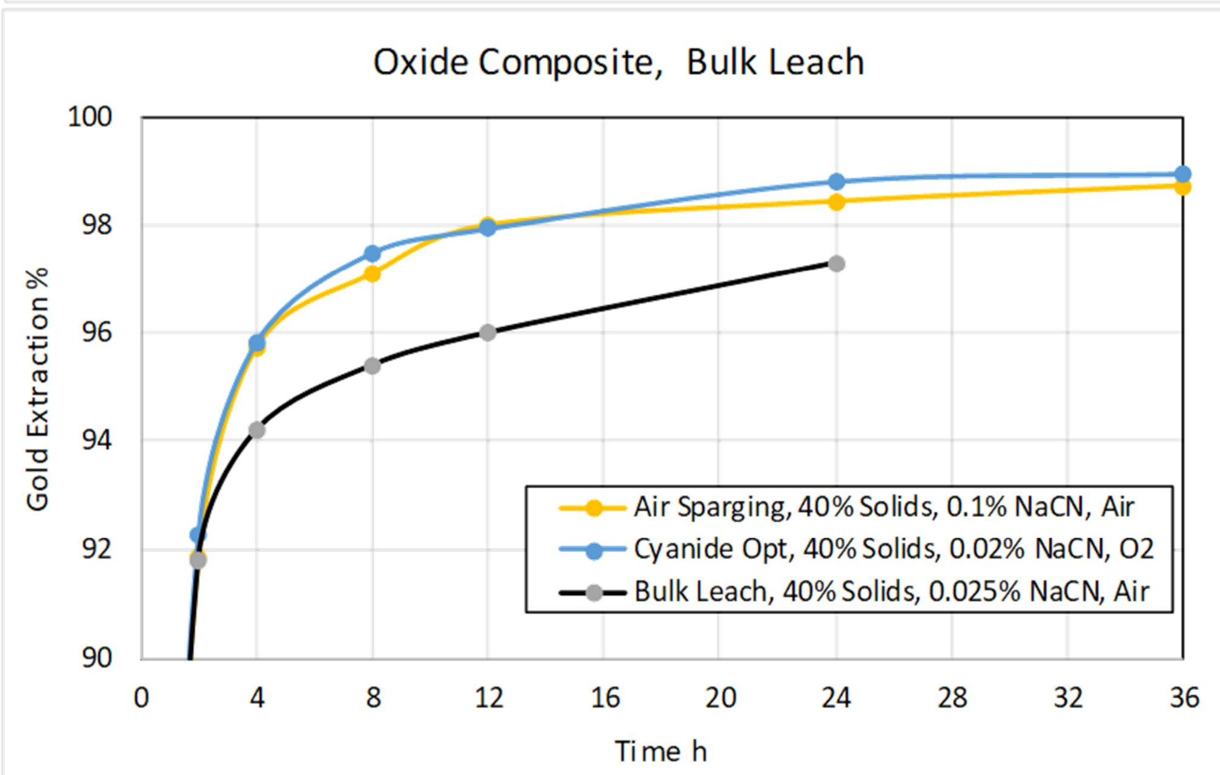


Figure 13-20: 2019 Testwork Master Composite Bulk Gravity Tails Leach Testwork

13.7.6 Master Composite Direct Leach (No Gravity) Testwork

The effect on overall gold extraction of inclusion of a gravity concentration stage prior to leaching was evaluated to confirm that this was an essential addition to the flowsheet.

A whole of ore leach test (without removal of the gravity gold prior to leaching) was conducted on the fresh master composite. The test results are included in Table 13-18 and shown in Figure 13-21.

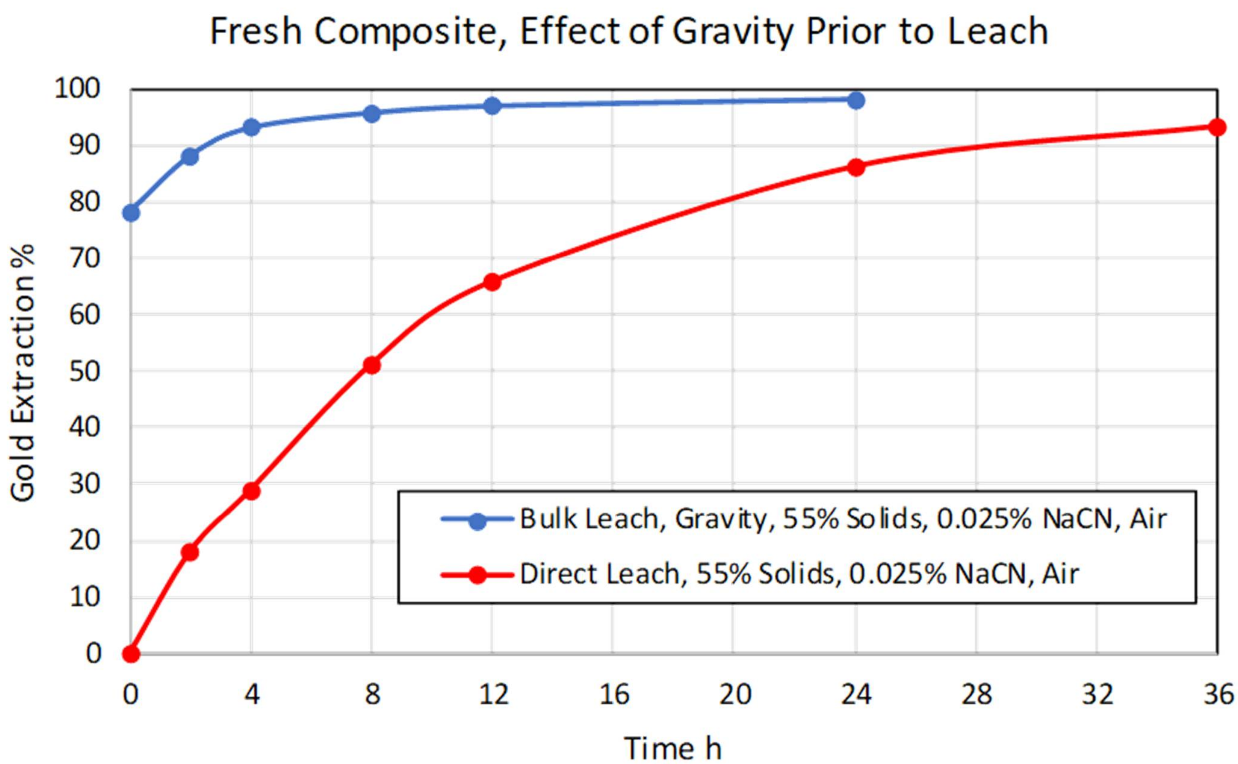


Figure 13-21: 2019 Testwork Fresh Master Composite Whole of Ore Leach Testwork

Without removal of the coarse gravity gold, leaching is very slow as indicated by the profile with a lower overall recovery after 36 hours than was achieved in 12 hours with gravity gold removal. The bulk gravity tails leach curve is shown for comparison as leach conditions were the same with air sparging and reduced cyanide addition. This performance strongly supports the inclusion of a robust gravity recovery stage and operation of this circuit at all times.

The recovery of a gravity concentrate for intensive cyanidation not only accelerates the cyanidation of the coarse gold, but is also likely to recover the higher density telluride and base metal sulphide host minerals. Intensive cyanidation conditions will more successfully recover gold associated with, or mostly enclosed in, these hosts than conventional cyanidation.

Cyanide and lime consumption was comparable for the tests with and without gravity gold recovery.

13.7.7 High-Grade Fresh Master Composite Gravity Leach Testwork

A demonstration test utilising the optimised leach conditions was run on the high-grade fresh composite. This composite was selected assuming that the very high-grade samples would prove more challenging to leach than the average ores because the mineralogy and associated elements might require more intensive treatment. In practice, and as demonstrated by the mineralogical examination, this sample was a clean, free milling gold sample with no deleterious elements and is likely typical of the Lafigué high grade mineralisation.

The high-grade fresh composite gravity/leach testing results are summarised in Table 13-18 and shown in Figure 13-22. Parallel tests were conducted with air and oxygen sparging to assess the oxygen benefits.

The high-grade fresh composite leach test results indicate that the gold content comprised almost all free grains (92% gravity gold recovery) with little gold locked in background mineralisation and apparent leach residue differences only resulting from rounding. The calculated head was, however, very different from the originally assayed value.

Both air and oxygen sparging provided similar gold extraction. Cyanide consumption was very low in both tests, but consumption was notably lower for the oxygen sparged leach, suggesting the presence of a reactive mineral (e.g., pyrrhotite) that was passivated by more intensive oxidation.

A reasonable fraction of the gold mineralisation in the Lafigué resource occurs as high-grade intercepts. This result suggests that very high recoveries and low reagent consumption is likely for this high-grade style of mineralisation.

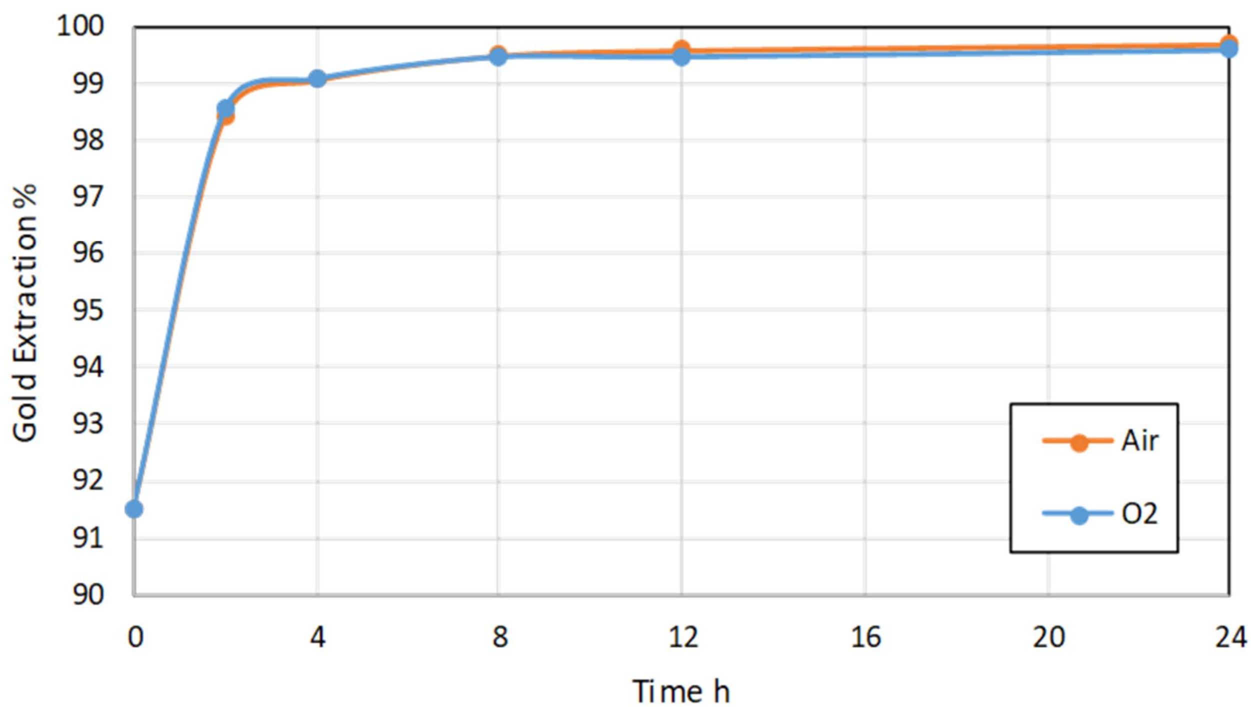


Figure 13-22: 2019 Testwork High-Grade Fresh Master Composite Gravity Leach Testwork

13.8 2019 Programme Ancillary Leach Testing

13.8.1 Preg-Robbing Assessment

An evaluation of the preg-robbing potential of the Lafigué composites was believed to be unnecessary given the consistently high gold leach extractions and the absence of organic carbon in the ore, as shown in Table 13-14.

The high correlation between fire assay results and BLEG testing over a wide cross section of samples confirms that preg-robbing will not be a concern for Lafigué ores.

13.8.2 Heap Leach Amenability Testing

Indicative heap leach amenability testwork was completed as an alternative processing approach to milling/CIL. The samples tested were not ideal as the work was requested after the bulk of the sample preparation had occurred. The results of the coarse bottle roll tests demonstrated that, for the samples tested, gold extraction was relatively high (~70% leach extraction), but leach recovery profiles were very slow given the coarse gold content. The significantly higher milling/CIL gold extractions (typically >95%) justify the additional processing costs.

13.9 2019 Programme Variability Testwork

Confirmatory testing using the optimised leach conditions established for the master composites was conducted on the 23 fresh and six oxide variability composites which had been selected to represent the various ore lithologies, weathered states, grade ranges and mineralisation styles of the 2019 Resource.

Variability composite samples for testing included most of the intercept composites selected on site for metallurgical testing. The majority of the composites were used to make up the master composites as described in Section 13.6.1 with a number of selected samples being set aside for variability testing only. Intercept Composites #2 and #4 were combined to make variability Composite #2A. Intercept Composites #12 and #13 made up the HG (high grade) fresh composite. Intercept Composites #27, #30 and #31 were excluded as these came from holes already represented by multiple samples.

13.9.1 Variability Composite Head Assays

Triplicate gold analyses were performed by standard fire-assay with the balance of the elements being determined by ICP scan or standard assay techniques. Variability composite details and key element assays are summarised in Table 13-20.

The fresh variability samples cover a gold grade range from 0.55 to 7.8 g/t Au (excluding the HG composite). Some samples had reasonably consistent triplicate gold assays suggesting more disseminated fine gold, but most of the samples had significant variation in the triplicate assay due to the presence of coarse gold.

Deleterious element levels are generally low, with very low base metal content. There is little correlation between the sulphide content in the fresh samples and the base metals and iron content, with typically low sulphide levels in all samples. Silver grades were all below the minimum ICP detection limit of 2 ppm.

Mercury levels were generally below the detection limit of 0.1 ppm although isolated intercepts were higher, up to 1.0 ppm.

The oxide variability samples selected have two high grade intercepts at 22.8 g/t Au (#22) and 7.1 g/t Au (#25), with the rest of the sample grades ranging from 1.1 to 1.7 g/t Au. Deleterious element levels are generally low. Silver grades were below the minimum ICP detection limit of 2 ppm with the exception of high-grade sample #22 with 6 g/t Ag.

13.9.2 Variability Composite Gravity Leach Test at Standard Conditions

Gravity/leach tests using the optimised conditions were conducted on the variability composites; grind size of P_{80} 106 µm, gravity concentration using amalgamation, 55% solids (fresh) or 40% solids (oxide), air sparging, initial 0.025% w/v NaCN maintained above 200 ppm NaCN and 24-hour leach.

The results of the variability tests are presented in Table 13-21 and Figure 13-23.

Overall gold extraction from the variability samples was fairly consistent and moderately well aligned with the master composite results. Five of thirty-five samples had overall gold extractions of less than 92%. These samples displayed slower leach kinetics with leaching continuing through to the end of the test (24 hours). These appear to be isolated instances with localised mineral associations that impact some aspect of the leach chemistry, but are not evident in the sample assays. When combined in the master composites, these impacts become insignificant and gold leach extractions below 98% are rare.

Average fresh ore residue grades are only slightly higher than the bulk leach residue result. Average residue grades for the oxide samples are significantly lower than the bulk leach residue grade.

The oxide gold ores are generally free milling with high extractions, particularly when there is high gravity recoverable gold.

As noted in the previous testwork, calculated head gold grades were generally higher than assayed heads due to the high amount of coarse gold present. Cyanide and lime consumption were similar to the master composite usage rates. Lime usage was notably higher for the oxide ores than the fresh ores, suggesting the presence of clays with some buffering effects.

Gold leach kinetics profile ranges are shown in Figure 13-24 to indicate the trends observed. The median leach extraction profiles for the fresh and oxide variability composites are plotted along with the 15^{th} and 85^{th} percentile profiles. The bulk leach results for the master composites are superimposed on each graph for comparison. The spread in the fresh ore kinetics is significantly greater than for the closely grouped oxide samples.

The fresh ore 85^{th} percentile results are similar to the bulk leach with the median being lower and the lower bound indicating slower leach kinetics and lower overall gold extractions. This contradicts the consistently high fast kinetics and leach extractions achieved in the master composite testwork.

The oxide variability results displayed a more expected trend with the median reflecting the bulk leach extraction profile with the ±one standard deviation curves close to the median, with similar final extractions in all cases.

Table 13-20: 2019 Testwork Variability Composite Detailed Head Assay

Variability ID	Drillhole ID	From m	To m	Head Assay											
				Au_{1-3} ppm	Au_{Ave} ppm	As ppm	C_{org} %	Cu ppm	Fe %	Hg ppm	Ni ppm	S^{2-} %	Sb ppm	Te ppm	Zn ppm
Var #1	LFDD17-233	152	162.6	3.52/3.72/2.66	3.3	<10	0.03	60	6.3	0.2	100	<0.02	0.2	2.0	120
Var #2a(#2 & #4)	LFDD17-243	199	205.4	3.37/1.93/2.28	2.53	20	<0.03	70	7.28	<0.1	25	0.46	0.3	0.4	102
	LFDD18-252	82.1	90												
Var #3	LFDD17-243	223.9	233.2	6.31/5.22/10.4	7.31	<10	<0.03	54	6.76	0.1	90	<0.02	0.2	0.2	118
Var #5	LFDD18-400	80.2	87.4	0.8/0.76/0.64	0.73	30	<0.03	82	8.7	<0.1	55	0.38	0.2	0.2	94
Var #6	LFDD18-400	97.9	106.9	1.14/1.6/1.15	1.3	20	0.06	76	8.94	<0.1	35	0.44	0.5	0.2	86
Var #7	LFDD18-404	100.8	107.6	2.18/0.59/0.55	1.11	<10	0.03	96	8.6	<0.1	50	0.24	0.1	<0.2	102
Var #8	LFDD18-404	111.2	115.7	0.75/0.63/0.65	0.68	<10	<0.03	118	13	0.4	25	0.68	0.3	<0.2	122
Var #9	LFDD18-405	164	169.7	2.68/1.35/2.27	2.1	<10	0.03	228	8.38	<0.1	35	0.62	0.1	0.4	98
Var #10	LFDD18-409	115.1	126.8	2.03/5.27/5.52	4.27	<10	<0.03	46	6.66	<0.1	65	0.02	0.2	0.6	108
Var #11	LFDD18-409	126.8	138	2.99/3.78/3.56	3.44	<10	<0.03	42	2.84	0.4	10	0.08	0.2	0.4	96
HG Comp(#12 & #13)	LFDD18-410	173.75	182.1	56.5/22.4/19.6	32.8	<10	<0.03	44	6.58	0.5	100	<0.02	0.2	4.8	114
	LFDD18-410	182.1	193												
Var #14	LFDD18-412	342.6	351	0.92/0.61/0.44	0.66	<10	<0.03	48	2.2	<0.1	5	0.18	<0.1	<0.2	44
Var #15	LFRCDD18-359	300.4	304.7	4.29/4.8/2.66	3.92	<10	<0.03	10	2.88	<0.1	20	<0.02	0.1	1.6	64
Var #16	LFRCDD18-361	258.8	270.6	0.75/1.41/10.1	4.09	<10	<0.03	28	5.32	0.1	70	<0.02	<0.1	0.4	122
Var #17	LFRCDD18-361	270.6	272.7	0.89/0.82/1.11	0.94	<10	<0.03	12	9.88	<0.1	145	<0.02	0.1	<0.2	330
	LFRCDD18-361	277.7	282.95												
Var #18	LFRCDD18-361	305.4	315.4	2/2.71/2.02	2.24	30	<0.03	52	8.28	<0.1	100	<0.02	0.2	1.0	128
Var #19	LFRCDD18-362	266.55	273.2	1.15/1.27/0.6	1.01	<10	<0.03	40	2.76	<0.1	5	0.16	<0.1	0.2	52
Var #20	LFRCDD19-441	245	252.7	1.74/3.02/2.8	2.52	<10	<0.03	4	1	1	5	<0.02	<0.1	4.8	24

Table 13-20: 2019 Testwork Variability Composite Detailed Head Assay

Variability ID	Drillhole ID	From m	To m	Head Assay											
				Au_{1-3} ppm	Au_{Ave} ppm	As ppm	C_{org} %	Cu ppm	Fe %	Hg ppm	Ni ppm	S^{2-} %	Sb ppm	Te ppm	Zn ppm
Var #21	LFRCDD19-459	181.4	190	0.97/1.5/1.38	1.28	<10	<0.03	150	9.42	<0.1	40	0.34	0.1	0.4	136
Var #22 [Ox]	LFDD19-670	0	7.3	17.1/23/27.9	22.67	<10	0.09	92	9.34	<0.1	105	<0.02	0.3	0.4	168
Var #23 [Ox]	LFDD19-670	7.3	10.4	0.68/1.21/1.45	1.11	<10	0.03	84	8.88	<0.1	135	<0.02	0.2	0.4	124
Var #24 [Ox]	LFDD19-671	0.9	7.7	1.31/1.39/1.34	1.35	40	0.06	82	8.36	<0.1	50	<0.02	0.5	0.4	118
Var #25 [Ox]	LFDD19-668	5.4	11.3	10.3/3.03/7.92	7.08	<10	0.03	148	2.4	<0.1	15	<0.02	<0.1	0.2	114
Var #26 [Ox]	LFDD19-669	3.4	6.7	1.07/0.74/1.62	1.14	10	0.06	28	9.02	<0.1	45	<0.02	0.2	<0.2	58
Var #28 [Ox]	LFDD19-669	13.7	17.2	0.33/0.48/4.15	1.65	<10	0.03	46	14.9	<0.1	5	<0.02	0.1	<0.2	154
Var #29	LFRCDD19-667	62	66	0.45/0.42/0.77	0.55	<10	<0.03	128	2.7	0.2	50	0.34	<0.1	<0.2	222
Var #32	LFRCDD19-667	90	95	8.9/8.17/6.26	7.78	<10	<0.03	86	2.5	0.3	5	0.32	0.2	1.0	86
Var #33	LFRCDD19-667	95	99	1.64/1.4/1.95	1.66	10	0.03	20	2.42	<0.1	5	0.32	0.2	0.6	72
Var #34	LFRCDD19-667	99	103.5	1.67/1.48/2.14	1.76	<10	0.03	24	2.78	0.2	5	0.42	0.1	0.8	86
Var #35	LFRCDD19-667	103.5	109.2	1.24/1.28/2.25	1.59	<10	<0.03	78	7.46	0.2	70	0.16	<0.1	<0.2	122

Figure 13-20 note: Ag grades are all below the detection limit of 2 ppm except for #22 which recorded 6 ppm.

Table 13-21: 2019 Testwork Variability Composite Gravity Leach Testwork at Standard Conditions

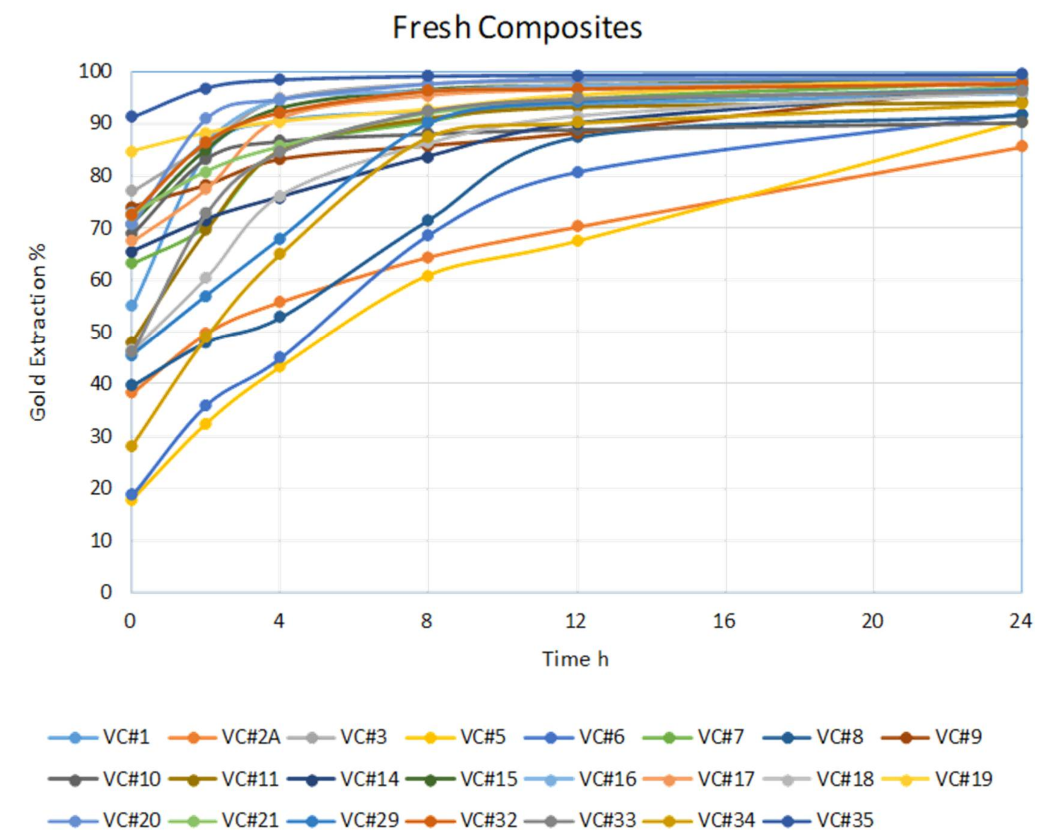
Test ID	Variability Comp No.	Test	Grind Size P_{80} µm	Initial NaCN % w/v	Leach % Solids w/w	Sparge Gas	Avg Head Au g/t	Calc. Head Au g/t	Res. Au g/t	% GRG	Overall % Gold Extraction @ Time hrs					Reag. Cons. kg/t	
											2	4	8	12	24	NaCN	Lime
JS4543	1	Std Cond	106	0.025	55	Air	3.30	3.31	0.14	55	84	91	92	94	95.8	0.06	0.35
JS4544	2A	Std Cond	106	0.025	55	Air	2.53	1.88	0.27	38	50	56	64	70	85.7	0.09	0.40
JS4545	3	Std Cond	106	0.025	55	Air	7.31	5.65	0.05	77	86	95	98	98	99.1	0.06	0.30
JS4546	5	Std Cond	106	0.025	55	Air	0.73	0.73	0.07	18	32	43	61	67	90.4	0.06	0.55
JS4547	6	Std Cond	106	0.025	55	Air	1.30	1.09	0.09	19	36	45	69	81	91.8	0.09	0.50
JS4548	7	Std Cond	106	0.025	55	Air	1.11	0.96	0.02	63	70	85	90	95	97.9	0.06	0.35
JS4549	8	Std Cond	106	0.025	55	Air	0.68	0.83	0.07	40	48	53	72	87	91.6	0.15	0.75
JS4550	9	Std Cond	106	0.025	55	Air	2.10	3.44	0.08	74	78	83	86	88	97.7	0.09	0.30
JS4551	10	Std Cond	106	0.025	55	Air	4.27	2.14	0.21	69	83	87	88	89	90.2	0.04	0.30
JS4552	11	Std Cond	106	0.025	55	Air	3.44	3.78	0.22	48	70	85	91	93	94.2	0.05	0.30
JS4553	14	Std Cond	106	0.025	55	Air	0.66	0.92	0.03	65	72	76	84	90	96.8	0.06	0.20
JS4554	15	Std Cond	106	0.025	55	Air	3.92	3.64	0.05	71	85	93	96	97	98.6	0.05	0.20
JS4555	16	Std Cond	106	0.025	55	Air	4.09	1.33	0.03	73	87	95	96	97	97.7	0.07	0.30
JS4556	17	Std Cond	106	0.025	55	Air	0.94	0.61	0.01	67	77	91	95	97	98.4	0.06	0.25
JS4557	18	Std Cond	106	0.025	55	Air	2.24	1.78	0.07	46	60	76	86	91	96.1	0.06	0.25

Table 13-21: 2019 Testwork Variability Composite Gravity Leach Testwork at Standard Conditions

Test ID	Variability Comp No.	Test	Grind Size P_{80} µm	Initial NaCN % w/v	Leach % Solids w/w	Sparge Gas	Avg Head Au g/t	Calc. Head Au g/t	Res. Au g/t	% GRG	Overall % Gold Extraction @ Time hrs					Reag. Cons. kg/t	
											2	4	8	12	24	NaCN	Lime
JS4558	19	Std Cond	106	0.025	55	Air	1.01	1.97	0.02	85	88	90	93	95	99.0	0.07	0.20
JS4559	20	Std Cond	106	0.025	55	Air	2.52	6.86	0.11	71	91	95	98	98	98.4	0.08	0.20
JS4560	21	Std Cond	106	0.025	55	Air	1.28	2.49	0.08	73	81	86	92	94	96.8	0.07	0.30
JS4567	29	Std Cond	106	0.025	55	Air	0.55	0.89	0.03	45	57	68	90	94	96.6	0.05	0.20
JS4568	32	Std Cond	106	0.025	55	Air	7.78	7.90	0.18	73	86	92	96	97	97.7	0.05	0.20
JS4569	33	Std Cond	106	0.025	55	Air	1.66	1.88	0.07	46	73	85	92	95	96.3	0.05	0.20
JS4570	34	Std Cond	106	0.025	55	Air	1.76	1.92	0.12	28	49	65	87	90	93.8	0.07	0.20
JS4571	35	Std Cond	106	0.025	55	Air	1.59	3.33	0.02	91	97	98	99	99	99.4	0.04	0.25
JS4561	22 [OXIDE]	Std Cond	106	0.025	40	Air	22.7	23.1	0.44	70	89	92	95	96	98.1	0.08	2.30
JS4562	23 [OXIDE]	Std Cond	106	0.025	40	Air	1.11	1.67	0.02	70	95	96	98	98	98.8	0.11	6.70
JS4563	24 [OXIDE]	Std Cond	106	0.025	40	Air	1.35	1.50	0.06	48	81	88	91	92	96.0	0.14	2.80
JS4564	25 [OXIDE]	Std Cond	106	0.025	40	Air	7.08	7.53	0.02	87	97	99	99	100	99.7	0.08	0.60
JS4565	26 [OXIDE]	Std Cond	106	0.025	40	Air	1.14	1.19	0.05	29	79	85	89	92	95.8	0.08	1.80
JS4566	28 [OXIDE]	Std Cond	106	0.025	40	Air	1.65	0.50	0.02	50	92	93	96	96	96.0	0.08	2.15

Table 13-21: 2019 Testwork Variability Composite Gravity Leach Testwork at Standard Conditions

Test ID	Variability	Test	Grind Size	Initial NaCN	Leach %	Sparge	Avg Head	Calc. Head	Res.	% GRG	Overall % Gold Extraction @ Time hrs					Reag. Cons. kg/t	
	Comp No.		P_{80} µm	% w/v	Solids w/w	Gas	Au g/t	Au g/t	Au g/t		2	4	8	12	24	NaCN	Lime
Average Fresh							2.47	2.58	0.09	58.0	58	71	80	88	91	95.6	0.31
Median Fresh							1.76	1.92	0.07	65.4	77.5	65	77	85	91	94	96.8
15th Percentile Fresh							0.80	0.90	0.02	38.7	49.2	39	49	59	75	88	91.6
85th Percentile Fresh							4.04	3.74	0.17	73.7	86.8	74	87	94	96	97	98.6
Average Oxide							5.83	5.91	0.10	59.0	88.8	59	89	92	95	96	97.4
Median Oxide							1.50	1.59	0.04	60.0	90.3	60	90	93	95	96	97.1
15th Percentile Oxide							1.14	1.02	0.02	43.1	80.7	43	81	87	90	92	96.0
85th Percentile Oxide							11.0	11.4	0.16	74.4	95.5	74	96	97	98	98	99.0
JJ4501	Fresh MC	Bulk Leach	106	0.025	55	Air	2.05	3.00	0.06	78	88	93	96	97	98.0	0.08	0.23
JS4513	Oxide MC	Bulk Leach	106	0.025	40	Air	3.06	4.78	0.13	72	92	94	95	96	97.3	0.25	2.46



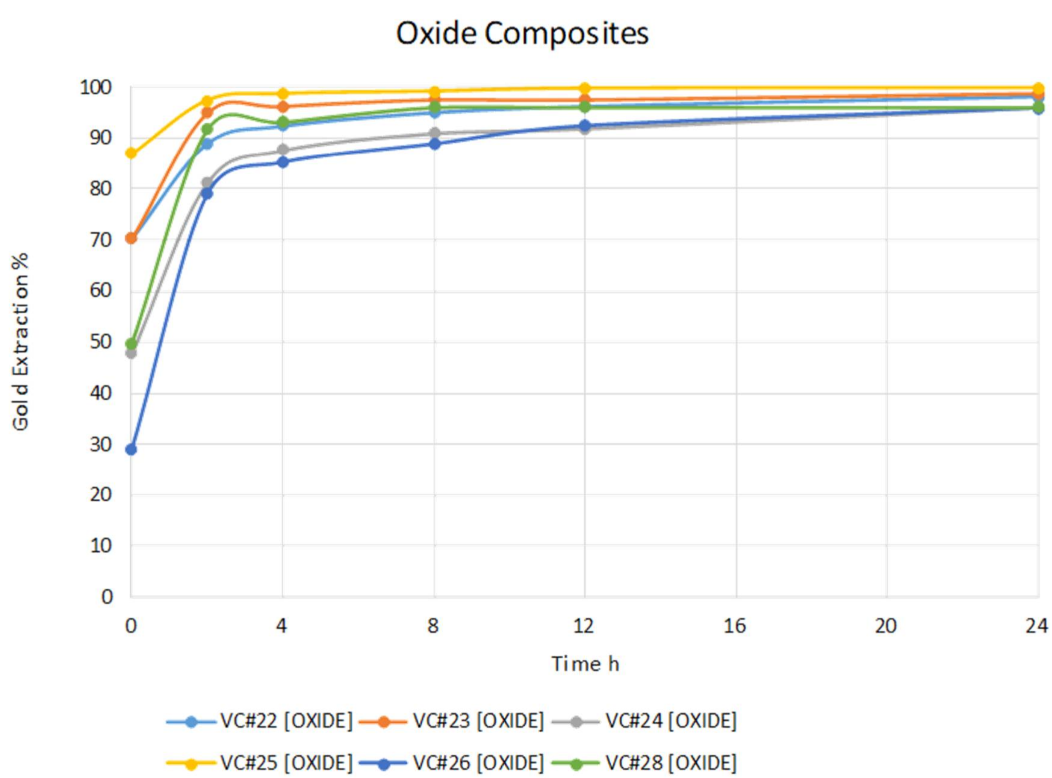
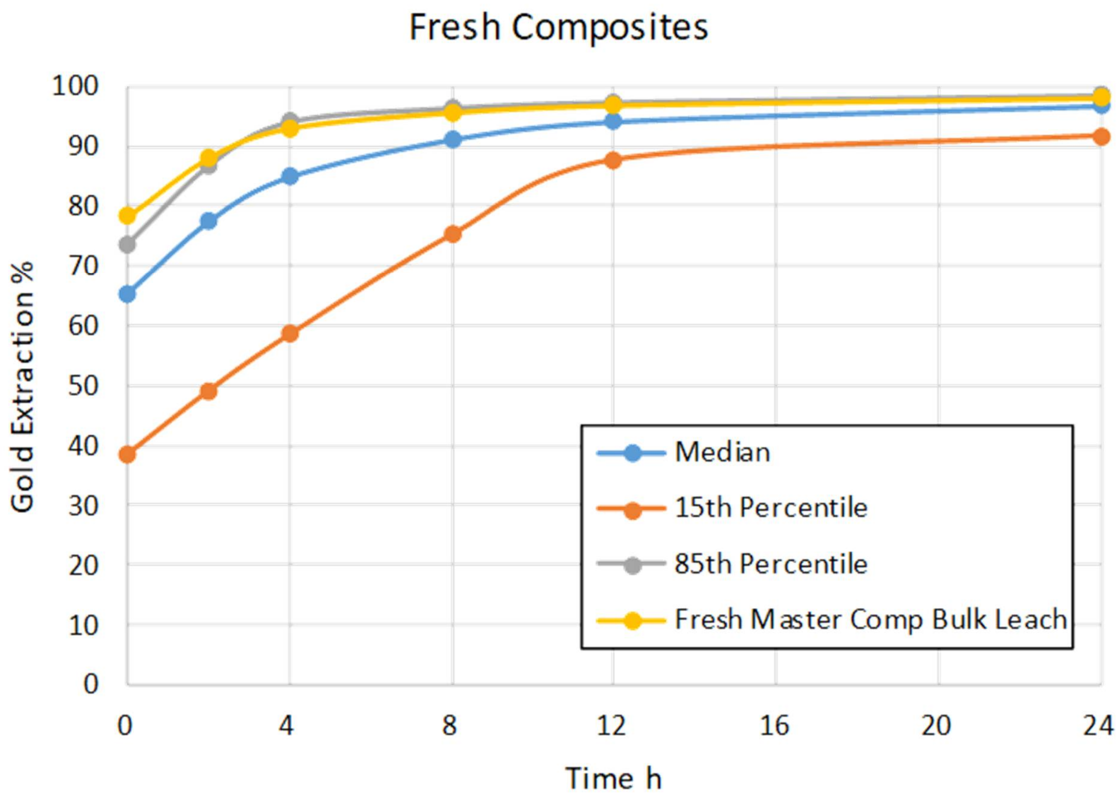


Figure 13-23: 2019 Testwork Variability Gravity/Leach Testwork at Standard Conditions



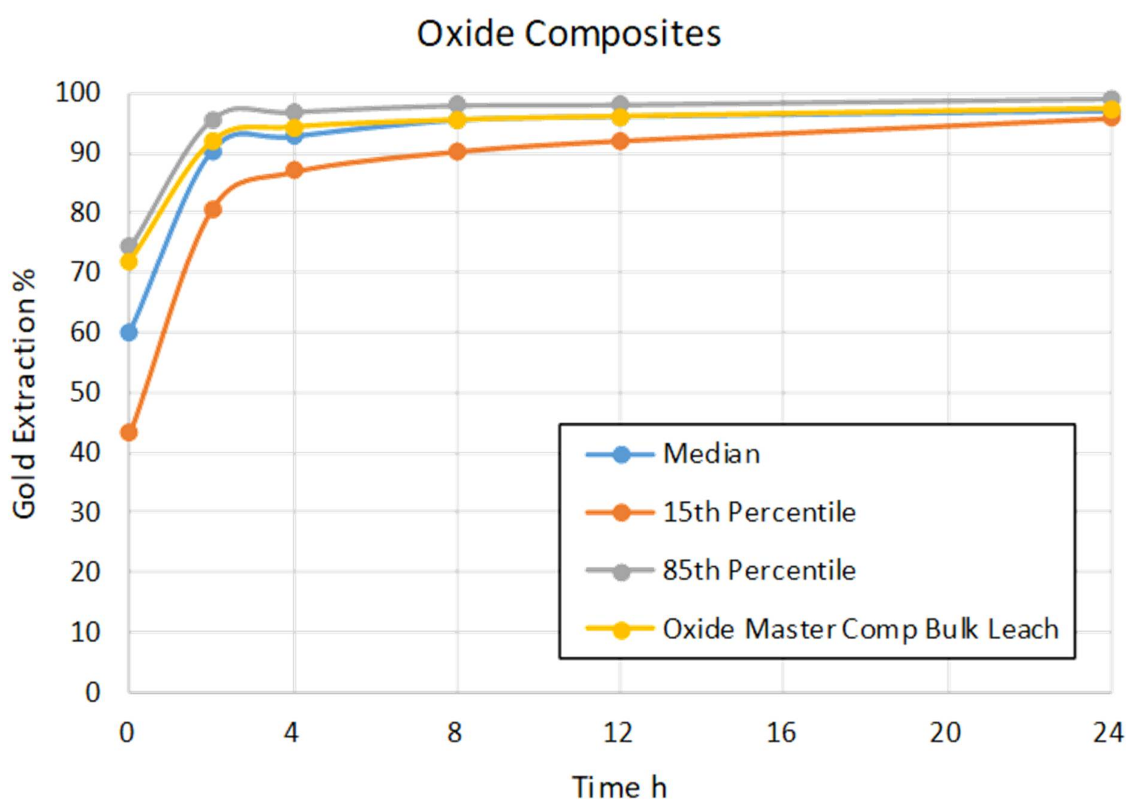


Figure 13-24: 2019 Testwork Variability - Comparative Gravity/Leach Test Profiles

13.9.3 2019 Testwork - Repeat Variability Composite Testwork

Additional testwork was completed to investigate the slower leach kinetics observed in some of the variability composites. Test results were grouped into three subsets based on the leach extraction profiles. These groups were 'lower gold extraction (<95%)', 'slow leach kinetics with mixed final extractions', and samples with 'high gold extractions with fast kinetics'.

The slower kinetics/lower extraction leach composites all had rising leach curves continuing to the end of the 24-hour test and were expected to achieve higher gold extractions with extended times or more aggressive leaching conditions that increase kinetics.

It was assumed that reactive pyrrhotite, consuming oxygen and cyanide, may be responsible for the slower leaches. Tellurium and silver are present in some gold associations as well and these are also known to impact leach kinetics. Also, as demonstrated in the bulk leach tests (Section 13.7.5), leaching at high slurry percent solids and low solution cyanide levels may result in lower free cyanide availability, possibly leading to slower leaching. Little evidence of these deficiencies was measured in the intermediate solution sampling or overall reagent consumption, but the kinetics were definitely affected.

A series of repeat tests was conducted trialling:

- pre-oxidation conditioning (using air) with lime
- increased cyanide addition
- increased leach residence time
- finer grinding to P_{80} 75 µm
- oxygen sparging.

The repeat testwork results are summarised in Table 13-22. Results for the standard condition tests are included for comparison.

The test results indicate:

- A higher cyanide addition (0.035% w/v NaCN maintained above 0.03% w/v NaCN) and increased leach time (36 hours) were sufficient to enhance the kinetics and overall leach gold extraction.
- Pre-oxidation generally reduced the cyanide consumption but did not always improve extraction.
- A higher cyanide addition with 24-hours residence time appears sufficient but increasing the residence time provides additional risk mitigation and caters for increased plant throughput on softer ores or lower operating leach densities.
- Finer grinding and oxygenation also increased leach kinetics but are considered unnecessary added operating costs when increased cyanide addition and extra residence time can be more readily and cost effectively implemented.

Selected repeat test gold leach kinetics are presented in Figure 13-25, along with the relevant test results at standard conditions. Solid lines indicate the baseline result with the same colour dashed line illustrating the enhanced leach extraction and kinetics.

Table 13-22: 2019 Testwork Variability Gravity/Leach Repeat Testwork Results

Test	Variability	Test	Grind	NaCN	Solids	Sparge	Avg Head	Calc. Head	Res.	GRG	Overall % Gold Extraction @ Time Hours						Reag. Cons. (kg/t)		% Au Ext'n Incr.	
ID	Comp No.	Condition	P_{80} µm	% w/v	% w/w	Gas	Au g/t	Au g/t	Au g/t	%	2	4	8	12	24	36	NaCN	Lime	24 h	Overall
JS4544	2A	Standard	106	0.025	55	AIR	2.53	1.88	0.27	38	50	56	64	70	85.7		0.09	0.40		
JS4666	2A	4hr Pre-Ox, Inc. CN	106	0.071	55	AIR	2.53	2.06	0.09	47	69	84	93	94	94.6	95.6	0.24	0.62	9.0	10.0
JS4667	2A	Inc. CN	106	0.035	55	AIR	2.53	2.21	0.10	43	67	83	92	93	95.0	95.5	0.17	0.52	9.3	9.8
JS4669	2A	4hr Pre-Ox, Inc. CN	106	0.035	55	AIR	2.53	1.95	0.09	44	64	84	92	94	94.6	95.4	0.05	0.64	9.0	9.7
JS4546	5	Standard	106	0.025	55	AIR	0.73	0.73	0.07	18	32	43	61	67	90.4		0.06	0.55		
JS4581	5	4hr Pre-Ox	106	0.025	55	AIR	0.73	0.71	0.04	25	36	48	82	89	92.5	94.3	0.09	0.60	2.1	3.9
JS4547	6	Standard	106	0.025	55	AIR	1.30	1.09	0.09	19	36	45	69	81	91.8		0.09	0.50		
JS4661	6	4hr Pre-Ox	106	0.035	55	AIR	1.30	1.19	0.09	25	44	66	89	90	91.8	92.4	0.10	0.71	0.1	0.6
JS4662	6	Inc. CN	106	0.035	55	AIR	1.30	1.10	0.07	31	55	77	90	91	93.0	93.6	0.15	0.66	1.2	1.8
JS4550	9	Standard	106	0.025	55	AIR	2.10	3.44	0.08	74	78	83	86	88	97.7		0.09	0.30		
JS4582	9	4hr Pre-Ox	106	0.025	55	AIR	2.10	2.96	0.05	65	67	69	72	76	87.9	98.3	0.10	0.35	-9.8	0.6
JS4584	9	+Oxygen	106	0.025	55	OXY	2.10	1.72	0.05	72	82	87	93	96	96.9	97.1	0.11	0.40	-0.8	-0.6
JS4585	9	Inc. CN	106	0.035	55	AIR	2.10	3.00	0.01	84	85	86	89	94	99.0	99.7	0.08	0.54	1.4	2.0
JS4551	10	Standard	106	0.025	55	AIR	4.27	2.14	0.21	69	83	87	88	89	90.2		0.04	0.30		
JS4583	10	4hr Pre-Ox	106	0.025	55	AIR	4.27	1.55	0.06	62	76	86	93	95	95.9	96.1	0.07	0.40	5.7	5.9
JS4586	10	Finer Grind	75	0.025	55	AIR	4.27	2.23	0.03	77	87	93	97	98	98.7	98.7	0.07	0.40	8.5	8.5
JS4670	10	Inc. CN	106	0.035	55	AIR	4.27	2.16	0.08	70	84	92	94	95	95.5	96.3	0.06	0.26	5.3	6.1
JS4552	11	Standard	106	0.025	55	AIR	3.44	3.78	0.22	48	70	85	91	93	94.2		0.05	0.30		
JS4663	11	Inc. CN	106	0.035	55	AIR	3.44	4.34	0.15	56	73	88	95	95	96.1	96.5	0.08	0.30	1.9	2.4

Table 13-22: 2019 Testwork Variability Gravity/Leach Repeat Testwork Results

Test	Variability	Test	Grind	NaCN	Solids	Sparge	Avg Head	Calc. Head	Res.	GRG	Overall % Gold Extraction @ Time Hours						Reag. Cons. (kg/t)		% Au Ext'n Incr.	
ID	Comp No.	Condition	P_{80} µm	% w/v	% w/w	Gas	Au g/t	Au g/t	Au g/t	%	2	4	8	12	24	36	NaCN	Lime	24 h	Overall
JS4570	34	Standard	106	0.025	55	AIR	1.76	1.92	0.12	28	49	65	87	90	93.8		0.07	0.20		
JS4664	34	Inc. CN	106	0.035	55	AIR	1.76	2.01	0.07	30	53	78	95	95	96.2	96.5	0.10	0.26	2.4	2.8
JS4561	22 [OXIDE]	Standard	106	0.025	40	AIR	22.7	23.1	0.44	70	89	92	95	96	98.1		0.08	2.30		
JS4587	22 [OXIDE]	Inc. CN, Finer Grind	75	0.035	40	AIR	22.7	15.9	0.07	75	96	98	99	99	99.3	99.6	0.17	3.10	1.2	1.5

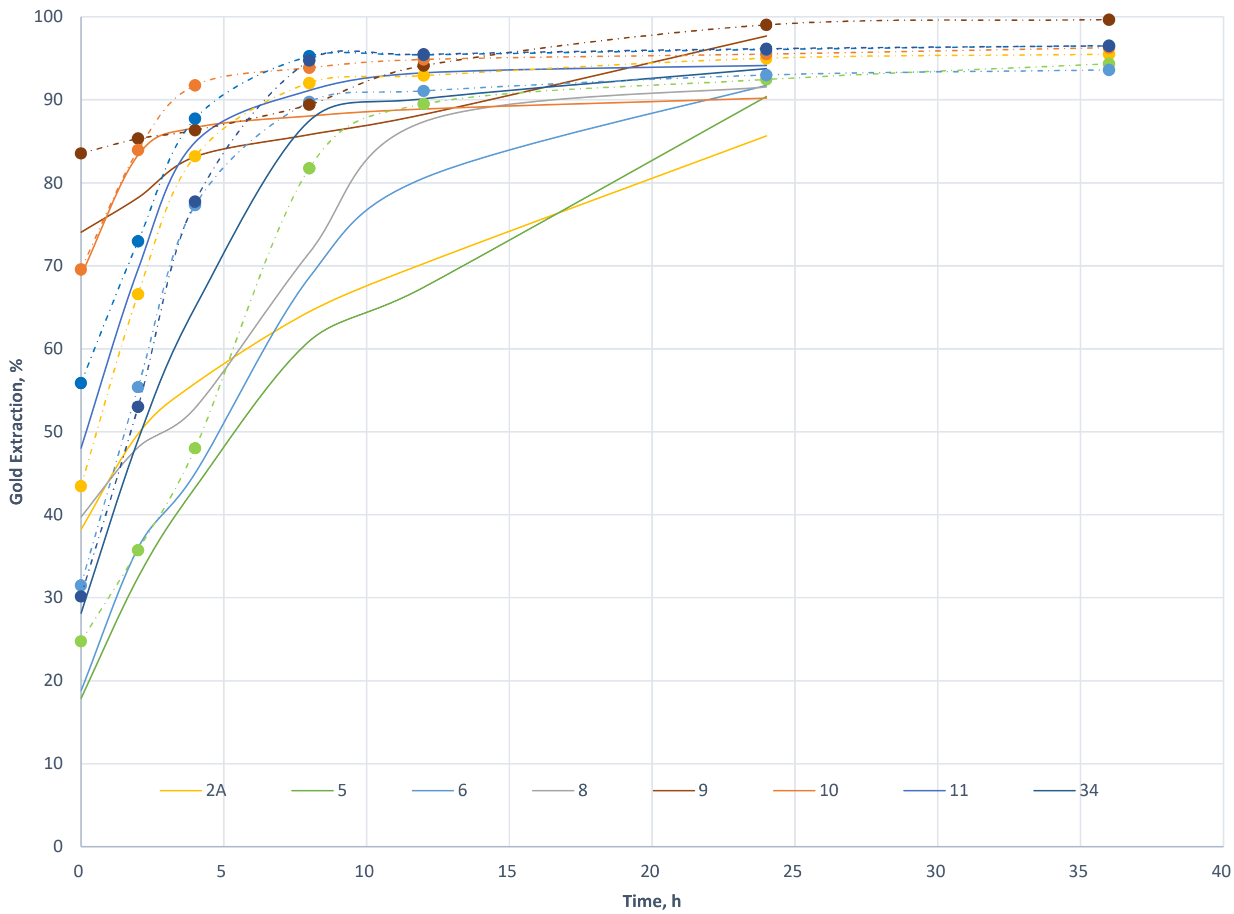


Figure 13-25: 2019 Testwork Standard and Repeat Variability Testwork Comparison

Table 13-25 notes to graph:

- Legend shows variability composite sample number.
- Repeat tests are graphed in the same colour with dashed lines.
- Sample #8 could not be repeat tested, due to insufficient mass.
- All repeated kinetics improved and recoveries at 24 hours were higher than the standard test.
- The longer design residence time is recommended as risk mitigation for the slower leaching examples as there is no conclusive finding regarding the causes of the slow leach kinetics and variability within sub-samples may be a factor here.

A full set of variability results, using the repeat tests where applicable, are presented in Table 13-23 and are the results used for estimation of the metallurgical recoveries and reagent consumptions.

The variability data set including the repeat tests, as presented in Figure 13-26, shows a closer grouping of the leach profile data compared to the standard conditions data set. It is noted that not all lower-performing composite samples had sufficient mass for repeat testing while the performance of some composites remained slightly below expectations. Only the underperforming samples were re-tested, so no change occurred in the upper percentile data.

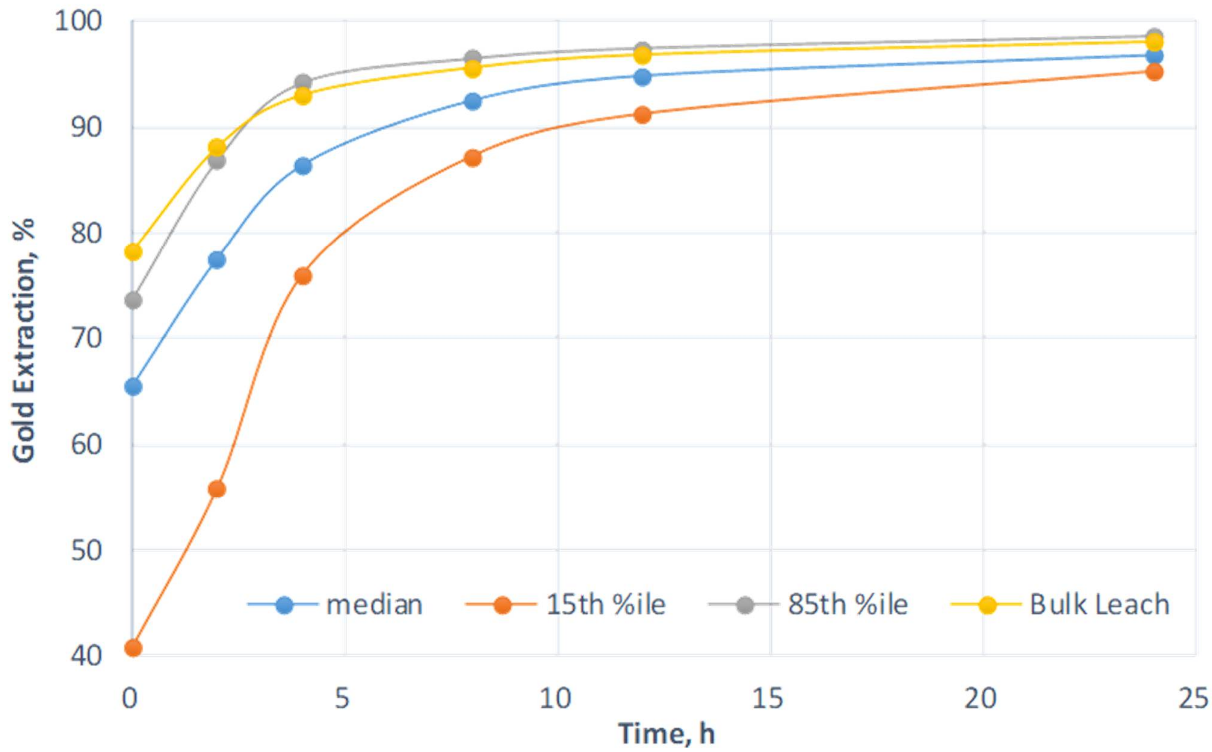


Figure 13-26: 2019 Testwork Gravity/Leach Variability Test Leach Profiles with Selected Repeated Tests

Table 13-23: 2019 Testwork Variability Gravity/Leach Results with Select Repeat Tests

Test	Sample	Calc. Head	Residue	% GRG	Overall % Gold Extraction @ Time Hours						Reagent Cons. kg/t	
No.	ID	g/t Au	g/t Au		2	4	8	12	24	36	NaCN	Lime
JS4543	#1	3.31	0.14	54.9	84	91	92	94	95.8		0.06	0.35
JS4667	#2A Rpt	2.21	0.10	43.4	67	83	92	93	95.0	95.5	0.17	0.52
JS4545	#3	5.65	0.05	77.0	86	95	98	98	99.1		0.06	0.30
JS4581	#5 Rpt	0.71	0.04	24.8	36	48	82	89	92.5	94.3	0.09	0.15
JS4662	#6 Rpt	1.10	0.07	31.5	55	77	90	91	93.0	93.6	0.15	0.66
JS4548	#7	0.96	0.02	63.2	70	85	90	95	97.9		0.06	0.35
JS4549	#8	0.83	0.07	39.7	48	53	72	87	91.6		0.15	0.75
JS4585	#9 Rpt	3.00	0.01	83.6	85	86	89	94	99.0	99.7	0.08	0.00
JS4670	#10 Rpt	2.16	0.08	69.6	84	92	94	95	95.5	96.3	0.06	0.26
JS4663	#11 Rpt	4.34	0.15	55.9	73	88	95	95	96.1	96.5	0.08	0.30
JS4502	HG Comp.	15.7	0.05	91.5	98	99	99	100	99.7		0.05	0.20
JS4553	#14	0.92	0.03	65.4	72	76	84	90	96.8		0.06	0.20
JS4554	#15	3.64	0.05	70.8	85	93	96	97	98.6		0.05	0.20
JS4555	#16	1.33	0.03	72.9	87	95	96	97	97.7		0.07	0.30
JS4556	#17	0.61	0.01	67.4	77	91	95	97	98.4		0.06	0.25

Table 13-23: 2019 Testwork Variability Gravity/Leach Results with Select Repeat Tests

Test	Sample	Calc. Head	Residue	% GRG	Overall % Gold Extraction @ Time Hours						Reagent Cons. kg/t	
No.	ID	g/t Au	g/t Au		2	4	8	12	24	36	NaCN	Lime
JS4557	#18	1.78	0.07	46.4	60	76	86	91	96.1		0.06	0.25
JS4558	#19	1.97	0.02	84.7	88	90	93	95	99.0		0.07	0.20
JS4559	#20	6.86	0.11	70.7	91	95	98	98	98.4		0.08	0.20
JS4560	#21	2.49	0.08	72.6	81	86	92	94	96.8		0.07	0.30
JS4567	#29	0.89	0.03	45.5	57	68	90	94	96.6		0.05	0.20
JS4568	#32	7.90	0.18	72.6	86	92	96	97	97.7		0.05	0.20
JS4569	#33	1.88	0.07	46.2	73	85	92	95	96.3		0.05	0.20
JS4664	#34 Rpt	2.01	0.07	30.2	53	78	95	95	96.2	96.5	0.10	0.26
JS4571	#35	3.33	0.02	91.2	97	98	99	99	99.4		0.04	0.25
Fresh Median		2.08	0.06	66.4	79	87	93	95	96.8		0.06	0.25
Fresh Average		3.15	0.06	61.3	75	84	92	95	96.8		0.07	0.29
15th Percentile		0.91	0.02	41.4	56	76	88	91	95.2		0.05	0.20
85th Percentile		5.06	0.11	80.6	88	95	97	98	99.0		0.10	0.35
JS4587	#22 [Ox] Rpt	15.9	0.07	75.2	96	98	99	99	99.3	99.6	0.17	3.10
JS4562	#23 [Ox]	1.67	0.02	70.3	95	96	98	98	98.8		0.11	6.70
JS4563	#24 [Ox]	1.50	0.06	47.9	81	88	91	92	96.0		0.14	2.80
JS4564	#25 [Ox]	7.53	0.02	86.9	97	99	99	100	99.7		0.08	0.60
JS4565	#26 [Ox]	1.19	0.05	29.0	79	85	89	92	95.8		0.08	1.80
JS4566	#28 [Ox]	0.50	0.02	49.8	92	93	96	96	96.0		0.08	2.15
Oxide Median		1.59	0.04	60.1	93	95	97	97	97.4		0.10	2.48
Oxide Average		4.72	0.04	59.8	90	93	95	96	97.6		0.11	2.86
15th Percentile		1.02	0.02	43.1	81	87	90	92	96.0		0.08	1.50
85th Percentile		9.62	0.06	78.1	97	98	99	99	99.4		0.15	4.00
Average All Var		3.46	0.06	61.0	83	89	93	95	96.8		0.07	0.30
Median All Var		1.99	0.05	66.4	78	86	93	95	97.0		0.08	0.83
15th Percentile		0.90	0.02	41.0	58	77	89	92	95.6		0.05	0.20
85th Percentile		6.44	0.09	81.3	94	96	98	98	99.1		0.13	1.43

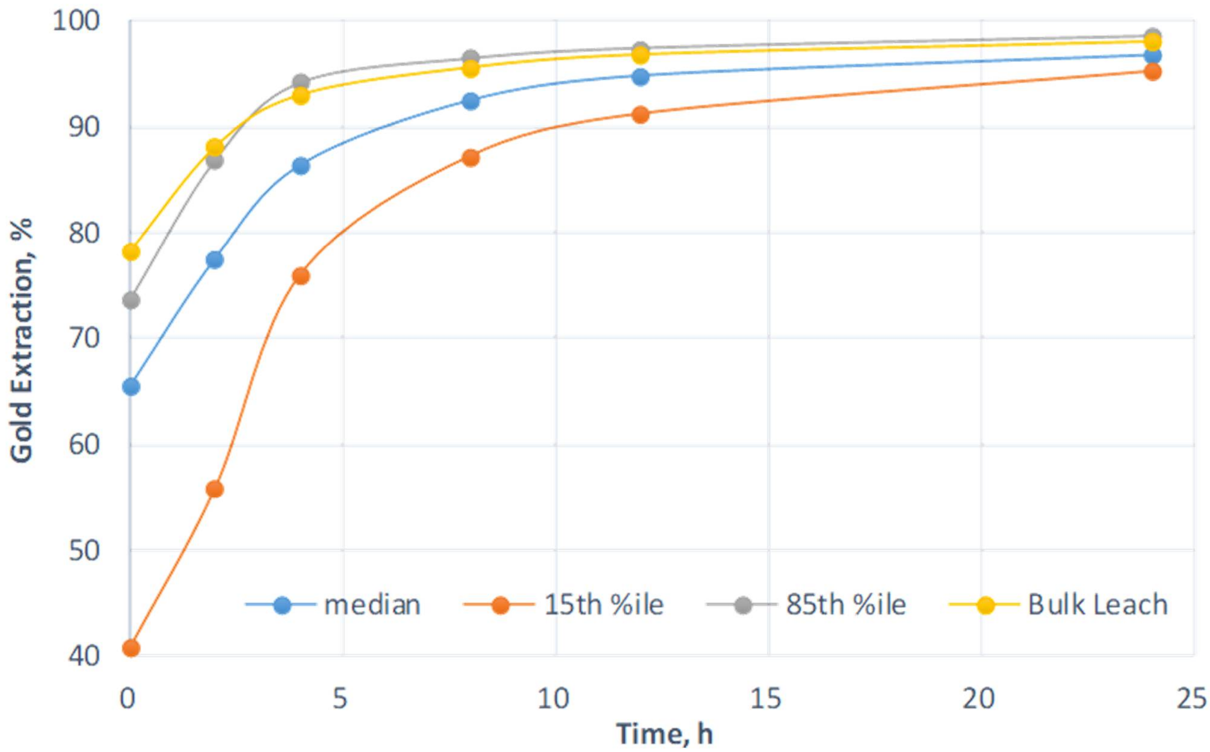


Figure 13-26: 2019 Testwork Gravity/Leach Variability Test Leach Profiles with Selected Repeated Tests

13.10 2021 Programme - Metallurgical Testing, Variability Samples

The 2021 resource extension programme followed a different approach, being focussed on confirming the suitability of the proposed processing route for the future ores. Forty variability metallurgical samples, comprising four groups of ten samples representing each of the major host lithologies: IFEL, IMAF, QTZ and VMAF (refer discussion in Section 13.4.1), and covering the geographical and depth ranges of the extended Lafigué deposit were submitted for variability testing.

The tests followed the established gravity separation and direct cyanidation flowsheet utilising the optimised conditions determined during the 2019 programme. The work focused on two grind sizes, P_{80} of 75 µm and 106 µm.

All metallurgical testwork was undertaken by ALS Metallurgy using Perth tap water.

13.10.1 Gravity/Leach Testing at Standard Conditions

Leach tests were conducted on the gravity tails following mercury amalgamation of the gravity concentrate from each head sample to recover the free gold. Cyanidation was carried out for 24 hours at 55% (w/w) solids with air sparging. NaCN concentration was initially adjusted to 0.025% (w/v) NaCN and allowed to decay to 0.02% (w/v) NaCN.

The variability test results are summarised in Table 13-24 and presented graphically in Figure 13-27.

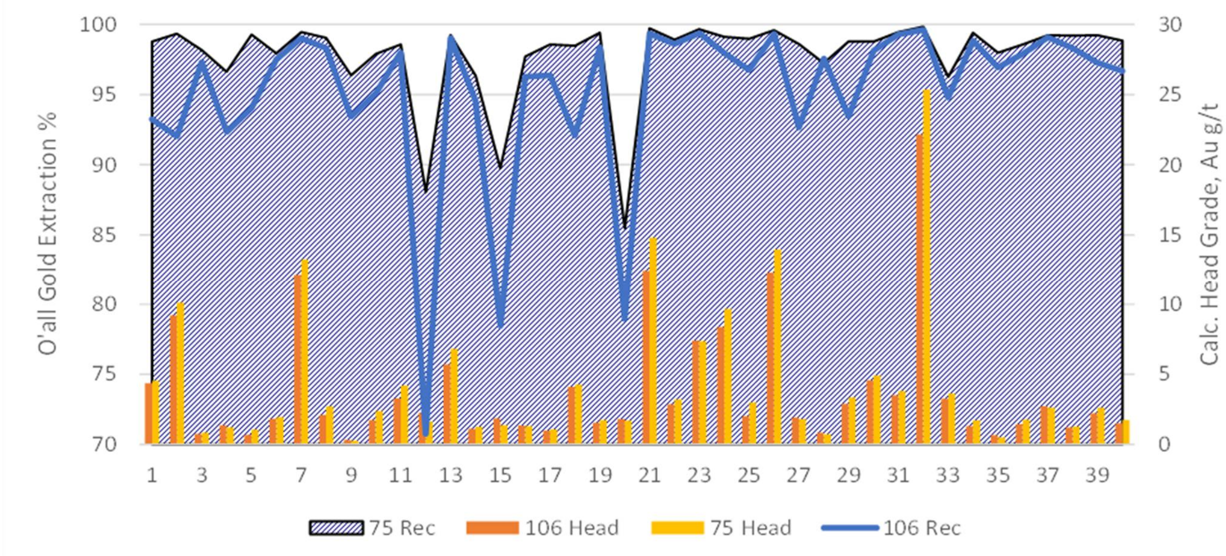


Figure 13-27: Overall Gold Extraction and Calc. Head for the 40 Variability Samples at P_{80} 75 and 106 µm

The overall gold extraction from the majority of the variability samples was consistently high. Considering the median extractions to minimise the impact of the outliers, the gold extraction was 98.8% for the 75 µm tests and 97.3% for the coarser grind size of 106 µm.

The three low recovery gold samples, MET 12, MET 15 and MET 20 were all from the IMAF hosted samples, but geological examination did not suggest any reasons for the notably poorer performance. Characteristics of the tests/samples that may be related (but were not unique to these samples) were:

- Relatively low gravity gold content.
- Relatively high iron and sulphide sulphur contents.
- A low dissolved oxygen concentration at the start of the test (before air sparging commenced).
- A lower pH at the end of the test and above average lime (and cyanide) consumption.

The leach kinetic curves for these three tests are all rising quite steeply at the termination of the test for both grind sizes (see the three stand-out lowest curves in Figure 13-28 and Figure 13-29). This suggests that the problem is more related to slow leaching (potential silver/telluride association, sulphur passivation, base metal competition for reagents, etc.) than poor gold liberation at the selected grind sizes. Repeat testing for these samples was planned with 36-hour residence time and 0.035% NaCN addition.

The variability in the sample head assays caused by the high gravity gold content and 'spotty' distribution results in an overall low bias when comparing the assay average with the calculated heads following gold extraction. This is to be expected with the probability of individual gold grains being missed when sub-sampling for assay, whereas all the gold in the larger leach feed sample is fully accounted for.

General agreement between the calculated heads for the 106 µm and 75 µm grinds appears to be moderately good, with some variability is evident and overall, the 106 µm heads are biased 18% low compared to the 75 µm samples (based on the median values). This is too large a bias to ascribe to sub-sampling differences alone and suggests there may be an assay bias as well. With the higher gravity recoverable gold (GRG) at the finer grind (11% greater) there is less leach feed gold. A low solution gold assay bias would then affect the coarser grind samples more. A slightly low solution assay bias is more likely than a bias in the gravimetric assays used for the GRG and residue samples. This bias would be within the nominated assay accuracy (typically ±(4 to 5)% for low grade gold samples). A small percentage increase in gold solution assays would tend not only to equilibrate the calculated heads, but also improve the relative extractions from the coarser grind.

The general trends in the gold extraction kinetics at the two selected grind sizes are presented in Figure 13-28 and Figure 13-29. At the P_{80} grind size of 75 µm, 37 out of 40 samples achieved an overall gold extraction above 96%. Gravity gold recovery (0 hours leach time), ranged from (60 to 90)% for the majority of the samples. Samples with lower gravity recoverable gold (GRG) tended to also have slower leach kinetics and in a few cases, lower overall gold extractions.

Similar characteristics were achieved for the coarser P_{80} grind of 106 µm, although gravity gold and leach extractions at 24 h were typically slightly lower than the 75 µm tests. 37 out of 40 samples achieved an overall gold extraction above 90% with the same three samples (MET 12, 15 and 20) having significantly lower gold extractions. The samples with slower leach kinetics were also more evident at the coarser grind size.

Sufficient numbers of finer ground samples showed higher GRG to demonstrate that gravity gold liberation was increased with finer grinding. Increased gravity recovery generally corresponded to better overall gold extraction results. The overall improvement in the median gold extraction with finer grinding for the samples is only 1.5%, but this would be sufficient to justify adopting the P_{80} grind of 75 µm based on the grind optimisation economics. This comparison does not take account of the observations from the 2019 testwork programme where additional liberation was largely discounted as being required for improved recovery, with similarly enhanced gold extractions at the coarser grind resulting from improved kinetics or extended leach residence times. Repeat testing of selected samples was proposed:

- Extend the leach durations to 36 hours. This should have been included from the outset to provide the additional data to assess if the finer grind should be targeted in place of, or in addition to the longer residence time.
- Increased cyanide addition was also tested for the slow leaching samples.

These solutions are more cost effective, flexible and energy efficient than finer grinding if similar gold extractions can be achieved.

Further learning from the previous programme that would have benefitted the comparative testing of the two grinds, would be the use of a common head sample ground to P80 of 106 µm for gravity concentration, with secondary grinding of half the tail to achieve the P80 of 75 µm. The differences in GRG and calculated heads would have been minimised so that the leach extraction and residue grades could be more meaningfully compared.

Table 13-24: Variability Gravity – Direct Tails Leach Test Results

Var. Com#	Test ID	P_{80}	Calc. Head	Tails	Gravity Gold	Gold - Leach Extraction at Hour Overall (% w/w)					Consumption (kg/t)	
		(µm)	(Au ppm)	(Au ppm)	(% w/w)	2	4	8	12	24	NaCN	Lime
MET 1	AM1124	106	4.37	0.30	68.9	79.5	86.8	91.5	92.3	93.3	0.04	0.29
MET 1	AM1125	75	4.55	0.06	78.1	88.8	94.5	97.5	98.0	98.8	0.06	0.28
MET 2	AM1126	106	9.21	0.74	71.2	78.4	83.6	88.6	89.8	92.0	0.05	0.37
MET 2	AM1127	75	10.14	0.07	68.5	77.1	83.0	91.0	94.5	99.4	0.04	0.50
MET 3	AM1128	106	0.75	0.02	34.1	60.9	81.0	92.1	93.1	97.3	0.02	0.27
MET 3	AM1129	75	0.82	0.02	67.5	85.9	93.6	96.8	98.2	98.2	0.04	0.30
MET 4	AM1130	106	1.37	0.11	34.7	41.5	47.0	62.8	80.9	92.3	0.04	0.21
MET 4	AM1131	75	1.19	0.04	45.0	58.0	72.9	93.3	95.8	96.6	0.05	0.25
MET 5	AM1132	106	0.67	0.04	66.4	81.1	87.5	90.4	90.9	94.0	0.04	0.28
MET 5	AM1133	75	1.04	0.01	86.3	94.1	97.1	98.6	98.9	99.3	0.05	0.29
MET 6	AM1134	106	1.82	0.05	79.3	83.8	86.0	90.1	92.5	97.5	0.06	0.39
MET 6	AM1135	75	1.94	0.04	82.2	89.1	91.4	93.9	95.8	97.9	0.07	0.37
MET 7	AM1136	106	12.14	0.12	55.3	62.7	69.5	83.5	94.9	99.0	0.06	0.43
MET 7	AM1137	75	13.24	0.07	78.1	84.2	88.1	94.7	98.6	99.5	0.06	0.30
MET 8	AM1138	106	2.09	0.04	58.8	77.0	86.3	94.3	95.7	98.3	0.06	0.33
MET 8	AM1139	75	2.68	0.03	80.0	93.9	96.6	98.3	98.7	99.1	0.06	0.38
MET 9	AM1140	106	0.30	0.02	47.6	59.7	64.9	74.9	79.7	93.4	0.08	0.27
MET 9	AM1141	75	0.28	0.01	61.6	72.0	77.7	87.4	91.3	96.4	0.05	0.29
MET 10	AM1142	106	1.74	0.09	67.5	76.2	81.0	87.1	91.5	95.1	0.05	0.36
MET 10	AM1143	75	2.38	0.05	72.0	79.4	83.2	87.3	91.7	97.9	0.08	0.28
MET 11	AM1144	106	3.27	0.07	72.7	86.7	91.3	95.0	96.4	98.0	0.04	0.51
MET 11	AM1145	75	4.21	0.06	77.7	86.2	90.5	94.9	96.6	98.6	0.10	0.63
MET 12	AM1146	106	2.24	0.66	17.2	30.7	36.2	48.2	56.0	70.7	0.13	0.60
MET 12	AM1147	75	1.63	0.20	30.7	47.0	54.0	64.6	71.6	88.0	0.17	0.62
MET 13	AM1148	106	5.73	0.06	73.9	89.0	93.1	96.3	97.5	99.0	0.07	0.37
MET 13	AM1149	75	6.82	0.05	74.0	87.8	92.0	96.2	97.1	99.3	0.10	0.28
MET 14	AM1150	106	1.11	0.06	39.2	54.0	73.5	87.8	91.1	94.6	0.10	0.45
MET 14	AM1151	75	1.24	0.05	59.0	67.6	72.7	84.3	89.9	96.4	0.08	0.53
MET 15	AM1152	106	1.86	0.40	28.8	38.1	42.5	47.8	56.0	78.5	0.13	0.57
MET 15	AM1153	75	1.36	0.14	44.3	57.2	62.1	71.6	78.2	89.7	0.15	0.60
MET 16	AM1154	106	1.34	0.05	50.6	74.4	86.1	92.3	93.7	96.3	0.05	0.51
MET 16	AM1155	75	1.31	0.03	59.0	79.9	88.3	92.6	94.0	97.7	0.08	0.62
MET 17	AM1156	106	0.96	0.04	44.2	66.7	81.4	90.1	92.7	96.4	0.07	0.42
MET 17	AM1157	75	1.06	0.02	65.3	81.1	89.9	94.9	96.3	98.6	0.08	0.47
MET 18	AM1158	106	4.11	0.33	71.6	81.3	86.1	89.5	90.3	92.1	0.05	0.34

Table 13-24: Variability Gravity – Direct Tails Leach Test Results

Var. Com#	Test ID	P_{80}	Calc. Head	Tails	Gravity Gold	Gold - Leach Extraction at Hour Overall (% w/w)					Consumption (kg/t)	
		(µm)	(Au ppm)	(Au ppm)	(% w/w)	2	4	8	12	24	NaCN	Lime
MET 18	AM1159	75	4.28	0.07	74.6	85.5	91.9	95.7	96.7	98.5	0.06	0.30
MET 19	AM1160	106	1.55	0.03	80.6	90.1	94.1	96.3	97.0	98.4	0.07	0.34
MET 19	AM1161	75	1.75	0.01	81.5	90.9	96.0	98.2	98.8	99.4	0.07	0.36
MET 20	AM1162	106	1.81	0.38	36.8	48.4	53.2	60.8	65.6	78.9	0.17	0.65
MET 20	AM1163	75	1.69	0.25	46.0	56.7	60.9	66.0	70.3	85.5	0.16	0.64
MET 21	AM1164	106	12.44	0.08	85.4	95.0	97.6	98.7	99.0	99.4	0.06	0.32
MET 21	AM1165	75	14.81	0.04	90.0	94.8	97.5	99.2	99.4	99.7	0.04	0.24
MET 22	AM1166	106	2.87	0.04	77.7	86.3	93.4	96.8	97.4	98.6	0.08	0.31
MET 22	AM1167	75	3.21	0.04	80.2	88.7	94.1	96.9	97.6	98.9	0.07	0.32
MET 23	AM1168	106	7.42	0.05	85.0	93.7	96.9	98.8	99.4	99.4	0.06	0.27
MET 23	AM1169	75	7.38	0.03	91.9	94.8	96.9	98.7	99.3	99.7	0.06	0.23
MET 24	AM1170	106	8.39	0.17	76.1	86.5	92.5	95.8	96.9	98.0	0.08	0.27
MET 24	AM1171	75	9.66	0.09	79.4	88.5	94.1	97.1	98.2	99.1	0.06	0.24
MET 25	AM1172	106	2.00	0.07	67.5	84.3	91.0	94.2	95.7	96.7	0.07	0.32
MET 25	AM1173	75	2.99	0.03	77.6	90.1	95.1	97.2	97.8	99.0	0.06	0.34
MET 26	AM1174	106	12.30	0.08	79.8	96.9	98.0	99.0	98.8	99.3	0.04	0.44
MET 26	AM1175	75	13.95	0.06	88.5	97.6	98.5	99.1	99.5	99.6	0.05	0.35
MET 27	AM1176	106	1.91	0.14	60.0	74.8	81.4	86.3	89.0	92.7	0.07	0.40
MET 27	AM1177	75	1.80	0.03	71.2	82.1	87.6	93.5	96.1	98.6	0.10	0.34
MET 28	AM1178	106	0.82	0.02	36.5	53.0	67.4	86.8	90.8	97.6	0.05	0.27
MET 28	AM1179	75	0.73	0.02	41.9	61.0	77.2	86.6	90.1	97.3	0.06	0.25
MET 29	AM1180	106	2.90	0.19	71.0	85.8	89.0	91.1	92.0	93.5	0.08	0.34
MET 29	AM1181	75	3.34	0.04	78.8	92.2	95.8	97.4	98.1	98.8	0.08	0.35
MET 30	AM1182	106	4.56	0.09	68.7	87.4	92.2	95.2	96.3	98.1	0.06	0.32
MET 30	AM1183	75	4.91	0.06	67.4	86.6	93.8	97.2	97.9	98.8	0.06	0.26
MET 31	AM1184	106	3.51	0.03	89.3	96.8	97.6	99.0	99.1	99.3	0.07	0.33
MET 31	AM1185	75	3.79	0.02	85.1	95.6	96.7	98.3	99.0	99.5	0.07	0.31
MET 32	AM1186	106	22.16	0.08	89.1	95.3	96.9	98.2	98.7	99.6	0.06	0.35
MET 32	AM1187	75	25.37	0.04	88.9	97.4	98.5	99.4	99.6	99.8	0.10	0.36
MET 33	AM1188	106	3.27	0.17	60.2	79.1	86.0	91.1	93.1	94.8	0.07	0.36
MET 33	AM1189	75	3.63	0.14	67.0	85.6	90.4	93.2	94.4	96.3	0.11	0.39
MET 34	AM1190	106	1.30	0.02	81.7	92.4	95.4	97.7	98.6	98.8	0.07	0.31
MET 34	AM1191	75	1.71	0.01	81.7	93.9	96.0	98.0	98.4	99.4	0.06	0.27
MET 35	AM1192	106	0.65	0.02	47.5	78.2	86.6	93.0	97.5	96.9	0.08	0.46
MET 35	AM1193	75	0.50	0.01	49.0	89.0	92.2	95.2	95.9	98.0	0.12	0.44

Table 13-24: Variability Gravity – Direct Tails Leach Test Results

Var. Com#	Test ID	P_{80}	Calc. Head	Tails	Gravity Gold	Gold - Leach Extraction at Hour Overall (% w/w)					Consumption (kg/t)	
		(µm)	(Au ppm)	(Au ppm)	(% w/w)	2	4	8	12	24	NaCN	Lime
MET 36	AM1194	106	1.44	0.03	65.7	74.0	81.9	92.5	95.0	97.9	0.08	0.40
MET 36	AM1195	75	1.77	0.03	69.4	84.2	88.9	93.4	95.2	98.6	0.13	0.39
MET 37	AM1196	106	2.73	0.03	83.5	90.2	93.0	96.5	97.8	99.1	0.07	0.32
MET 37	AM1197	75	2.61	0.02	84.1	93.2	96.1	98.0	98.6	99.2	0.10	0.41
MET 38	AM1198	106	1.19	0.02	72.0	88.5	92.5	95.9	96.9	98.3	0.05	0.24
MET 38	AM1199	75	1.27	0.01	78.6	94.8	96.6	97.5	98.1	99.2	0.08	0.28
MET 39	AM1200	106	2.21	0.06	77.6	88.7	91.5	93.9	95.2	97.3	0.08	0.32
MET 39	AM1201	75	2.59	0.02	80.4	93.3	96.2	98.0	98.6	99.2	0.07	0.36
MET 40	AM1202	106	1.51	0.05	57.8	82.8	89.1	93.9	94.6	96.7	0.07	0.26
MET 40	AM1203	75	1.73	0.02	65.2	90.7	95.5	97.0	97.8	98.8	0.07	0.31

Table 13-24 notes: Repeat test results are not included in this table and the tests were carried out at standard conditions, namely: P80= 75 µm and 106 µm, 24 h Air Sparged Leach @ 55% w/w solids, 0.025% w/v NaCN)

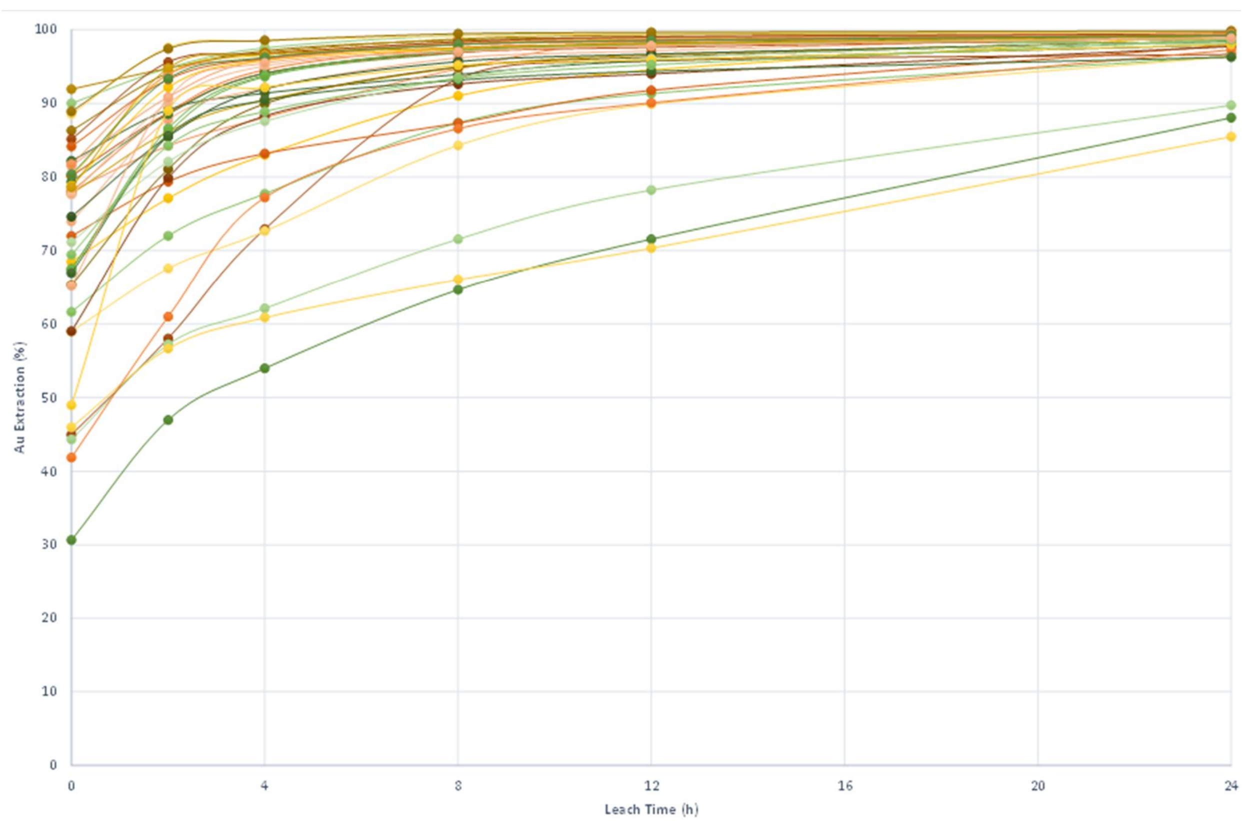


Figure 13-28: Overall Gold Extraction Kinetic Curves for the 75 µm Variability Samples

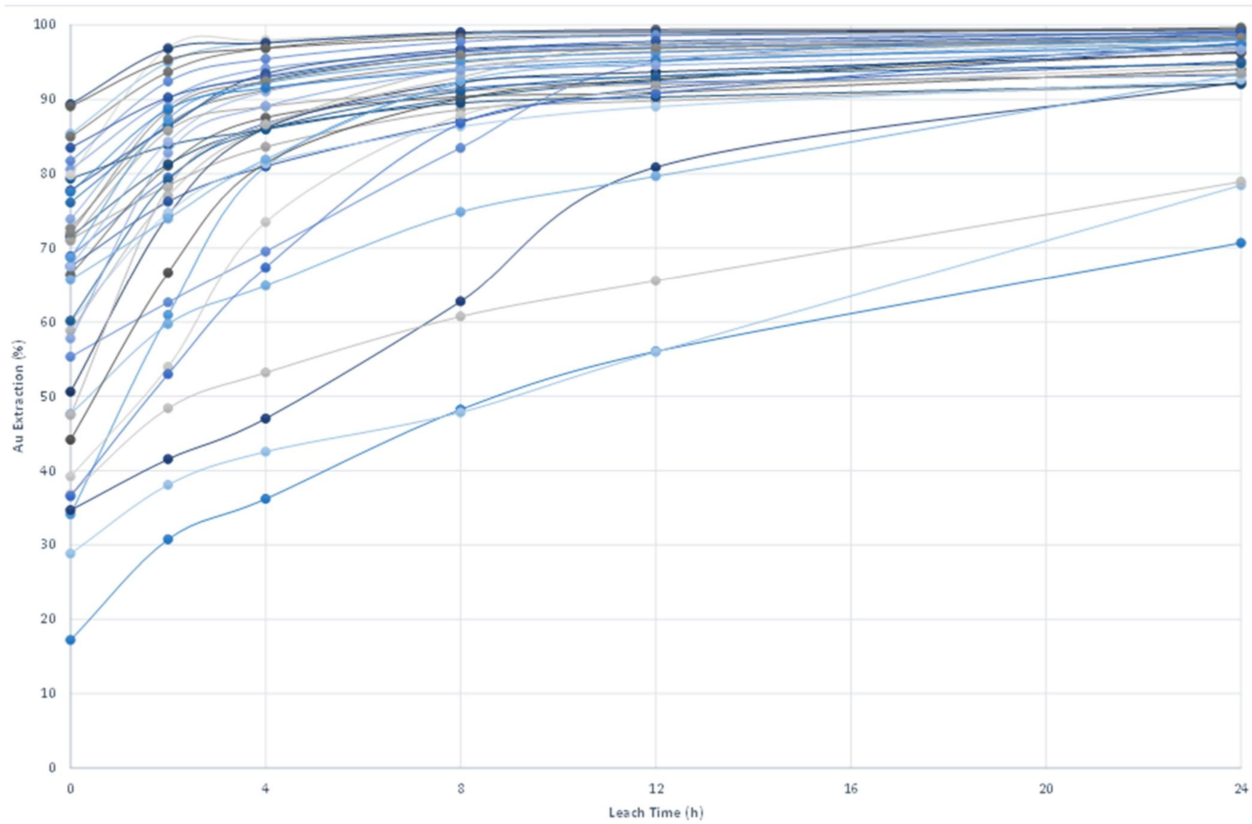


Figure 13-29: Overall Gold Extraction Kinetic Curves for the 106 µm Variability Samples

13.10.2 Repeat Variability Leach Testwork

The three low recovery gold samples, MET 12, MET 15 and MET 20, were retested with longer residence time (36-hour leach) and higher initial cyanide addition (0.035% w/v NaCN) found to be successful in the 2019 programme. A P_{80} grind of 75 µm was nominated by Endeavour which did not demonstrate the potential for improvement in the 106 µm results (MET 20 had limited sample availability and no further testing was possible). Although the finer grind improves kinetics, the objective was to achieve a similar outcome using a more energy efficient approach. It is unlikely that slower leaching ores will be readily identified using grade control assays since the occurrences noted appear to be quite discrete with few tracers to signal this changed behaviour. Adjustments to the plant processing baseline will need to be made retrospectively, so measures that can be flexibly applied will be required to ensure that acceptable recoveries for the majority of samples can be achieved at the nominated 106 µm grind.

Further analysis of the testwork indicated that the high gravity gold recoveries were in fact masking a number of other samples that had slower leaching characteristics that had not been immediately apparent. Isolating the leach data highlighted slow leaching examples with higher recovery potential, as well as some samples demonstrating high overall recoveries because of high gravity gold contents, but having poor leach characteristics. Additional samples were nominated for repeat testwork at a P_{80} of 106 µm to provide a weight of data to confirm that the finer grind is not required and to justify the investment in extra leach time. The additional sample mass available also provided further opportunity to better understand this behaviour and to assess if there are deleterious elements in common.

The further samples selected for testing were MET 1, 2, 4, 18, and 29. Samples 12 and 15 also had some remaining mass.

Table 13-25: Repeat Test Results for Samples MET 12, 15 and 20

Var. Comp.	Test ID	Grind P_{80}	Calc. Head	Gold Tail	GRG	Initial NaCN	Gold - Leach Extraction %		Consumption (kg/t)	
		(µm)	(Au ppm)	(Au ppm)	%	(w/v %)	24 h	36 h	NaCN	Lime
MET 12	AM1146	106	2.24	0.66	17.2	0.025	70.7	0.0	0.13	0.6
MET 12	AM1147	75	1.63	0.20	30.7	0.025	88.0	0.0	0.17	0.6
MET 12 Rpt	AM1232	75	2.09	0.24	17.5	0.035	85.5	88.8	0.23	0.8
MET 15	AM1152	106	1.86	0.40	28.8	0.025	78.5	0.0	0.13	0.6
MET 15	AM1153	75	1.36	0.14	44.3	0.025	89.7	0.0	0.15	0.6
MET 15 Rpt	AM1233	75	1.71	0.13	33.1	0.035	89.7	92.7	0.19	0.5
MET 20	AM1162	106	1.81	0.38	36.8	0.025	78.9	0.0	0.17	0.7
MET 20	AM1163	75	1.69	0.25	46.0	0.025	85.5	0.0	0.16	0.6
MET 20 Rpt	AM1234	75	1.93	0.10	40.9	0.035	92.9	94.8	0.29	0.7

13.10.2.1 Repeat Test Result Comparisons

Repeat testing followed the same gravity/leach flowsheet used for the variability testing with leaching at 55% solids sparged with air.

A summary of the results for the initial repeat tests is presented in Table 13-25 with comparative kinetic curves for the gold extraction vs the baseline tests presented in Figure 13-30.

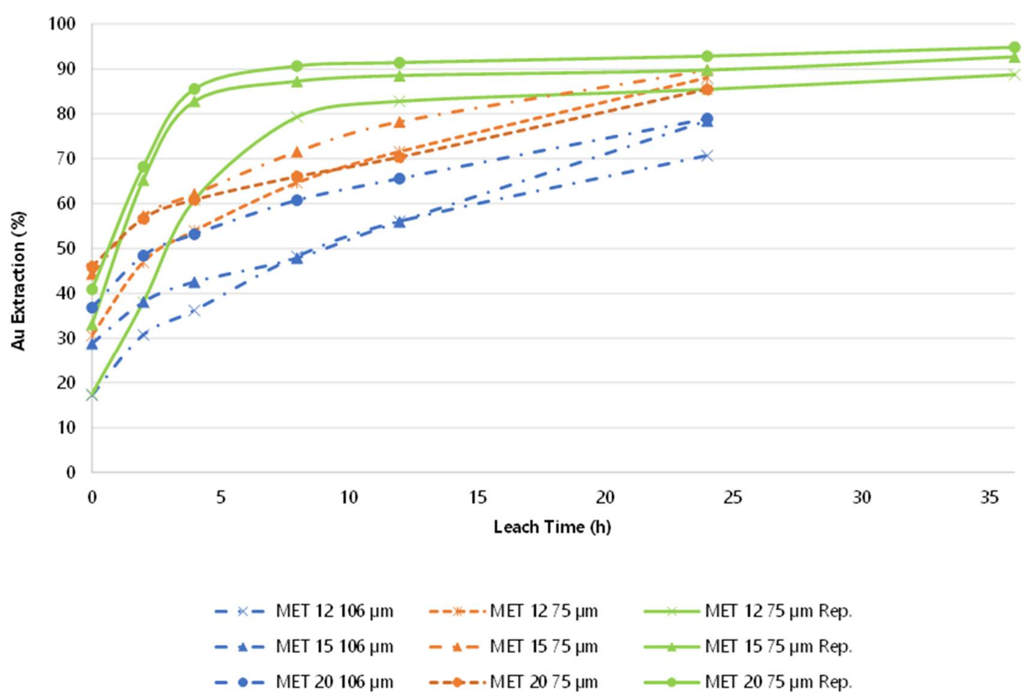


Figure 13-30: Comparison of Repeat vs. Initial Variability Leach Test Results for Samples MET 12, 15 and 20

Leach results for the original variability samples are shown as dashed lines and the repeat results are shown as solid lines.

The graph shows that repeat tests had faster leach kinetics, and that there was an improved overall gold extraction in each case. The residue grade for sample MET 12 repeat was higher than the original 75 µm test, but the calculated head was also higher. The repeat tests all benefitted from the extended leach time.

A summary of the results for the additional repeat tests at P_{80} 106 µm is presented in Table 13-26. Comparative kinetic curves for the gold extraction versus the baseline tests are presented in Figure 13-31 with relative gold extraction between tests shown in Figure 13-32.

Table 13-26: Repeat Test Results for Additional Samples at P_{80} 106 µm Grind

Var. Comp.	Test ID	Grind P_{80} (µm)	Initial NaCN (w/v %)	Calc. Head (Au g/t)	Gold Tail (Au g/t)	Gold - Leach Extraction %		Consumption (kg/t)	
						24 h	36 h	NaCN	Lime
MET 1	AM1124	106	0.025	4.37	0.30	93.26	-	0.04	0.29
	AM1262	106	0.035	3.85	0.15	94.06	96.10	0.07	0.26
MET 2	AM1126	106	0.025	9.21	0.74	92.02	-	0.05	0.37
	AM1263	106	0.035	8.22	0.03	98.22	99.63	0.06	0.46
MET 4	AM1130	106	0.025	1.37	0.11	92.33	-	0.04	0.21
	AM1264	106	0.035	1.27	0.04	94.69	96.84	0.11	0.24
MET 6	AM1134	106	0.025	1.82	0.05	97.53	-	0.06	0.39
	AM1265	106	0.035	0.97	0.01	96.57	98.96	0.07	0.32
MET 9	AM1140	106	0.025	0.30	0.02	93.43	-	0.08	0.27
	AM1266	106	0.035	0.39	0.01	93.02	98.70	0.08	0.27
MET 10	AM1142	106	0.025	1.74	0.09	95.11	-	0.05	0.36
	AM1267	106	0.035	2.09	0.05	96.06	97.61	0.15	0.21
MET 12	AM1146	106	0.025	2.24	0.66	70.72	-	0.13	0.60
	AM1268	106	0.035	2.05	0.14	88.35	93.41	0.19	0.77
MET 15	AM1152	106	0.025	1.86	0.40	78.45	-	0.13	0.57
	AM1269	106	0.035	1.85	0.07	92.66	96.22	0.20	0.61
MET 18	AM1158	106	0.025	4.11	0.33	92.08	-	0.05	0.34
	AM1270	106	0.035	4.42	0.18	94.78	96.04	0.10	0.28
MET 20	AM1163	75	0.025	1.69	0.25	85.47	-	0.16	0.64
	AM1234	75	0.035	1.93	0.10	92.87	94.82	0.29	0.71
MET 29	AM1180	106	0.025	2.90	0.19	93.45	-	0.08	0.34
	AM1271	106	0.035	3.12	0.08	96.25	97.60	0.08	0.23

Tails grades improved significantly in all cases which is most relevant since improved leaching was the focus of the repeat testwork. The significant gravity recovery can mask this improvement if only the overall extraction results are considered. Overall gold extraction improved in all cases, although a few tests (MET 6 and 9) had slightly lower 24-hour extractions than the originals, suggesting that slower kinetics remain a factor affecting gold extraction in individual samples. Cyanide consumption increased only marginally in most cases, and it appears that having the free cyanide available is more important than simply satisfying the consumption requirement.

Comparative leach kinetic curves for the repeat test results with the baseline variability tests are shown in Figure 13-31. The repeat test results are plotted as solid green lines with the original variability test results in dashed blue. Under the repeat test conditions, the leaching rate was faster in all cases with better characteristics for CIL operation.

Interestingly, under the same repeat test conditions, MET 12 and MET 15 samples showed greater improvement at 106 µm relative to the repeat 75 µm tests, with lower residues and higher overall extractions despite still having relatively slower kinetics. This observation shows that the localised variability within the samples remains high and comparative test results will not provide definitive answers aligned with expectations. General behavioural trends are the more important outcomes from this programme. The sample behaviour in the repeat tests vindicated the selection of the P_{80} 106 µm grind size (i.e., this is not a grind size/liberation issue) and confirmed that the extended leach time and higher cyanide addition were a preferred optimum baseline for further work.

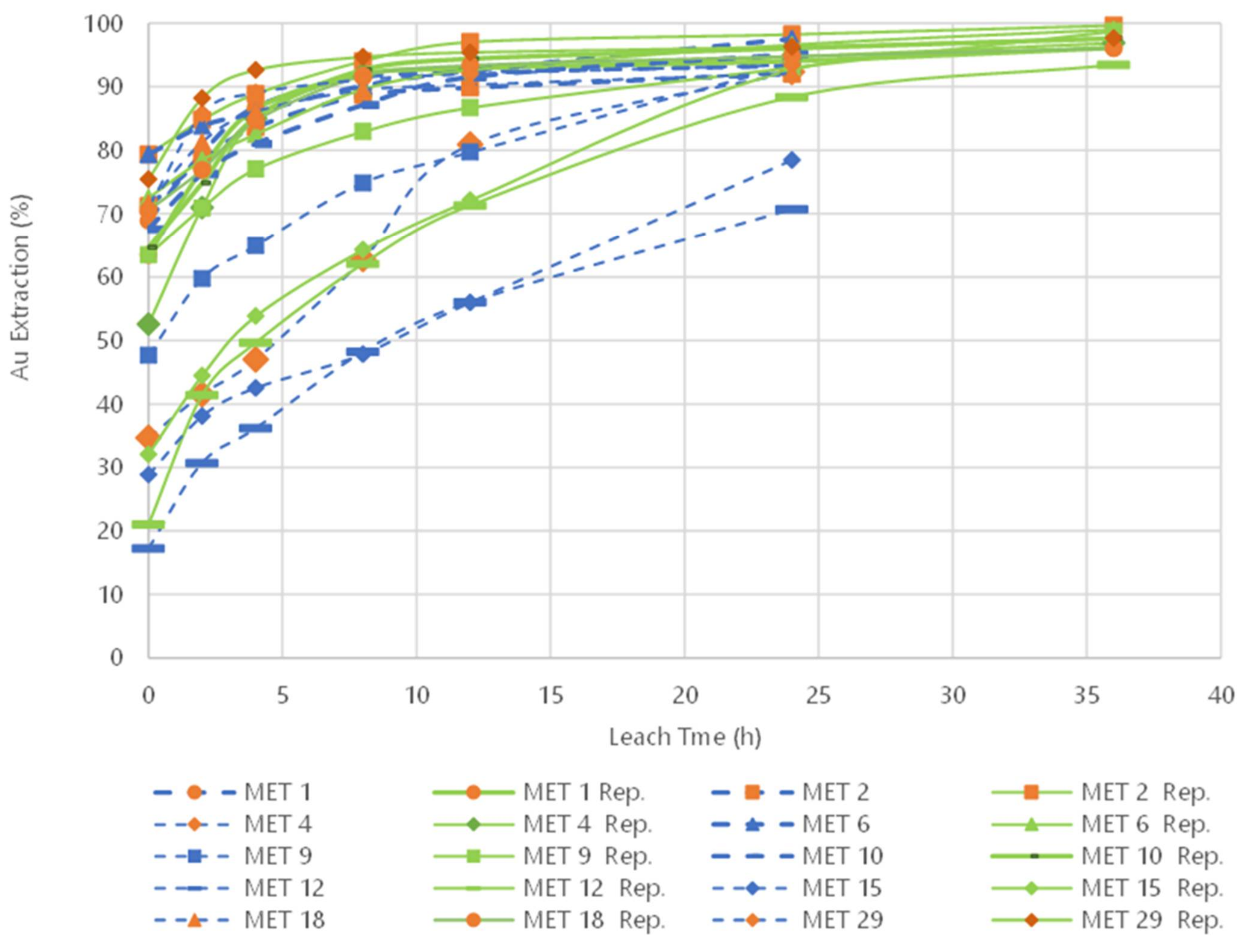


Figure 13-31: Comparison of Additional Repeat vs. Variability Leach Test Results at P_{80} 106 µm Grind

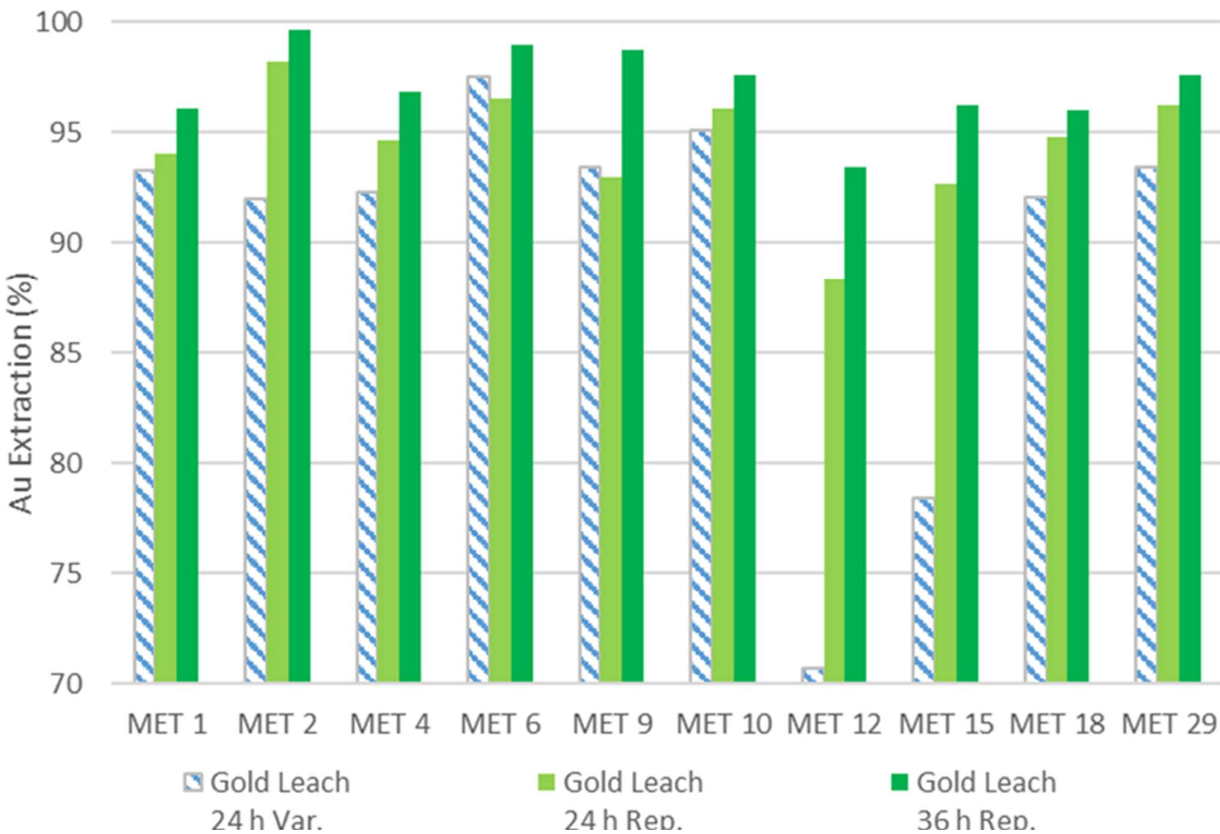


Figure 13-32: Comparison of Repeat vs. Variability Gold Extraction Results at P_{80} 106 µm

Figure 13-33 is presented to show the overall improvement in gold extractions compared to Figure 13-27. This data includes only the repeat test results from the selected samples for repeat testing as not all lower recovery samples had sufficient residual mass to allow retesting.

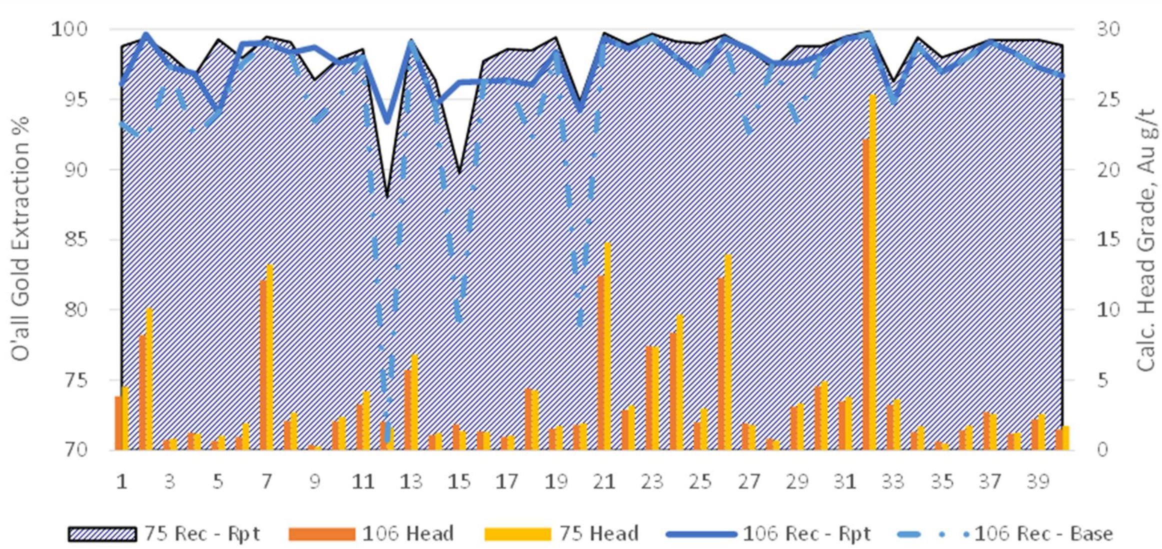


Figure 13-33: Overall Gold Extraction and Calc. Head for the 40 Variability Samples at P_{80} 75 and 106 µm with Repeat Tests

Table 13-27: 106 µm Variability Test Results Including Repeats

Sample ID	Test #	Grind Size (P80 um)	Residue Grade (Au g/t)	Gravity Gold (%)	Leach Ext'n (%)	Total Gold Ext'n (%)	Calc. Head (Au g/t)	NaCN Add'n (kg/t)	NaCN Cons. (kg/t)	Lime Add'n (kg/t)	Leach Ext'n (%) 0 h	Leach Ext'n (%) 2 h	Leach Ext'n (%) 4 h	Leach Ext'n (%) 8 h	Leach Ext'n (%) 12 h	Leach Ext'n (%) 24 h	Leach Ext'n (%) 36 h
MET 1	AM1262	106	0.15	63.44	86.15	96.10	3.85	0.43	0.07	0.26	63.44	76.95	86.48	91.99	92.94	94.03	96.10
MET 2	AM1263	106	0.03	79.35	94.68	99.63	8.22	0.41	0.06	0.46	79.35	84.77	88.85	94.02	97.03	98.22	99.63
MET 3	AM1128	106	0.02	34.12	95.94	97.32	0.75	0.21	0.02	0.27	34.12	60.95	80.99	92.15	93.12	97.32	
MET 4	AM1264	106	0.04	52.56	89.96	96.84	1.27	0.45	0.11	0.24	52.56	70.99	85.00	91.89	92.76	94.69	96.84
MET 5	AM1132	106	0.04	66.41	82.23	94.03	0.67	0.21	0.04	0.28	66.41	81.06	87.53	90.36	90.90	94.03	
MET 6	AM1265	106	0.01	72.50	92.79	98.96	0.97	0.43	0.07	0.32	72.50	78.43	82.51	89.58	92.59	96.57	98.96
MET 7	AM1136	106	0.12	55.32	97.79	99.01	12.14	0.23	0.06	0.43	55.32	62.66	69.54	83.47	94.93	99.01	
MET 8	AM1138	106	0.04	58.84	95.93	98.33	2.09	0.23	0.06	0.33	58.84	77.04	86.28	94.26	95.65	98.33	
MET 9	AM1266	106	0.01	63.47	93.26	98.70	0.39	0.43	0.08	0.27	63.47	70.89	77.02	82.91	86.69	93.02	98.70
MET 10	AM1267	106	0.05	64.64	89.82	97.61	2.09	0.47	0.15	0.21	64.64	74.83	85.21	92.65	94.39	96.06	97.61
MET 11	AM1144	106	0.07	72.65	92.73	98.01	3.27	0.21	0.04	0.51	72.65	86.67	91.25	94.96	96.41	98.01	

Table 13-27: 106 µm Variability Test Results Including Repeats

Sample ID	Test #	Grind Size (P80 um)	Residue Grade (Au g/t)	Gravity Gold (%)	Leach Ext'n (%)	Total Gold Ext'n (%)	Calc. Head (Au g/t)	NaCN Add'n (kg/t)	NaCN Cons. (kg/t)	Lime Add'n (kg/t)	Leach Ext'n (%) 0 h	Leach Ext'n (%) 2 h	Leach Ext'n (%) 4 h	Leach Ext'n (%) 8 h	Leach Ext'n (%) 12 h	Leach Ext'n (%) 24 h	Leach Ext'n (%) 36 h
MET 12	AM1268	106	0.14	21.00	88.28	93.41	2.05	0.51	0.19	0.77	21.00	41.38	49.65	62.06	71.30	88.35	93.41
MET 13	AM1148	106	0.06	73.88	96.33	99.04	5.73	0.23	0.07	0.37	73.88	89.01	93.14	96.31	97.46	99.04	
MET 14	AM1150	106	0.06	39.21	91.10	94.59	1.11	0.23	0.10	0.45	39.21	53.96	73.50	87.85	91.13	94.59	
MET 15	AM1269	106	0.07	32.11	90.86	96.22	1.85	0.51	0.20	0.61	32.11	44.48	53.83	64.27	72.13	92.66	96.22
MET 16	AM1154	106	0.05	50.62	92.46	96.28	1.34	0.21	0.05	0.51	50.62	74.38	86.11	92.32	93.68	96.28	
MET 17	AM1156	106	0.04	44.16	93.49	96.36	0.96	0.23	0.07	0.42	44.16	66.68	81.42	90.09	92.74	96.36	
MET 18	AM1270	106	0.18	70.52	83.38	96.04	4.42	0.45	0.10	0.28	70.52	76.91	84.68	91.71	93.36	94.78	96.04
MET 19	AM1160	106	0.03	80.58	91.70	98.39	1.55	0.23	0.07	0.34	80.58	90.07	94.13	96.33	97.04	98.39	
MET 21	AM1164	106	0.08	85.37	95.60	99.36	12.44	0.23	0.06	0.32	85.37	94.97	97.64	98.67	98.96	99.36	
MET 22	AM1166	106	0.04	77.75	93.73	98.61	2.87	0.23	0.08	0.31	77.75	86.30	93.45	96.75	97.39	98.61	
MET 23	AM1168	106	0.05	84.96	95.97	99.39	7.42	0.23	0.06	0.27	84.96	93.67	96.86	98.80	99.39	99.39	
MET 24	AM1170	106	0.17	76.14	91.50	97.97	8.39	0.25	0.08	0.27	76.14	86.49	92.50	95.85	96.89	97.97	

Table 13-27: 106 µm Variability Test Results Including Repeats

Sample ID	Test #	Grind Size (P80 um)	Residue Grade (Au g/t)	Gravity Gold (%)	Leach Ext'n (%)	Total Gold Ext'n (%)	Calc. Head (Au g/t)	NaCN Add'n (kg/t)	NaCN Cons. (kg/t)	Lime Add'n (kg/t)	Leach Ext'n (%) 0 h	Leach Ext'n (%) 2 h	Leach Ext'n (%) 4 h	Leach Ext'n (%) 8 h	Leach Ext'n (%) 12 h	Leach Ext'n (%) 24 h	Leach Ext'n (%) 36 h
MET 25	AM1172	106	0.07	67.54	89.98	96.75	2.00	0.21	0.07	0.32	67.54	84.32	91.02	94.24	95.70	96.75	
MET 26	AM1174	106	0.08	79.85	96.77	99.35	12.30	0.21	0.04	0.44	79.85	96.94	97.97	99.02	98.84	99.35	
MET 28	AM1178	106	0.02	36.54	96.16	97.56	0.82	0.21	0.05	0.27	36.54	52.98	67.38	86.77	90.76	97.56	
MET 29	AM1271	106	0.08	75.46	87.01	97.60	3.12	0.41	0.08	0.23	75.46	88.17	92.59	94.66	95.36	96.25	97.60
MET 30	AM1182	106	0.09	68.71	94.05	98.14	4.56	0.23	0.06	0.32	68.71	87.36	92.19	95.18	96.30	98.14	
MET 31	AM1184	106	0.03	89.35	93.32	99.29	3.51	0.23	0.07	0.33	89.35	96.80	97.58	98.99	99.09	99.29	
MET 32	AM1186	106	0.08	89.06	96.70	99.64	22.16	0.23	0.06	0.35	89.06	95.34	96.87	98.24	98.66	99.64	
MET 33	AM1188	106	0.17	60.17	86.93	94.79	3.27	0.23	0.07	0.36	60.17	79.09	85.97	91.08	93.08	94.79	
MET 34	AM1190	106	0.02	81.72	93.70	98.85	1.30	0.21	0.07	0.31	81.72	92.39	95.42	97.74	98.58	98.85	
MET 35	AM1192	106	0.02	47.48	94.17	96.94	0.65	0.21	0.08	0.46	47.48	78.18	86.62	93.01	97.47	96.94	
MET 36	AM1194	106	0.03	65.72	93.91	97.91	1.44	0.25	0.08	0.40	65.72	73.97	81.92	92.47	95.00	97.91	
MET 37	AM1196	106	0.03	83.48	94.45	99.08	2.73	0.23	0.07	0.32	83.48	90.24	92.99	96.47	97.80	99.08	

Table 13-27: 106 µm Variability Test Results Including Repeats

Sample ID	Test #	Grind Size (P80 um)	Residue Grade (Au g/t)	Gravity Gold (%)	Leach Ext'n (%)	Total Gold Ext'n (%)	Calc. Head (Au g/t)	NaCN Add'n (kg/t)	NaCN Cons. (kg/t)	Lime Add'n (kg/t)	Leach Ext'n (%) 0 h	Leach Ext'n (%) 2 h	Leach Ext'n (%) 4 h	Leach Ext'n (%) 8 h	Leach Ext'n (%) 12 h	Leach Ext'n (%) 24 h	Leach Ext'n (%) 36 h
MET 38	AM1198	106	0.02	72.04	94.01	98.32	1.19	0.21	0.05	0.24	72.04	88.49	92.45	95.95	96.86	98.32	
MET 39	AM1200	106	0.06	77.56	87.91	97.29	2.21	0.23	0.08	0.32	77.56	88.66	91.52	93.92	95.23	97.29	
MET 40	AM1202	106	0.05	57.80	92.13	96.68	1.51	0.23	0.07	0.26	57.80	82.80	89.08	93.87	94.59	96.68	
Median			**0.05**	**67.0**	**93.0**	**97.8**	**2.07**	**0.23**	**0.07**	**0.32**	**67.0**	**80.1**	**87.1**	**93.4**	**95.1**	**97.3**	**97.8***

Table 13-27 notes: Blue shading indicates repeat test result; Median result assumes 24 h extractions for the non-repeated tests, Median results are presented as they reduce the impact outliers as would be the case with a larger sample set or practical operating results.

13.10.3 Slower Leaching Composite Make-Up

Further variables in addition to cyanide concentration warranted investigation to better understand the causes of and provide mitigation measures for the slow leaching if it occurs. Insufficient sample remained to pursue further testing on individual slower leaching variability samples, so composites were made of samples showing similar characteristics. Two separate composites were felt to provide a better outcome than a single sample, as these would illustrate areas of commonality and variability.

It was assumed that pyrrhotite (Po) association was likely to be a dominant factor in this behaviour. Low initial dissolved oxygen levels, decreasing pH at the end of the tests and requiring higher free cyanide were all indicative of a reactive sulphide mineral being locally present. The two slower leaching composites made up were named the P and O composites.

The slow leaching samples (five variability samples per composite) were differentiated by oxygen uptake ahead of leaching. The P composite set of variability samples displayed low levels of dissolved oxygen ahead of cyanide leaching while the O composite variability samples set displayed normal dissolved oxygen levels. Once air sparging commenced, there was no differentiation as the oxygen demand was low in all cases and saturation dissolved oxygen levels were readily maintained. The P composite comprised equal masses of samples MET 6, 9, 10, 12, 15 while the O composite comprised equal masses of samples MET 1, 2, 4, 18, 29. The P composite source samples generally exhibited slower leach kinetics.

The composites were sub-sampled for mineralogical examination of the gravity concentrates with XRD on the tails. This investigation is described in detail in Section 13.6.4 but in summary the findings were:

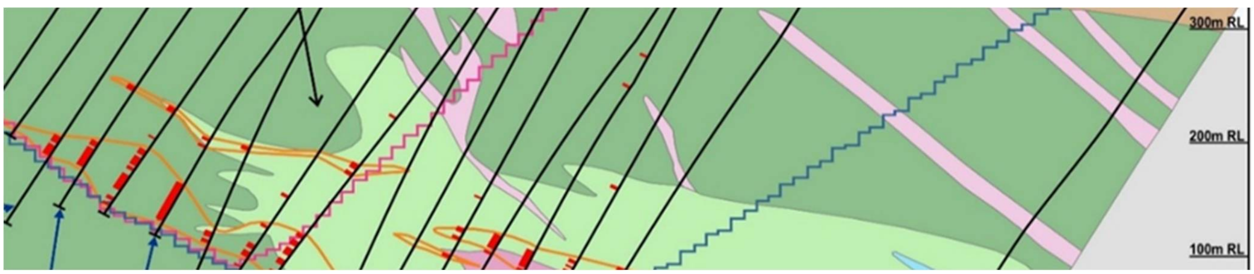
- Pyrrhotite is very much the dominant sulphide mineral in both composites comprising 5% of the gravity concentrate in O and 12% in P. This was similar to the mineralogy of the 2019 master composite, so the mineral associations were not unusual.
- Coarse gold grains were detected using optical microscopy, but gold grains were hard to find using QEMSCAN and the two detected were a liberated gold/silver grain and a much smaller gold/silver grain locked in pyrrhotite. Neither of these isolated occurrences could be considered to be diagnostic of the mineralisation style.
- This investigation did not provide additional information to demonstrate why these samples have slower leach kinetics.

Although comparison of the mineralogical differences between the O and P composites (Section 13.6.4) does not provide evidence to support a slower leaching gold fraction, it does highlight that there are notable differences in the background mineralogy. Composite P contains fewer alteration products and significant ilmenite as well as magnetite/hematite that is barely evident in the O composite.

It is also noteworthy that the samples that stood out as being 'low recovery' in the original testing, #12, 15 and 20 (composite P candidates) were distinctly shallower than the balance of the samples (between 115 and 145 m compared with an average down hole depth of 280 m for the balance of the 2021 samples). Samples #12 and 15 may have contributed the different mineralisation in the P composite that is not evident in the deeper O composite.

Shallower depths were also generally associated with the lower recovery/slower leaching samples in the 2019 programme, although with sampling being at the upper end of the footwall, the differences are not as apparent. It is possible that the mineralisation differences away from the footwall result in more reactive pyrrhotite or that the shallower samples display mildly transitional weathering characteristics with possible surface oxidation of the pyrrhotite making this more reactive. This observation is of value as it suggests indicators for slow leaching mineralisation that may warrant more attention during processing.

The excerpt below from Figure 13-2 illustrates the occurrence of the shallower ore zones away from the main footwall mineralisation. The darker green is the IMAF lithology with the light green being the VMAF.



Further review of the leach characteristics suggests that the O samples were differentiated by higher gold tails grades (106 µm tests) but approached final gold extraction levels (typically >80%) in 8 hours, so did not present a slow leaching as much as a lower recovery problem. Repeat testing with increased cyanide addition reduced the tails grades for most of the O constituents to levels similar to those for the 75 µm tests. This appeared to be an improvement in kinetics since the whole extraction curve moved upwards. Samples 1 and 18 may benefit marginally from finer grinding, but these rare instances would be difficult to identify in practice.

Comp. P samples demonstrated distinctly slow leaching behaviour however, often achieving low final residues, but having less than 60% of final gold extraction after 8 hours. Repeat testing with increased cyanide addition significantly improved leach kinetics (pre-8-hour extraction) and further reduced residue grades.

Bottle roll checks should be conducted on the grade control samples with IMAF/VMAF mineralisation away from the footwall. If slow leaching is evident, sulphide sulphur assays should be requested and repeat bottle rolls with adjusted conditions as per the optimisation testing reported here.

13.10.4 Leach Optimisation Testwork

Leach optimisation tests were performed on the P and O master composite samples with the objective of further improving the slow leach kinetics observed during the testing of the variability samples. Pre-oxidation and lead nitrate addition had shown some benefits when investigating the lower recovery variability samples in the 2019 testwork. A test series aimed at further investigating these options was proposed.

The tests on the larger composite masses (5 kg) allowed a bulk gravity concentration stage with intensive cyanidation of the concentrate. The gravity tails leach tests for the P and O composite samples were conducted under the following constant conditions:

- a grind size P_{80} of 106 µm
- slurry at 55% solids (w/w) with air sparging
- an initial cyanide concentration of 0.035% (w/v) maintained at 0.030% (w/v) at sampling intervals.

Tests investigated the effect of a four-hour slurry pre-oxidation stage with lime and the addition of 100 g/t of lead nitrate independently and sequentially. A baseline result was established for each composite to highlight the benefits of the alternate processes. A summary of the test results with overall gold extractions is presented in Table 13-28.

Table 13-28: Leach Optimisation Testwork - P & O Composites

Test ID	4 h Pre-Ox.	$Pb(NO_3)_2$ Add'n	Calc. Head Au (ppm)	Tail Au (ppm)	GRG (%)	Gold - Extraction %						Consumption (kg/t)	
						2 h	4 h	8 h	12 h	24 h	36 h	NaCN	Lime
P Composite													
AM1281	No	No	1.71	0.06	65.4	75.9	85.4	91.8	93.7	96.1	96.5	0.07	0.30
AM1282	No	Yes	1.70	0.05	66.2	86.2	92.9	95.2	95.8	97.1	97.1	0.09	0.30
AM1283	Yes	No	1.71	0.05	65.5	76.0	85.0	92.5	95.1	96.7	97.1	0.07	0.43
AM1284	Yes	Yes	1.70	0.05	66.0	86.9	93.4	95.8	96.2	97.1	97.1	0.07	0.40
O Composite													
AM1289	No	No	4.38	0.07	85.7	90.4	93.8	96.2	97.1	98.0	98.5	0.18	0.40
AM1290	No	Yes	4.35	0.06	86.2	94.0	96.5	97.5	98.0	98.3	98.7	0.05	0.32
AM1291	Yes	No	4.31	0.04	87.1	92.5	94.9	97.7	98.3	98.9	99.2	0.08	0.40
AM1292	Yes	Yes	4.34	0.03	86.4	94.2	96.2	98.2	98.6	99.0	99.3	0.07	0.40

The P and O composite leach results were more consistent and comparable than previous repeat testing of the variability samples, with similar tails grades and calculated heads being recorded. This was most likely as a result of having the common gravity removal stage for each composite ahead of the leach stages. The intensive cyanidation of the gravity concentrate would also oxidise any coarser reactive sulphides reporting to the concentrate, improving consistency in the comparable leach tests, whereas this would not occur with the gravity concentrate amalgamation approach.

The gravity tails leach optimisation results are illustrated in Figure 13-34 and Figure 13-35 for comparative purposes.

13.10.4.1 P Composite

The baseline overall gold extraction, test AM1281 for the P composite, was high at 96.5% with moderately fast leach kinetics. The pre-oxidation stage in test AM1283 showed little improvement in leach kinetics, however the overall gold extraction increased slightly to 97.1%. The addition of lead nitrate, test AM1282, resulted in notably faster leach kinetics, achieving an overall gold extraction of 92.9% within the first four hours. The overall gold recovery was similar to the pre-oxidation test. A combination of both pre-oxidation and lead nitrate, test AM1284, had a similar leach kinetics to the lead nitrate only with little further improvement in overall gold extraction.

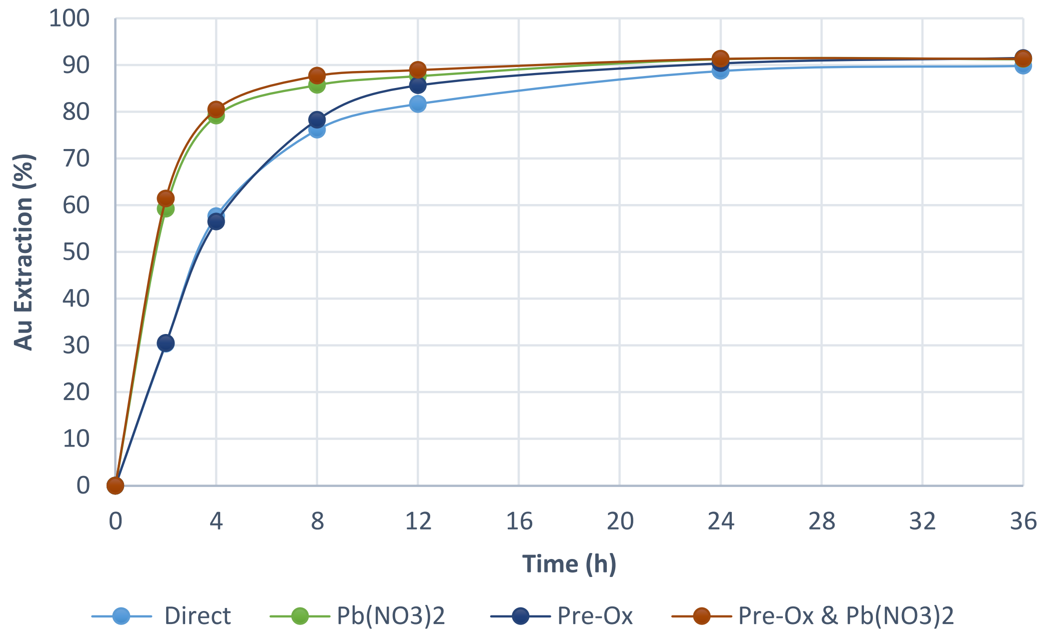


Figure 13-34: P Composite Leach Only Kinetics - Optimisation Tests

13.10.4.2 O Composite

The baseline overall gold extraction, test AM1289, for the O composite, was higher than that for the P composite at 98.5%, but the grade and GRG content were higher such that leach residue grades were similar for both cases. The O composite leach tests showed similar trends to the P composite apart from a greater recovery benefit from the pre-oxidation. This was unexpected since the O composite was made up from the samples that showed lower initial oxygen demand on slurring for the variability leach testing.

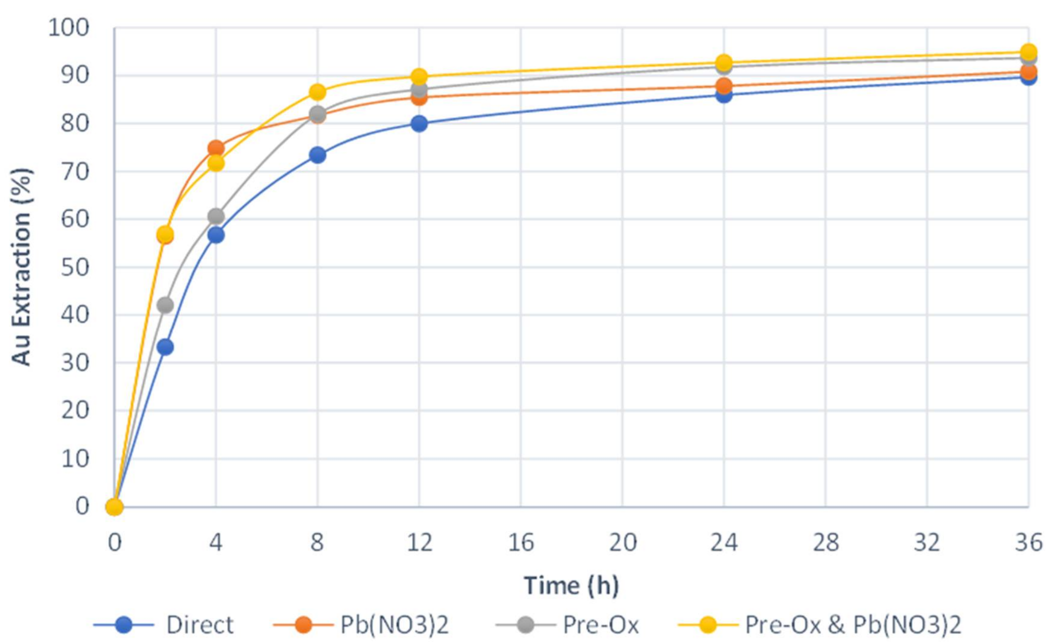


Figure 13-35: O Composite Leach Only Kinetics – Optimisation Tests

It was decided to make provision for lead nitrate addition to the leach should grade control indicate pockets of slower leaching ore, but pre-oxidation was not felt to be justified with the ores that would potentially benefit likely to be rare.

13.11 2021 Master Composite Testing

13.11.1 2021 Master Composite Make-up and Leach Feed Sample Preparation

A master composite was made up from equal contributions from each of the 40 variability samples excluding the higher sulphide samples MET 12, 15, 20 and 27 (based on Endeavour advice). This was termed the 'Average Grade Master Composite' and served to provide the bulk leach sample for further carbon kinetic testwork, physical characterisation of the ore and slurry samples, rheology testwork, cyanide detoxification testwork, thickening testwork and tails settling testwork.

13.11.1.1 Bulk Leach Testwork

A bulk gravity concentration test was conducted followed by mass reduction and intensive cyanidation of the concentrate to determine an indicative average gravity recoverable gold estimate for the project. Gravity recoverable gold (GRG) determination involved the use of a laboratory Knelson centrifugal concentrator. Sub samples were split from the master composite sample and processed through the unit in 1 kg portions. The concentrate mass was reduced to less than 0.5% of the feed mass to better align with actual plant practice. The gravity concentrate underwent intensive cyanidation, and the gravity gold leach solution was assayed to determine gravity gold recovery.

The bulk gravity tailings were thoroughly homogenised and split into sub-samples again for the downstream testwork programme (reported in Section 13.12). The gold recovery from the gravity stage was incorporated into the metallurgical balances for the subsequent gravity tails leach test series. The 2021 master composite sample bulk leach result (24 h residence time with 0.025% starting NaCN) is presented in Table 13-31. The bulk leach result from the 2019 composite is included in the summary table below for comparison showing a high degree of consistency across the orebody and with depth. A whole of ore leach test was conducted as before to demonstrate the gravity recovery advantages.

Table 13-29: Summary of 2021 Master Composite Bulk Gravity and Leach Test Results

Composite Sample ID	Test No.	Calc'd Head Au (g/t)	Residue Assay Au (g/t)	GRG Au Extr'n (% w/w)	Overall Au Extr'n (% w/w)
Average Grade Master Composite	AM1205	2.59	0.06	69.69	97.88
Whole of Ore Leach	AM1275	5.40	1.23	-	77.22
2019 Master Composite – Bulk Leach	JS4501	3.00	0.06	78.31	98.00

13.11.2 2021 Master Composite Leach Optimisation Testing

Separate testing was conducted on the higher sulphide variability samples (MET 12, 15, 20 and 27) to assess the options for improving leach kinetics and gold recoveries. It was unfortunate that sub-samples of this mineralisation were not included in the master composite to improve overall representivity and assess the impact of the slower leaching components in a larger sample where the deleterious constituents would be diluted.

However, when modelling the CIL using the above bulk leach result, it was noted that although the master composite had demonstrated high gold recoveries, the leach kinetics were notably slow and impacted carbon profiles and concentrations required. Since sufficient mass remained in the available sample reserves, confirmatory testing on the composite to demonstrate the efficacy of increased cyanide and lead nitrate addition in improving leach kinetics was proposed.

The following tests were conducted on the master composite sub-samples. Additional master composite sub-samples were made up from the original sample reserves. In addition to the bulk leach tests described above, two optimisation test series were conducted. In each case, a bulk gravity test was conducted on the combined master composite sub-samples at the nominated P_{80} grind size of 106 µm using the laboratory Knelson concentrator. The Knelson concentrate was subjected to intensive cyanidation and the intensive leach residue recombined with the bulk gravity tailings.

Baseline leach conditions from the variability testing were applied with any variations from these test conditions (below) noted for each case:

- A cyanide concentration of 0.025% w/v NaCN (allowed to decay to 0.02%).
- pH maintained at 10.5 with lime.
- A leach feed slurry density of 55% solids.
- Air sparged leaching with a residence time of 24 hours.

13.11.2.1 Leach Test Series

(Case numbering relates to the test numbers in Table 13-31):

1. Baseline bulk leach (15 kg) test, 24 h residence time to create products for further downstream physical characterisation and determination of specific process parameters.
2. Comparative whole of ore 24 h leach test (no gravity).
3. Following the above testing, all tests were extended to 36 h residence time to quantify the additional gold extractions from the slower leaching examples. Cyanide starting concentration was also increased to 0.035% w/v, maintained above 0.025% w/v. Gravity recovery from a 5 kg sub-sample from the master composite reserves produced the gravity tails for leach tests 3 to 7. Test 3 included addition of 100 g/t lead nitrate to the feed.
4. Addition of 200 g/t lead nitrate to the feed.
5. 4 h pre-oxygenation @ pH 11.0.
6. 4 h pre-oxygenation @ pH 11.0 and addition of 100g/t lead nitrate to the feed.
7. 4 h pre-oxygenation @ pH 12.0.
8. Gravity recovery from a 6 kg sub-sample produced the gravity tails for leach tests 8 to 13. For tests 10 to 13 the gravity tails were reground to a P_{80} of 90 µm to demonstrate the benefit of the finer grind achievable with the selected mill size. Test 8 aimed to establish a baseline result for 0.035% w/v starting NaCN for improved comparison between subsequent tests given sub-sample variability observed.
9. Addition of 100 g/t lead nitrate to the feed.
10. Test 10 revisited the initial baseline with 0.025% w/v starting NaCN for the finer grind.
11. As per test 10 with addition of 100 g/t lead nitrate to the feed.

12. Comparable test to 10 with 0.035% w/v starting NaCN for the finer grind.
13. As per test 12 with addition of 100 g/t lead nitrate to the feed.

In addition to the above test series results, the average results for the 75 µm P_{80} variability tests have been presented to confirm that the enhanced leaching conditions result in similar or better performance for the coarser 106 µm P_{80} grind size.

A number of the above tests appear to be redundant duplication, but it was found that, even when testing this supposedly homogenous composite, considerable variability in the gravity recovery and calculated head between samples was evident with the gold 'nugget' effect and localised mineralisation differences.

13.11.3 2021 Master Composite Leach Testwork Results

Comparative leach kinetic data was extracted from the testwork results, rather than overall gold extraction results, as these are typically dominated by the high gravity gold recovery fraction and the subtle differences in leach performance can be overlooked.

The gravity recoveries and calculated heads indicate the major differences between the sub-samples for the various test series. The early test 1 results (numbered test references above) based on the largest sub-sample appear to be fairly average relative to the variability sample set, with a head of 2.59 g/t Au and a gravity gold content of 70%. In comparison, the whole of ore leach sample (test 2) would have had significant gravity gold with a head of 5.40 g/t, which unsurprisingly resulted in slow leaching and lower gold extraction with no gravity stage.

Tests 3 to 7 had a low head grade of 1.32 g/t (common sub-sample) and reduced gravity recovery of 46 to 47%. Tests 8 to 13 countered this with a very high gravity recovery averaging 80%, from a head of 4.10 g/t Au. Testwork procedures were consistent and considered reliable in all cases.

Most importantly, the overall gold extractions are high in all cases and the residue grades vary between 0.03 and 0.05 g/t Au, indicating very robust process performance and confirming that this work is somewhat academic with little overall impact from the sample variability or modified leach conditions.

The results are best compared graphically. Figure 13-36 shows the leach profiles from tests 1 and 3 with the 75 µm P_{80} variability test average results for reference.

Figure 13-37 shows the clear kinetic enhancement of the tests with the addition of 100 g/t of lead nitrate. However, increasing the lead nitrate addition to 200 g/t had only a marginal benefit. Pre-oxidation in conjunction with lead nitrate may enhance overall recovery slightly, but on its own does not improve kinetics sufficiently to justify the additional capital and operating expense. Operation at pH 12 is detrimental to leaching.

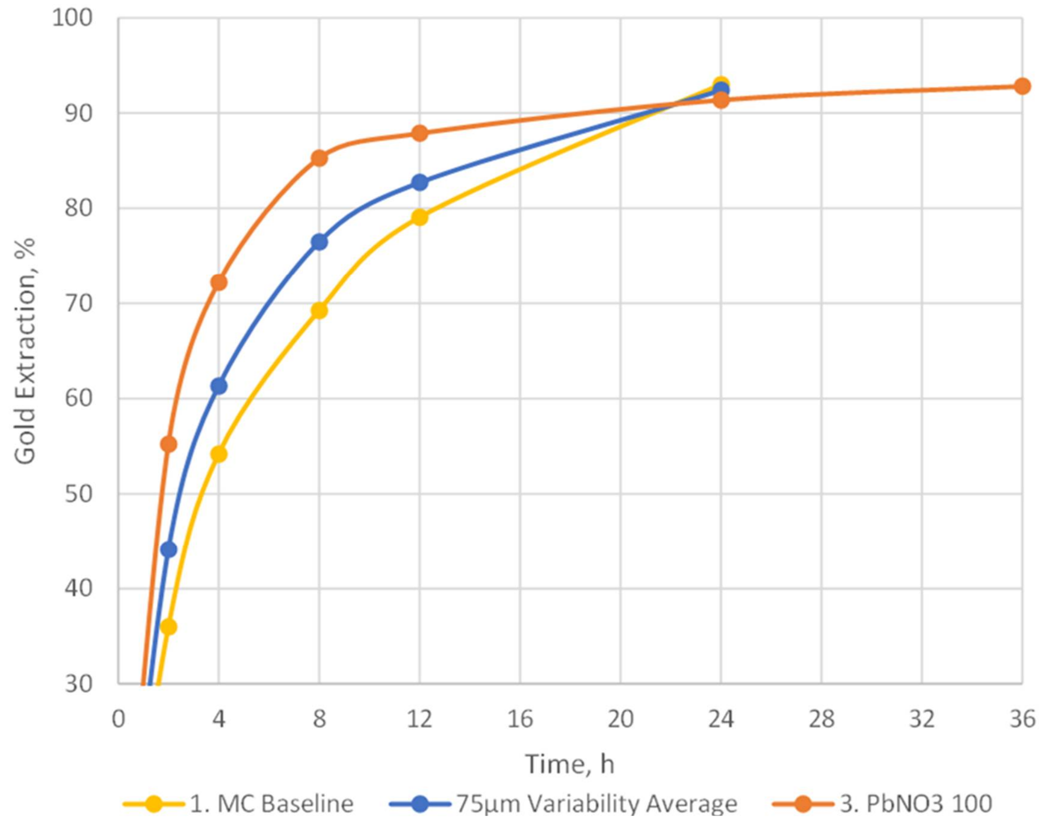


Figure 13-36: Baseline Leach Test Comparisons

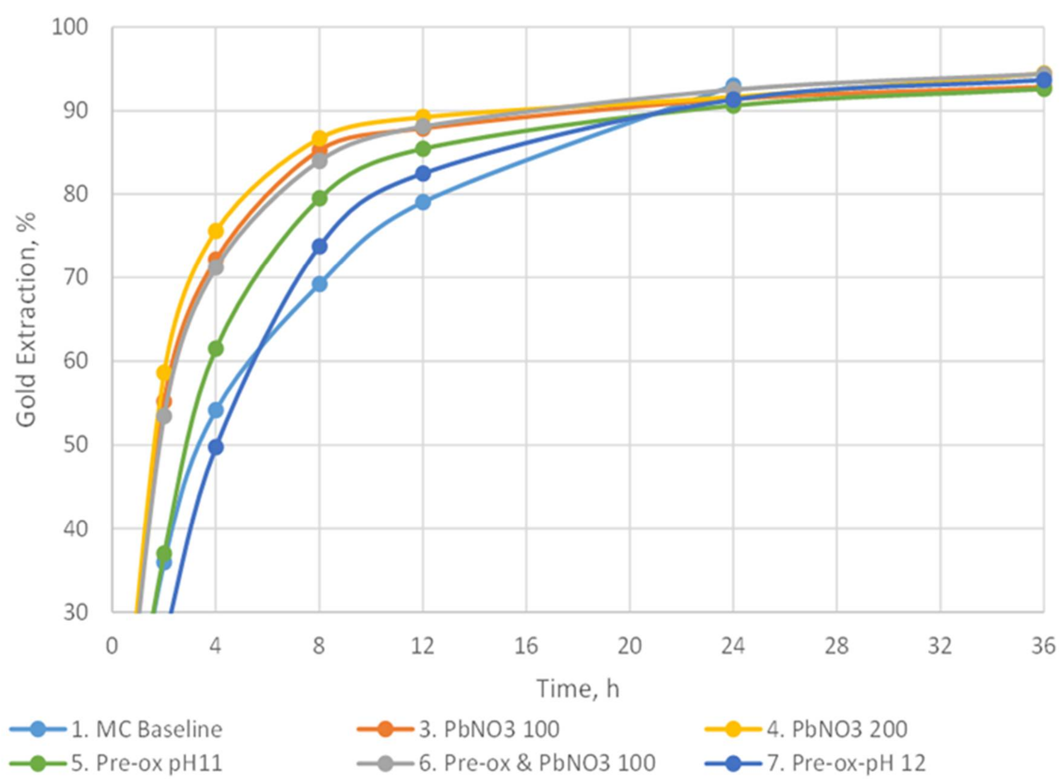


Figure 13-37: Lead Nitrate and Pre-Ox Comparison (Tests 3 to-7)

Figure 13-38 shows the leach results from tests 8 to 13. The trends are less discernible here with the high gravity recovery having left only a small fraction of gold for leaching. Small differences in gold size distribution, degree of liberation and possibly variable pyrrhotite association may have had a bearing on these outcomes.

The P80 of 106 µm comparative baseline test results appear to be reversed, with the non-lead nitrate test (8) having better kinetics than the test with lead nitrate added (9). A possible explanation is that Test 9 had a higher grade and gravity recovery, suggesting possibly more sulphide/pyrrhotite association. The Test 8 sample contrastingly showed no evidence of elements that negatively impacted the leach performance, and the extra cyanide addition was sufficient to enhance the kinetics. Test 9 kinetics are slower to 12 h, but with the sulphide effectively oxidised, the leach kinetics improve markedly and a higher overall extraction than test 8 is achieved.

The 90 µm P80 results show good kinetic enhancement with the lead nitrate addition, demonstrating that the increased cyanide level is less important at the finer grind if lead nitrate is present. The tests without lead nitrate (10 and 12) must have had slow leaching component associations, as they performed very poorly compared to the 106 µm P80 test (8). As with test 9, the higher cyanide in test 12 accelerated the kinetics after 12 h and improved the extraction compared to test 10.

Result comparisons were not as conclusive as hoped, but the main elements show sufficient consistency to indicate that higher cyanide and lead nitrate are beneficial for slower leaching ores. Slower leaching character is unlikely to be determined during grade control as it appears to occur on a very small scale. In all likelihood, the effects observed will be diluted out of mass ore treatment, but alert operators monitoring leach progress should develop early warnings that trigger increased downstream cyanide additions, should critical extraction hurdles be missed by tank 3 or 4.

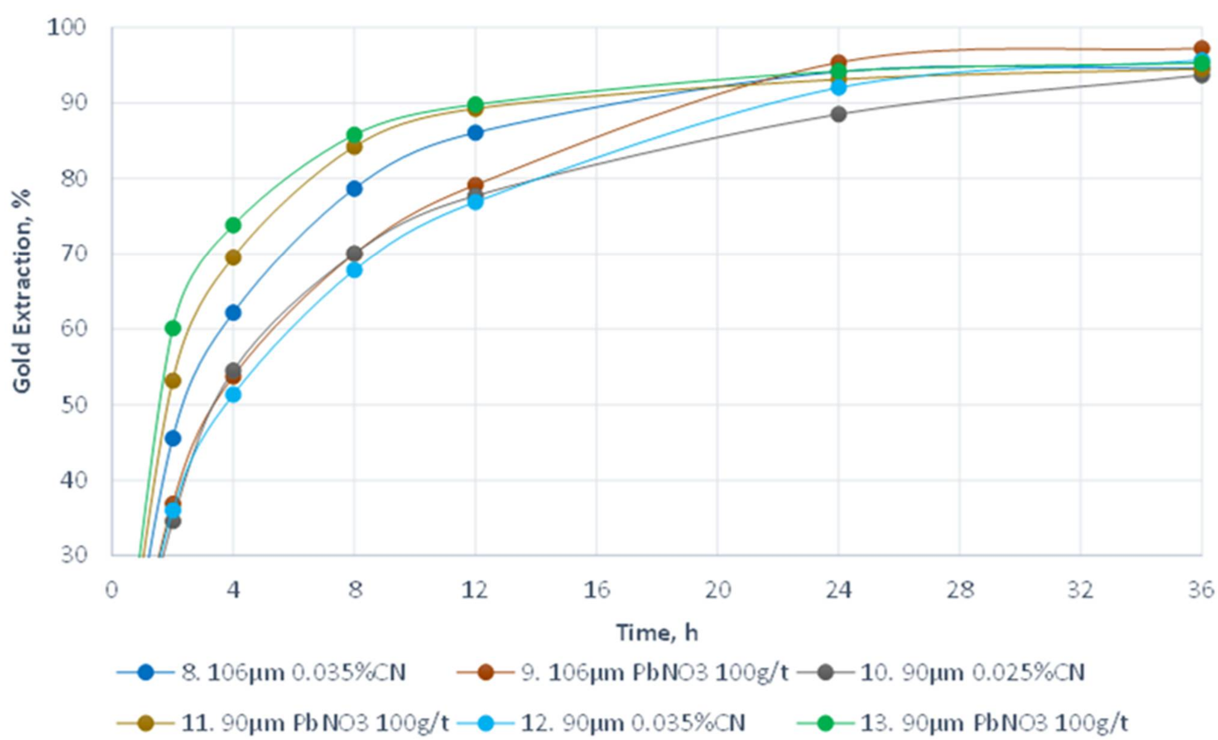


Figure 13-38: Master Composite Lead Nitrate, Cyanide and Grind Comparison

13.11.3.1 Gravity Assessment – Whole of Ore Leach Test

The benefit of a gravity gold recovery stage on the overall gold extraction was evaluated (test 2).

A whole of ore leach test was conducted on the Average Grade Master Composite to compare the gold recovery with and without removal of the gravity concentrate, as illustrated by the red curves in Figure 13-39. The whole of ore leach shows that the coarser gold associated with the gravity leads to very slow leach kinetics. The test results showed that the whole of ore leach achieved a lower overall recovery of 77% after 24 h as compared to the gravity and leach test, which achieved an overall gold recovery of 98% after 24 h leach.

A final whole of ore leach gold tail grade of 1.23 g/t was achieved compared to 0.03 to 0.06 g/t for the master composite sample gravity test series.

The average, 15th and 85th percentile leach kinetics curves for the 40 variability samples at a both P80 grind sizes of 75 and 106 µm are illustrated by the dashed lines. The master composite tracks the average leach kinetics curve of the variability samples, whereas the whole of ore leach curve is well below the 15th percentile of the variability samples.

This performance strongly supports the inclusion of a robust gravity recovery stage and operation of this circuit at all times.

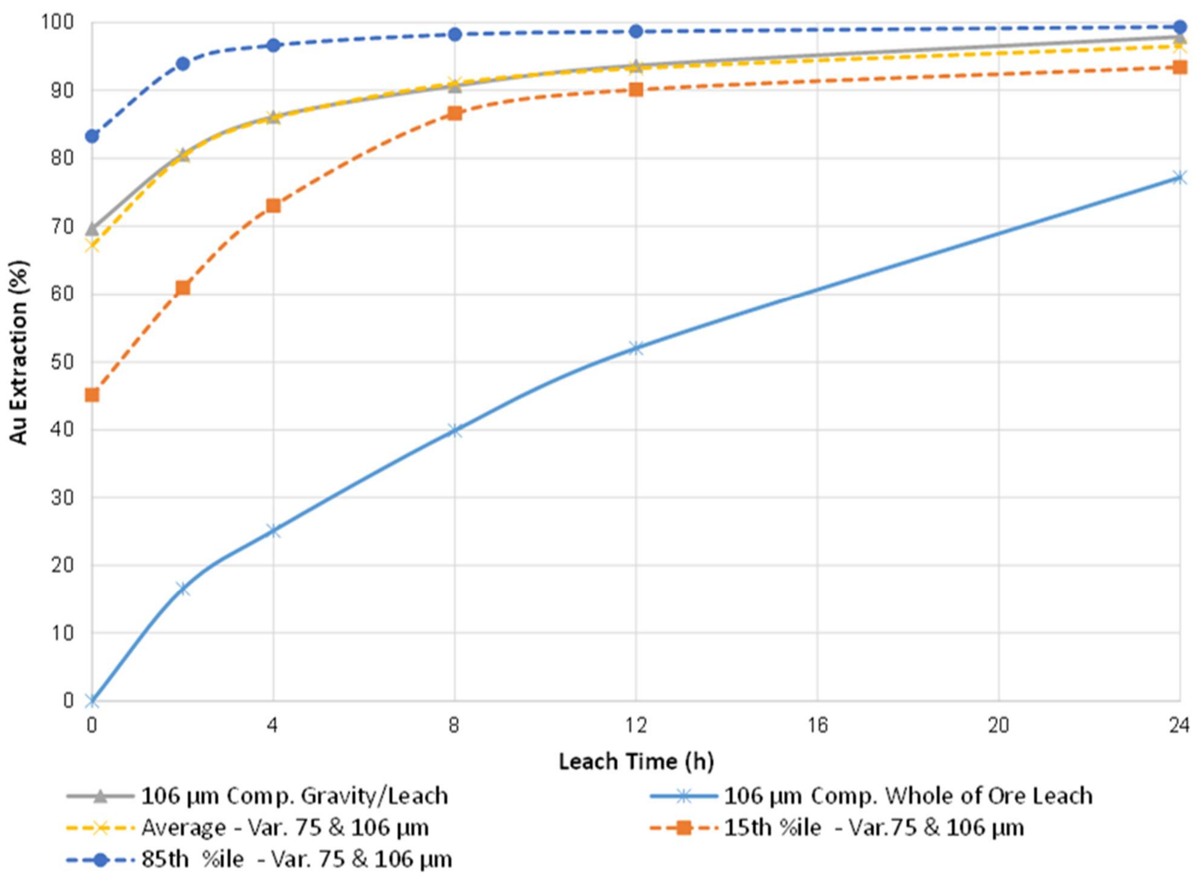


Figure 13-39: Leach Kinetics Comparing Whole of Ore with Gravity/Leach

13.11.3.2 Final Leach Solution Assay

A full ICP scan of the Average Grade Master Composite leach tails solution was conducted to determine the base metals and other elements in solution. This multi-element assay also indicates if there are any species of concern in the leach solution that may affect the carbon adsorption or tails disposal. As can be seen from the data in Table 13-30, the solution has very few elements with significant solution concentrations. This is consistent with the very low reagent consumption indicating that few side reactions are occurring.

Table 13-30: 2021 Master Composite Cyanidation Tails Solution Assay

Analyte	Units	AM1304 Final Leach Solution
Ag	mg/l	0.54
Al	mg/l	1.80
Au	mg/l	0.850
Ca	mg/l	6.50
Cu	mg/l	1.82
Fe	mg/l	1.60
Hg	mg/l	0.004
K	mg/l	14.0
Mg	mg/l	1.60
Na	mg/l	320
Ni	mg/l	0.10
Pb	mg/l	<0.05
Sr	mg/l	0.04
Te	mg/l	<0.01
Ti	mg/l	<0.10
Zn	mg/l	0.82

Most of the below detection limit elements have been deleted and the only significant element is Na from the cyanide addition. The very low base metal dissolution indicates that most of the tails cyanide will be free, with only a small CN_{WAD} content. This is an ideal scenario for cyanide recovery within the circuit rather than destruction.

Table 13-31: Summary of Master Composite Bulk Gravity and Leach Test Results

	Test #	Grind Size P_{80} (µm)	Flowsheet	Residue Grade (Au g/t)	Gravity Gold Extr'n (%)	Total Gold Extr'n (%)	Calc. Head (Au g/t)	NaCN Add'n (kg/t)	NaCN Cons. (kg/t)	Lime Add'n (kg/t)	Leach Only Au Extr'n @Time (2 h)	Leach Only Au Extr'n @Time (4 h)	Leach Only Au Extr'n @Time (8 h)	Leach Only Au Extr'n @Time (12 h)	Leach Only Au Extr'n @Time (24 h)	Leach Only Au Extr'n @Time (36 h)
1	AM1205	106	Gravity Separation / Direct Cyanidation	0.06	69.7	97.9	2.59	0.25	0.17	0.31	36.0	54.2	69.2	79.1	93.0	
2	AM1275	106	Whole of Ore Direct Cyanidation	1.23	NA	77.2	5.40	0.21	0.07	0.34	16.5	25.1	39.9	52.0	77.2	
3	AM1301	106	$PbNO_3$ 100 g/t	0.05	46.5	96.2	1.30	0.29	0.07	0.30	55.2	72.2	85.3	87.9	91.4	92.8
4	AM1302	106	$PbNO_3$ 200 g/t	0.04	45.7	97.0	1.32	0.31	0.07	0.28	58.6	75.6	86.7	89.2	91.6	94.4
5	AM1303	106	4 h Pre-Ox @ pH 11.0	0.05	47.3	96.1	1.28	0.31	0.04	0.32	37.0	61.5	79.5	85.4	90.6	92.6
6	AM1304	106	4 h Pre-Ox @ pH 11.0, $PbNO_3$ 100 g/t	0.04	46.0	97.0	1.32	0.29	0.07	0.30	53.5	71.2	84.0	88.1	92.5	94.4
7	AM1305	106	4 h Pre-Ox @ pH 12.0	0.05	46.0	96.6	1.32	0.29	0.04	1.00	26.5	49.8	73.8	82.5	91.3	93.7
8	AM1311	106	106 µm 0.035% NaCN	0.04	81.7	99.0	4.03	0.34	0.10	0.28	45.6	62.2	78.6	86.1	94.1	94.6
9	AM1312	106	106 µm $PbNO_3$ 100g/t	0.03	75.4	99.3	4.37	0.34	0.10	0.28	36.9	53.8	70.0	79.1	95.3	97.2
10	AM1313	90	90 µm 0.025% NaCN	0.05	82.3	98.9	4.00	0.20	0.05	0.32	34.6	54.5	70.0	77.7	88.5	93.7
11	AM1314	90	90 µm $PbNO_3$ 100g/t, 0.025% NaCN	0.04	82.0	99.0	4.01	0.20	0.05	0.32	53.2	69.5	84.2	89.2	93.1	94.5
12	AM1315	90	90 µm 0.035% NaCN	0.04	78.1	99.1	4.21	0.31	0.09	0.28	36.0	51.4	67.8	76.9	92.0	95.7
13	AM1316	90	90 µm $PbNO_3$ 100g/t, 0.035% NaCN	0.03	83.9	99.2	3.92	0.33	0.08	0.26	60.1	73.8	85.8	89.8	94.2	95.3
		75	**75 µm Variability Average**	**0.05**	**71.2**	**97.8**	**4.28**	**0.37**	**0.08**		**44.1**	**61.3**	**76.5**	**82.7**	**92.4**	

13.12 Determination of Engineering Design Parameters

Following the comminution and metallurgical testing phases it was also necessary to define physical characteristics of the slurry and the engineering design parameters required for implementation of the process selected. This phase of work included oxygen uptake tests, standard carbon loading (kinetic) tests, cyanide detoxification tests, slurry rheology and dynamic thickening testwork.

13.12.1 Aging Testwork

The aging tests aimed to investigate the potential for sulphides in the ore to oxidise in the crushed ore or ROM stockpiles over time, resulting in more reactive minerals that might negatively impact the gold leaching when treated. The tests used a sub sample of the crushed master composite sample with a P_{100} of 3.35 mm stored under hot, humid conditions to simulate accelerated aging (oxidising) conditions that may occur for stockpiled ore. The effect of progressive aging was investigated by conducting standard grind, gravity, leach tests at two weekly intervals on sub-samples of the aging composite, over an eight-to-nine-week period. The impact of progressive oxidation on the metallurgical performance was observed by comparing the leach test outcomes.

13.12.1.1 2019 Master Composite Testing

The 2019 aging tests were originally planned for the high-grade fresh composite sample, which assumed that high grade gold would be associated with increased sulphide mineralisation. However, the Lafigué ores do not display typical sulphide gold association with the sulphides that are present being more randomly distributed. As a result, the aging testwork was performed on the fresh master composite which did indicate minor pyrrhotite content in the mineralogical investigation, despite generally low sulphur grades for most of the constituent samples.

The test results are summarised in Table 13-32 and appear to show an initial decrease in gold extraction compared to the baseline. However, the variability in the head assay and GRG content complicates comparison of the results.

Table 13-32: 2019 Aging Testwork - Fresh Master Composite Leach Results

Test ID	Aging Weeks	Calc. Head Au g/t	Tail Au g/t	24-hour Gold Extraction %			Initial DO mg/L	Reag. Cons. (kg/t)	
				GRG	Leach	Overall		NaCN	Lime
JS4501	0	3.00	0.06	78.3	90.8	98.0	8.4	0.08	0.23
JS4534	2	1.57	0.08	34.3	92.3	94.9	7.6	0.05	0.25
JS4535	4	1.33	0.09	37.2	89.2	93.2	7.8	0.07	0.25
JS4536	6	1.70	0.10	49.8	88.2	94.1	7.8	0.07	0.27
JS4537	9	2.88	0.07	69.0	92.2	97.6	6.3	0.07	0.28

The tails grades do increase slightly with increased sample age, but there was a high degree of variability between sub-samples in terms of gold mineralisation. The improved final sample extraction suggests the tails variance may simply be a feature of the higher grade, high GRG mineralisation versus the slightly lower recovery, lower grade mineralisation. It appears unlikely that aging/sulphide association is a major factor in the leaching differences. The cyanide consumption was comparable for all tests. The decreasing dissolved oxygen levels and increasing lime required may be indicators of some reactive sulphide oxidation, but this is not significant being within the expected range of reagent addition rates.

Although the result of this testwork is not definitive, the fresh master composite had only average pyrrhotite grades and was not the selectively higher example intended. In principle with ores of this nature, quick stockpile turnaround times are advocated. Oxide ores and those with lower sulphide grades can be more readily stockpiled without adverse effects.

13.12.1.2 2021 Master Composite Testing

The 2021 aging testwork was conducted on two variability samples, MET 20 and MET 27, that had higher sulphur assays (1.08% and 1.38% sulphide sulphur respectively; 0.28% variability average). Gravity and leach tests were performed at a P_{80} grind size of 106 µm on sub-samples of the aged test samples at two weekly cycles to observe the impacts of progressive oxidation.

The test results show a general trend corresponding to an increase in overall gold extraction with aging compared to the baseline test conducted at week 0. This is further confirmed by the residue assays which showed a corresponding decrease in gold grade.

Table 13-33: 2021 Aging Test - Gold Extraction vs. Oxidation Time

Test ID	Aging Duration (Weeks)	Calc. Head (Au ppm)	Gold Tail (ppm)	Gold Extraction			Initial DO Level	Consumption (kg/t)	
				GRG (%)	24 h Leach (%)	GRG and Leach (%)	ppm	NaCN	Lime
MET 20									
AM1162	0	1.81	0.38	36.84	42.11	78.95	0.9	0.17	0.65
AM1224	2	1.74	0.12	28.62	64.48	93.10	6.9	0.15	0.54
AM1226	4	1.93	0.20	33.92	55.96	89.88	7.4	0.07	0.58
AM1228	6	1.88	0.15	24.07	67.96	92.02	7.3	0.09	0.47
AM1230	8	1.88	0.15	31.73	60.27	92.00	7.6	0.11	0.45
MET 27									
AM1176	0	1.91	0.14	59.98	32.69	92.67	6.4	0.07	0.40
AM1225	2	1.71	0.06	66.79	29.99	96.78	7.4	0.07	0.27
AM1227	4	1.72	0.05	57.55	39.84	97.38	7.9	0.05	0.29
AM1229	6	1.99	0.03	72.88	25.86	98.75	7.4	0.06	0.28
AM1231	8	1.83	0.03	82.86	15.78	98.63	8.4	0.04	0.20

Met 20 sample displayed higher cyanide consumption during the Week 0 and Week 2 periods. Met 27 sample showed a consistent cyanide consumption throughout the entire aging test period. Both Met 20 and Met 27 samples displayed an initial higher lime consumption, which stabilised by Week 2.

The aging tests on the two samples with relatively high sulphide content, confirmed that there were no adverse effects with ore aging. The overall gold recovery improvement with aging suggests that the sulphide oxidation that may be occurring is metallurgically beneficial.

13.12.2 Oxygen Uptake Rate Tests

The oxygen uptake rate test provides an instantaneous measure of the natural usage of oxygen by the ore for side reactions such as sulphide mineral oxidation. Clays in the oxide ores can also demand significant oxygen. This test is used to assess whether oxygen sparging is necessary to satisfy the oxygen demand of the sample to allow dissolution of the gold and silver, or if simple aeration with low pressure air is sufficient.

Oxygen uptake rate tests were performed with air sparging on the fresh and oxide master composites under the standard leach conditions of a grind size P80 of 106 µm, pulp densities of 55% w/w solids for the fresh composite and 40% w/w solids for the oxide composite, pH 10.5 and a starting cyanide concentration of 0.025% w/v. The results are shown in Table 13-34 and Table 13-35.

The standard test procedure used for the oxygen uptake rate determination was adopted. A typical cyanidation slurry sample is aerated in a mechanically agitated vessel. At each measurement interval, aeration is stopped, the agitator slowed to avoid inducement of air, and the rate of decay of the dissolved oxygen (DO) concentration measured at one-minute intervals over a 15-minute period. Thereafter agitation returns to normal and air injection recommences.

Table 13-34: 2019 Testwork Oxygen Uptake Rate Results Summary

Composite	Oxygen Uptake Rate ($mg.L^{-1}.min.^{-1}$) at Elapsed Time (hours)							
	0	1	2	3	4	5	6	24
Oxide Composite	0.0018	0.0011	0.0012	0.0025	0.0020	0.0025	0.0006	0.0014
Fresh Composite	0.0055	0.0043	0.0054	0.0064	0.0065	0.0051	0.0062	0.0030

Table 13-35: 2021 Summary of Oxygen Uptake Rate Results

Composite	Oxygen Uptake Rate ($mg.L^{-1}.min.^{-1}$) at Elapsed Time (hours)							
	0	1	2	3	4	5	6	24
Ave. Grade Master Composite	0.0126	0.0118	0.0067	0.0101	0.0085	0.0077	0.0042	0.0035
O Composite	0.0000	0.0026	0.0061	0.0033	0.0072	0.0024	0.0006	0.0899
P Composite	0.0119	0.0186	0.015	0.0144	0.014	0.0125	0.0074	0.0069

The results indicate that the oxygen demand is particularly low for all of the composites tested although this might be greater if the gravity recoverable gold and possibly coarse reactive sulphides were not removed ahead of the leach. The low oxygen uptake rate is consistent with the high DO levels with air sparging throughout the testing to date and the lack of improvement resulting from oxygen sparging in the leach. The ores are clearly very clean with few side reactions that might demand oxygen.

13.12.3 Rheology Tests

An understanding of pulp rheology is fundamental to establish materials handling characteristics for the various pumping duties and to enable optimisation of mass transfer within the leach and adsorption circuits (launders and intertank screens). High viscosity slurries may cause pumping difficulties and impact on flow through the intertank carbon screens. Tank agitation, air/oxygen dispersion and leach/adsorption mass transfer kinetics within leach/adsorption and reaction rates in the detoxification circuit are all strongly dependent on slurry viscosities in the tanks.

13.12.3.1 2018 Scouting Rheology Testwork

Rheology testwork was completed on the scouting programme oxide and fresh composite slurries at a grind size of P_{80} 106 µm, pH 10.5 and slurry densities between (40 and 60)% w/w solids.

The fresh composites had low to moderate viscosities and conventional agitators and pumps will be suitable.

The oxide composite tested had extreme viscosity - the slurry was diluted to 32.5% w/w solids in order to achieve a test reading, however the slurry viscosity remained very high. The oxide composite exceeded the viscosity limits for CIL and centrifugal pumping. This sample appears to be an exception based on subsequent testwork but blending of this type of ore with lower viscosity fresh ore at all times (at a low oxide proportion) is recommended to improve material handling and thickener settling rates and ensure that the agitators and pumps can perform in line with their design intent. The high viscosity oxide composite results are summarised in Table 13-36 and Figure 13-40 for comparison.

13.12.3.2 2019 Rheology Tests

Rheology testwork was undertaken on the 2019 oxide and fresh master composites ground to a P_{80} of 106 µm with the pH adjusted to 10.5 using lime. Testing was conducted on the bulk leach tails samples at 40, 50, 55 and 60% w/w solids for the fresh ore and 35, 40 and 45% w/w solids for the oxide in order to investigate slurry flow properties. Further testing on a blended 40% oxide/60% fresh composite at 45, 50 and 55% w/w solids was also conducted.

A summary of the viscosity testing results (slurry at ambient temperature, viscosity measured in centipoise, cP) against nominated shear rates per second is presented in Table 13-36 and Figure 13-41. The figure also shows the accepted viscosity design limits for equipment types.

The viscosities measured indicate that these slurries should be readily managed through the proposed treatment plant with low to moderate slurry viscosities across the range of densities tested.

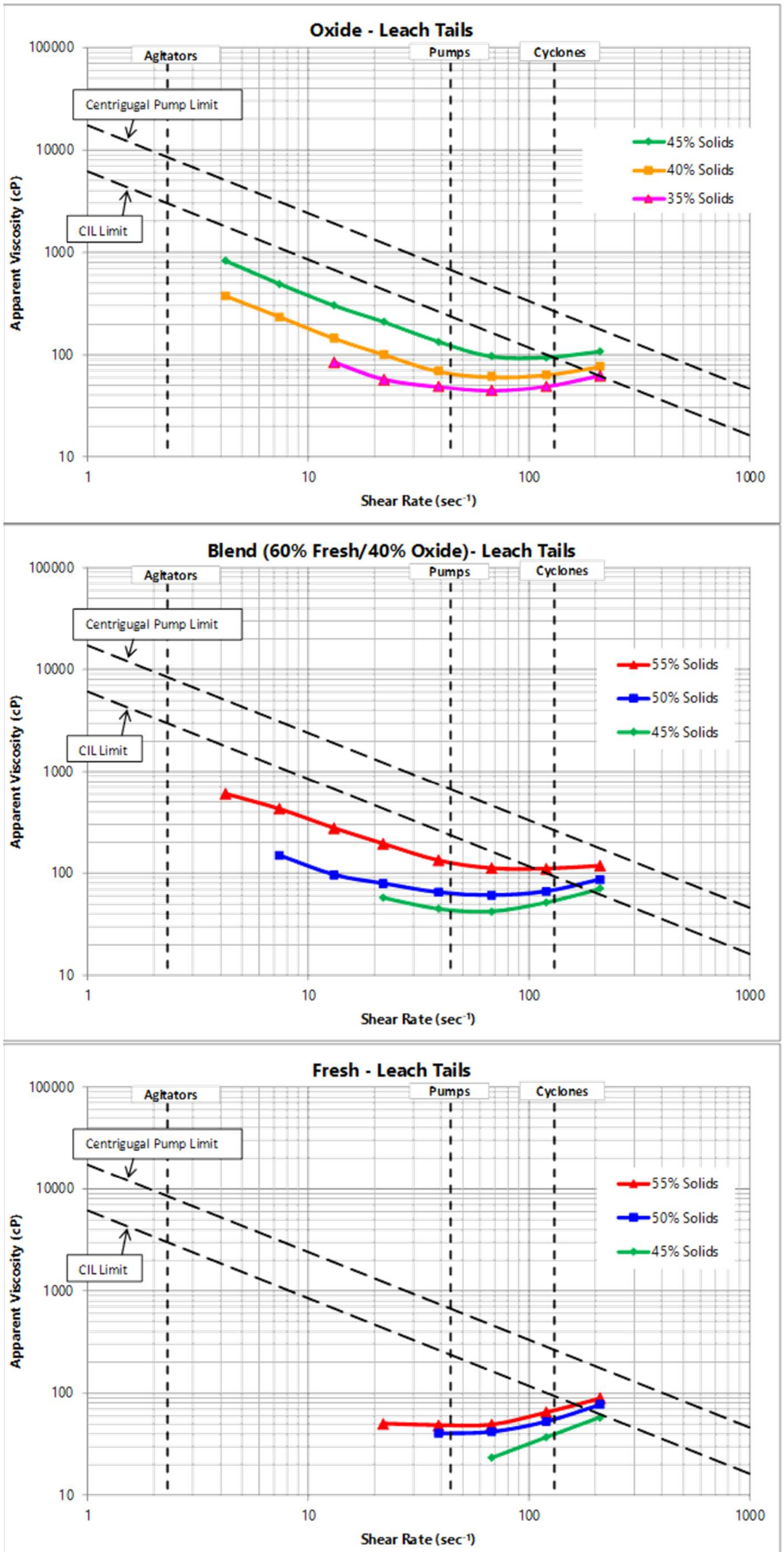


Figure 13-41: 2019 Testwork Slurry Viscosity vs. Shear Rate

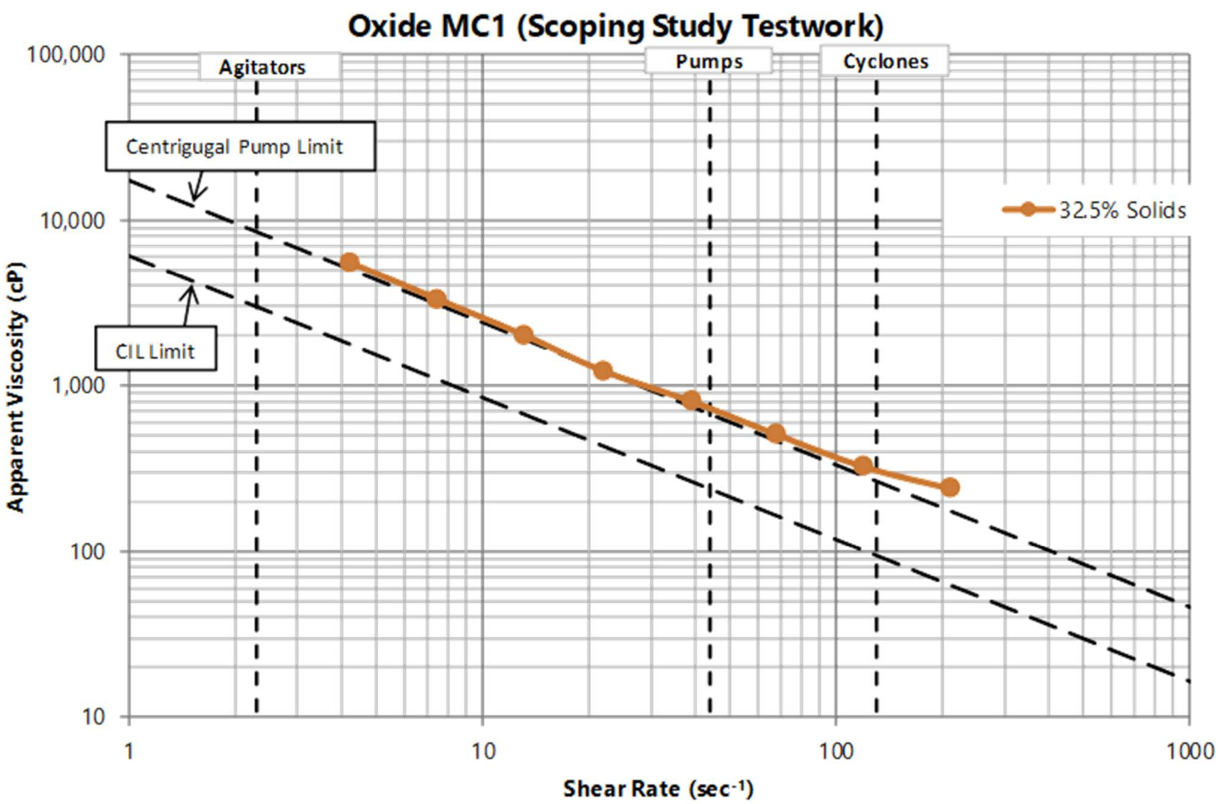


Figure 13-40: 2018 Testwork High Viscosity Oxide Composite Results

Table 13-36: 2018 and 2019 Testwork – Rheology Results Summary

	Sample	% Solids w/w	Apparent Viscosity (cP) @ Shear Rate (s^{-1})							
		Shear Rate, s^{-1}	4.2	7.4	13.1	21.9	38.9	67.4	119.2	209.5
2019 PFS Testwork	Oxide	35			84	57	48	44	49	62
		40	374	234	144	100	69	61	63	77
		45	823	489	300	208	133	96	94	107
	Fresh	45						23	37	58
		50					40	42	53	77
		55				50	48	49	65	89
	Blend (60%Fr: 40% Ox)	45				57	44	42	51	
		50		149	96	79	65	61	66	
		55	599	425	276	194	133	112	111	
2018 SS Testwork	Oxide	32.5	5539	3335	2016	1229	808	513	327	242
		40	Oxide sample was too viscous to test at higher % solids.							
		45								
	Fresh (High Grade)	40							24	47
		50						14	34	60
		60					48	58	76	108
	Fresh (Low Grade)	40							21	44
		50						16	30	32
		60				79	82	85	88	113

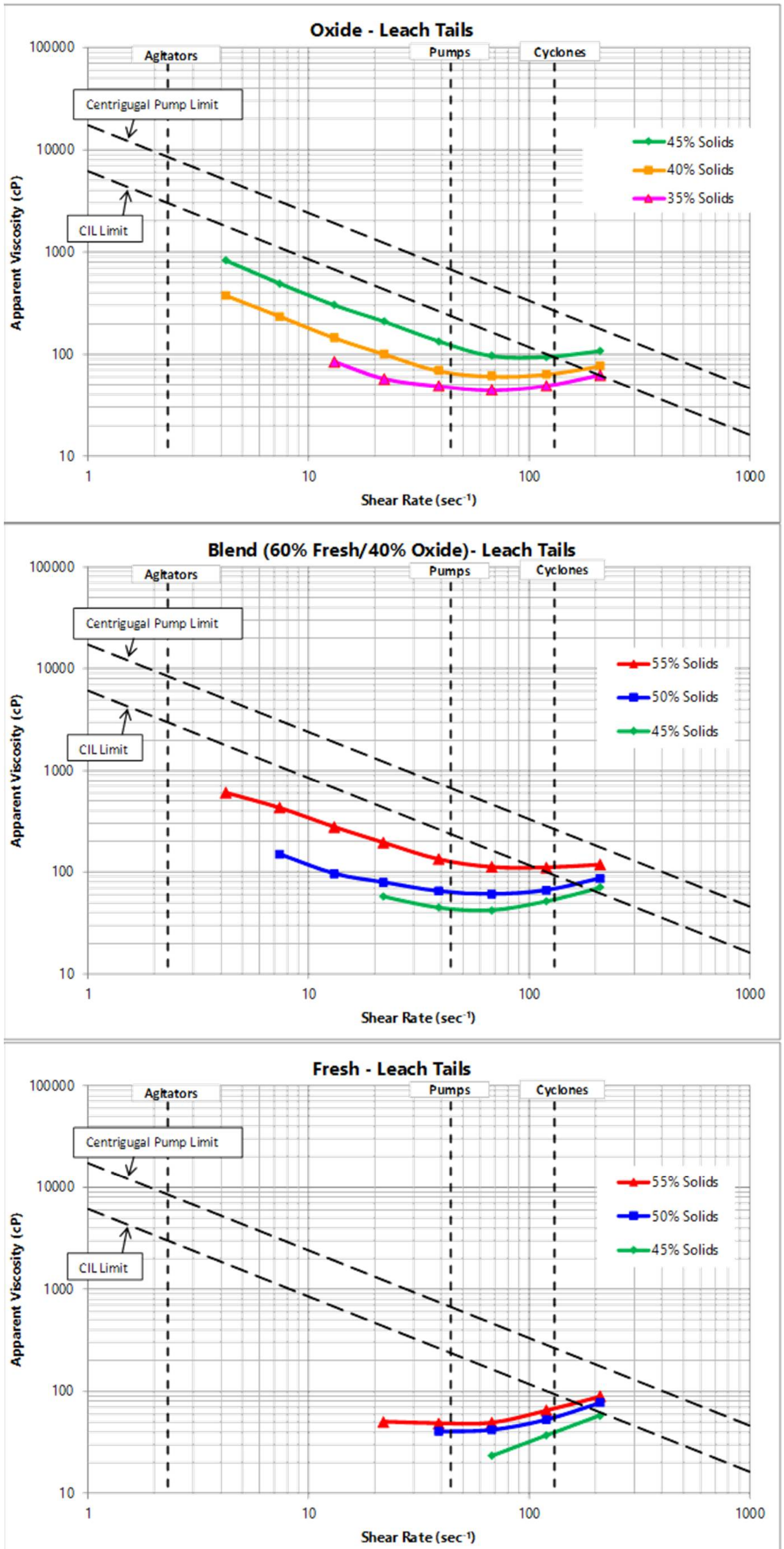


Figure 13-41: 2019 Testwork Slurry Viscosity vs. Shear Rate

13.12.3.3 2021 Rheology Tests

Rheology testwork was undertaken on the bulk cyanidation tailings of the Average Grade Master Composite sample ground to P_{80} 106 µm. Testing was conducted at 55, 60, 65 and 70% w/w solids to investigate the full range of slurry flow properties.

A summary of the viscosity testing results is presented in Table 13-37 and a plot of the apparent viscosity against shear rate is presented in Figure 13-42. This figure also shows the accepted viscosity design limits for equipment types.

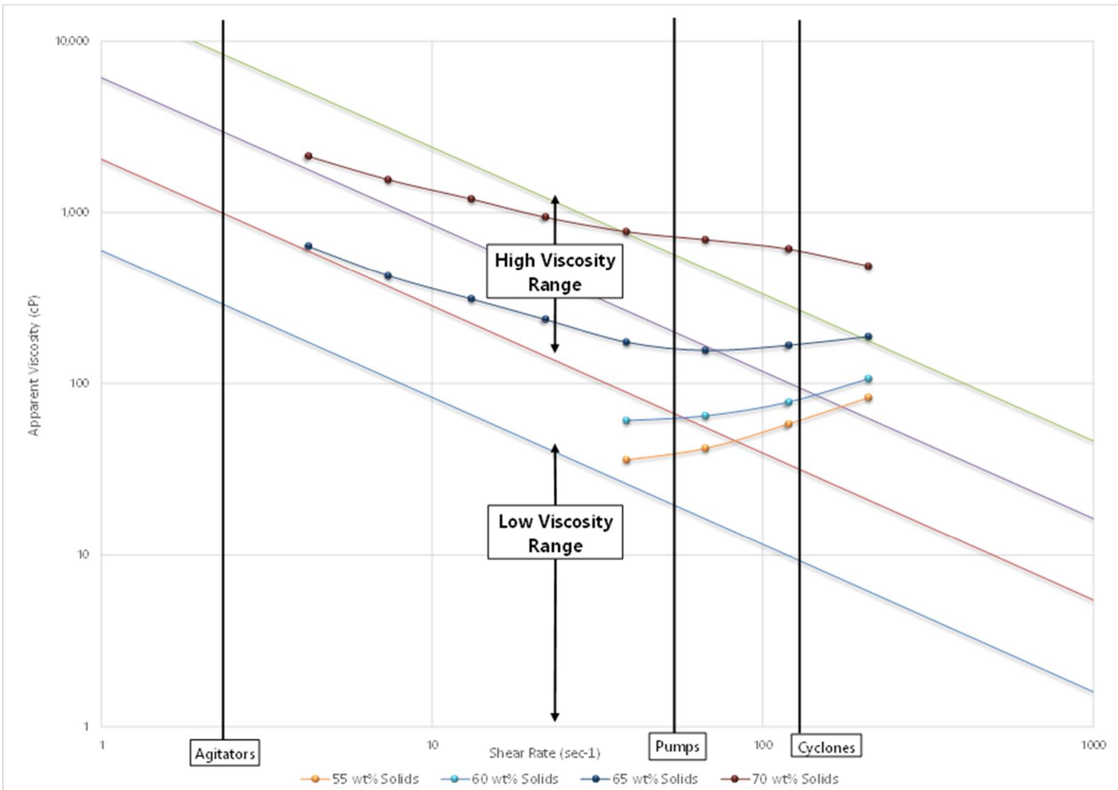


Figure 13-42: Shear Rate vs. Apparent Viscosity

The Average Grade Master Composite sample shows that at the proposed plant slurry solids content of 50% to 55 % by weight, viscosity is not expected to be a problem for the proposed operational units. Operation at above 65% solids is unlikely, but it is useful to have the upper viscosity limit established.

Table 13-37: 2021 Summary of Rheology Results

Sample	% Solids w/w	Bohlin Visco 88 Viscosity (cP) @ Shear Rate (s^{-1})							
	Shear Rate, s^{-1}	4.2	7.4	13.1	21.9	38.9	67.4	119.2	209.5
Ave. Grade Master Composite	70	2133	1551	1200	940	772	693	612	485
	65	636	425	312	237	174	156	167	188
	60	0	0	0	0	61	65	78	107
	55	0	0	0	0	36	42	58	83

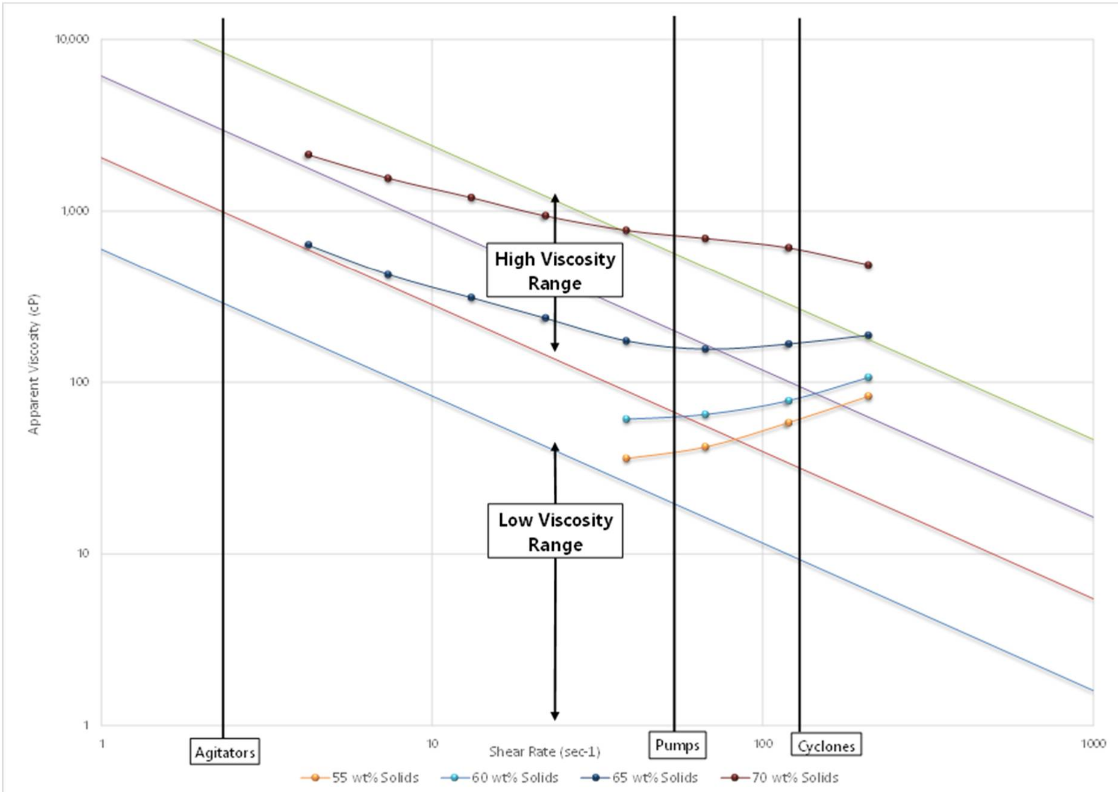


Figure 13-42: Shear Rate vs. Apparent Viscosity

13.12.4 Carbon Adsorption Tests

Carbon adsorption kinetics were characterised by using the sequential triple contact adsorption test to determine the Fleming k and n rate equation constants for the CIL/CIP system. Slurry from the fresh composite bulk leach test under optimised conditions following gravity separation was used for the adsorption testing. The results of the Fleming constants arising from the triple contact sequential carbon loading kinetic tests are summarised in Table 13-38.

Table 13-38: 2019 Testwork Summary of Fleming Constants for Carbon Adsorption

Fleming Rate Constant, k (h^{-1})	Fleming Equilibrium Constant, n	Cumulative Carbon Loading (g/t C)
184	0.687	917

The carbon loading kinetics are within the range typically observed in CIL/CIP operations. The residual gold in the fresh gravity tailings is relatively low, hence the low overall carbon loading. Carbon loading and adsorption kinetics would likely increase, should higher gold solution grades be experienced with lower gravity recoveries or higher leach feed head grades.

13.12.5 Cyanide Detoxification Tests

The SO_2/Air oxidation process was adopted for the cyanide detoxification testwork. In this process, a mixture of sulphur dioxide (SO_2) and air is used to oxidise free cyanide and weakly complexed cyanide to cyanate (OCN) at pH 8 to 10, using copper as a catalyst.

Standard continuous cyanide detoxification tests were performed on the fresh and blended oxide/fresh composites using slurry from the sequential carbon adsorption tests. An ICP scan of the bulk leach tails solution was conducted to determine the cyanide assays and base metals in solution. The multi-element assay also indicates if there are any deleterious ions in the tails solution that need to be addressed. As can be seen from the data in Table 13-39, the solution has very few elements with significant solution concentrations.

Table 13-39: 2019 Testwork Bulk Cyanidation Tails Solution Assay

Element	JS4501 Fresh Ore Bulk Leach Tails (mg/L)	Fresh Ore Detox Tails Solution (mg/L)	JS4501 + JS4513 35% Oxide Blend Bulk Leach Tails (mg/L)	35% Oxide Blend Detox Tails Solution (mg/L)
CN_{free}	78.5	n/a	67.6	n/a
CN_{wad}	83.6	0.49	73.7	2.90
pH	9.70	8.54	9.43	8.51
Ag	0.18	<0.02	0.30	<0.02
Al	0.40	0.20	<0.20	0.20
Cu	3.98	0.06	4.28	0.04
Fe	9.40	0.80	3.30	2.50
K	20.0	41.0	5.00	18.0
Mg	1.20	14.8	0.80	10.4
Mo	0.05	0.10	0.05	0.10
Ni	0.25	<0.05	0.55	0.40
Pb	<0.05	<0.05	<0.05	<0.05
Sr	0.04	0.30	0.06	0.46
Zn	0.14	<0.02	0.34	<0.02

The CN_{WAD} concentration was determined directly by a colorimetric method, using picric acid reagent. Cyanide determinations using this method are identified as CN_P and the values obtained are fundamentally equivalent to CN_{WAD}. A residence time of nominally 60 minutes was selected.

A summary of the cyanide detoxification testwork is provided in Table 13-40.

Further cyanide detoxification testwork was conducted on the 2021 Average Grade Master Composite. A summary of the cyanide detoxification testwork is provided in Table 13-42.

Table 13-40: 2019 Testwork - Summary of Cyanide Detoxification Testwork

Sample	Slurry Solids (% w/w)	Slurry pH	Solution Analysis (mg/L)						
			Titrated NaCN	Calc CNT	CNwad	Cu	Fe	Ni	Zn
JS4501 (Fresh)	55	9.70	150	110	83.6	3.98	9.40	0.25	0.14
JS4513 (35% Oxide Blend)	48	9.43	130	83	73.7	4.28	3.30	0.55	0.34
Detox Tails (average)		8.54		6.1	1.49	0.06	1.65	0.21	<0.02

Table 13-40 notes:

- 6:1 SO2:CNWAD ratio
- Tails solution assays are averages of last two samples

Table 13-41: 2019 Testwork – Reagent Consumptions

Reagent Consumption (kg/tonne of solids)			Reagent Consumption (kg/m³ of solution)		
$Na_2S_2O_5$ (kg/t)	$CuSO_4.5H_2O$ (kg/t)	Lime (60% CaO) (kg/t)	$Na_2S_2O_5$ (kg/m³)	$CuSO_4.5H_2O$ (kg/m³)	Lime (60% CaO) (kg/m³)
0.66	0.14	1.02	0.70	0.16	1.04

Table 13-42: 2021 Summary of Cyanide Detoxification Testwork (Avg. Grade Master composite AM1205)

Sample	Solids (% w/w)	Slurry pH	Solution Analysis (mg/L)						
			Titrated NaCN	Calc CN_T	CN_{wad}	Cu	Fe	Ni	Zn
Detox Feed	55	9.7	100	120	59.8	5.74	21.6	0.95	0.18
Detox Treated Effluent, D1		8.6		8.41	1.56	0.13	2.45	0.10	<0.02

Table 13-43: 2021 Testwork - Reagent Consumptions

Reagent Consumption (kg/tonne of solids)			Reagent Consumption (kg/m³ of solution)		
$Na_2S_2O_5$ (kg/t)	$CuSO_4.5H_2O$ (kg/t)	Lime (60% CaO) (kg/t)	$Na_2S_2O_5$ (kg/m³)	$CuSO_4.5H_2O$ (kg/m³)	Lime (60% CaO) (kg/m³)
0.38	0.10	0.15	0.46	0.13	0.18

The cyanide detoxification testwork indicated the following:

- CNWAD concentrations in the cyanidation tailings slurry comprised mainly CNfree with only trace base metal concentrations in solution. The discharge liquor following cyanide detoxification for each of the composites was reduced to <3 ppm CNWAD.
- The target CNWAD value of <5 mg/L in the cyanide detoxification discharge liquor was achieved with a standard excess stoichiometric addition of SMBS (Na2S2O5). Optimisation of the reagent dosage was not included in the scope given the low CNWAD levels present.
- It is not intended to include cyanide detoxification in the flowsheet. This testwork served only to demonstrate that the SO2-Air detoxification could be implemented successfully if required. With the low CNWAD levels, cyanide in solution recovery from the tails thickener overflow will be practised, with dilution of the underflow reporting to tails to achieve target discharge levels.

13.12.6 Thickening Testwork

Thickening testwork was completed on a fresh and oxide/fresh blend slurry composite. The testing involved:

- Flocculant type screening to allow selection of the optimum flocculant type and dosage rate range.
- Flux rate determination to assess the range of feed densities and solids loadings to be tested in the dynamic thickening test rig.

- Dynamic settling tests at various solid loadings and flocculant dosage rates to determine ultimate settled density and overflow solution clarity.

Settling rates for the fresh ores were moderately fast with high underflow densities and clear overflow at low flocculant dosages (10 g/t). The oxide blend settled more slowly requiring higher flocculant dosages (30 g/t) at the same flux rates. Lower underflow densities were achieved, suggesting that the maximum operating oxide blend should be reduced, despite the relatively low viscosities recorded. The poorer oxide thickening performance, appears to be mainly related to the natural fineness of the material. The yield stresses were not measured for these test products.

Table 13-44: 2019 Testwork - Thickening Test Results for Oxide Blend and Fresh Ore

Ore Type	Settling / Flux Rate		Product U/F		Flocculant	
	Flux Rate ($t.m^{-2}.h^{-1}$)	Feed Solids % w/w	U/F (% w/w)	Yield Stress (Pa)	Testwork (g/t)	Design (g/t)
35% Oxide Blend	0.8	5	58.9	n/a	30	45
Fresh Composite	0.8	5	64.6	n/a	10	15

Further thickening testwork was commissioned on the 2021 bulk leach tailings since more sample was available than had been previously. Cyanide detoxification tailing from the Average Grade Master Composite sample was used for the thickening testwork.

Based on static cylinder tests, BASF Magnafloc 10 was selected for the dynamic thickening testwork. The dynamic thickening tests were conducted with a flocculant dosage rate of 30 g/t and Perth tap water adjusted to a pH of 10 with lime. Settling rates for the sample were moderately fast with a high underflow density and a clear overflow as shown in Table 13-45.

Table 13-45: 2021 Fresh Ore Dynamic Thickening Test Results

Test	Settling / Flux Rate		Product U/F		Overflow	
	Flux Rate ($t.m^{-2}.h^{-1}$)	Liquor Rise Rate (m/h)	Feed Solids (% w/w)	(% w/w)	Yield Stress (Pa)	Clarity (mg/L)
Run 1	0.50	2.46	18.0	65.6	44	100
Run 2	1.00	4.92	18.0	62.6	46	150
Run 3	0.75	3.69	18.0	63.5	35	130
Run 4	0.25	1.23	18.0	66.8	55	110

The design thickening flux rate of 0.75 $t.m^{-2}.h^{-1}$ was selected providing the best balance of U/F density and overflow clarity with a relatively low yield stress.

It is not planned to feed the plant with high oxide blends, as these will be unsuitable for HPGR processing, so it is assumed that oxide feed contents will not be throughput limiting.

13.12.7 2019 Tailings and Geochemical Testwork

Bulk tailings samples were prepared for tailings settling and geochemical testwork carried out by the consultants responsible for the design of the tailings storage facility (Knight Piésold). Further detail is provided in Section 20 of this Report.

13.12.8 Net Acid Producing Capacity Testwork

Acid mine drainage (AMD) prediction analysis was conducted on each of the 40, 2021 variability samples. The analysis indicated that none of the samples were likely to become net acid producers. Further detail is provided in Section 20 of this Report.

13.13 Metallurgical Recoveries and Reagent Consumption

The 2019 and 2021 variability testwork results at the proposed P_{80} grind size of 106 µm, as detailed in Table 13-23 and Table 13-27, were used to estimate the expected plant gold recoveries and the expected plant reagent consumption over a range of plant feed head grades.

13.13.1 Gold Recovery

Although there were a few outliers in the testwork results, following repeat testing under optimised conditions, the majority of the samples have gold extractions clustered around the mean with 24-hour extractions within one standard deviation of the mean of 97.0% for fresh ore (97.7% for the oxide samples). These values aligned fairly consistently with the master composite results although improvements were demonstrated with extended leach time, lead nitrate addition and finer grinding (P_{80} = 90 µm).

The 24-hour median extractions from the variability data sets are presented in Table 13-46 following. A model of the variation with grade was developed to allow application to the block model to estimate recoveries over time. This will improve the returns from the higher-grade blocks while further differentiating more marginal ore blocks where lower gold grades tend to result in below average extractions. An allowance for a typical soluble loss of 0.015 g/t Au equivalent in the leach tails solution was added to the residue grade. This amounts to less than 1.0% gold recovery loss at the median head grade, but is only applied to the leach tails with the gravity recoverable gold being separately accounted for as detailed in Table 13-46. An industry efficiency factor 95% is applied to the tested gravity gold recoveries in line with vendor advice.

Table 13-46: Overall Gold Recovery and Soluble Loss

Item	Unit	Fresh Ore			Oxide Ore
		50th %ile	85th %ile	15th %ile	50th %ile
Median Calculated Head (Variability Samples)	Au g/t ore	2.05			1.59
Overall Gold Testwork Extraction	% Au	97.0%	99.1	95.7	97.7%
Gravity Gold Recovery	% Au	60.6%	80.7	42.4	56.8%
Median Leach Feed Head	Au g/t ore	0.81			0.69
Leach Testwork Gold Extraction	% Au	92.7%	95.6	88.8	95.1%
Soluble Au Loss Equiv.	Au g/t ore	0.015			0.015
Soluble Au Loss	Au Rec. %	1.85%			2.17%
Overall Gold Recovery	**% Au**	**96.4%**			**96.9%**

These recoveries assume treating the Lafigué fresh and blended fresh/oxide/transition ores via the proposed gravity and direct cyanidation treatment route described in Section 17.

Although there are outliers in all the data sets, a good correlation between head grade and gravity gold recovery was discerned for the QTZ hosted samples. Since quartz veining is the predominant host of coarse gravity gold, although often finely striated and therefore nominated as other lithologies, this was deemed representative of the typical mineralisation. The modelled recovery curve, using a fitted log curve with reasonable correlation coefficient, is shown in Figure 13-43.

The following head grade/recovery relationship can be applied to the fresh ore resource model. This formulation is suitable up to 30 g/t head grade. Above this grade recoveries are assumed to be flat at 99.6%:

- Overall Gold Recovery=95% of Gravity Gold Recovery + Leach Recovery less Soluble Loss
- Gravity Gold Recovery=13.52*ln(Head Grade)+54.4 GRG %
- Leach Feed Grade =(100-GRG%)*(Head Grade)LF (g Au/t)
- Leach Recovery=0.97*ln(LF)+92.7 – 1.85 LR, Sol. Loss = 1.85%
- Overall Gold Recovery=95%*GRG + LR*(1-95%*GRG/100)

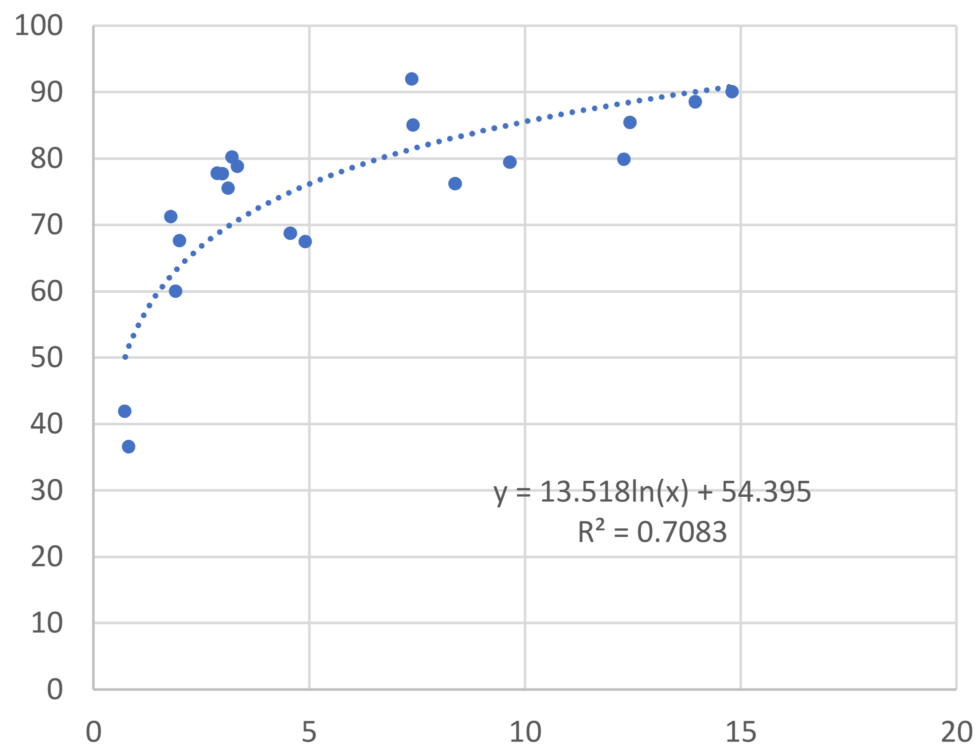


Figure 13-43: Gravity Gold Extraction (%) vs. Head Grade (g/t)

Leach gold extractions are largely clustered about the mean for the data sets. Representing this with a log fit is a poor correlation, but it provides an incremental improvement in recovery with increasing head grade and the data range is quite contained, so this is considered a suitable representation of the data for indicative recoveries about the mean.

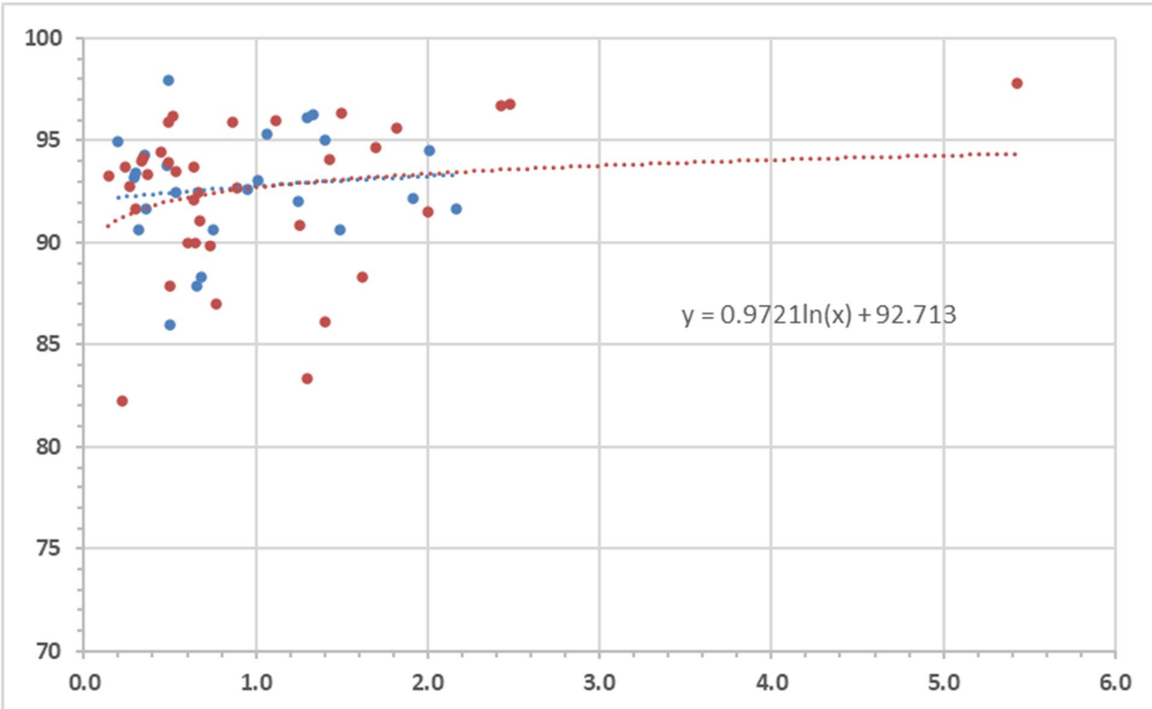


Figure 13-44: Gravity Tails Leach Gold Extraction (%) vs. Leach Feed Grade (g/t)

Silver recovery can be based on the 2019 grind optimisation tests at P80 106 µm in Table 13-16. A soluble loss of 1.5% is assumed, resulting in a 73% silver recovery.

13.13.2 Reagent Consumption

The variability testwork dataset provides a good basis for determining the reagent consumption rates to be used for engineering design and the operating cost estimate inputs.

For the Lafigué ores it is appropriate to consider average reagent usage rates for the operating cost estimates since the usage rates across the testwork have been very consistent for all the tested samples.

Cyanide usage includes an allowance for a free cyanide residual (100 mg/L NaCN) in the CIL tails taking no further credit for cyanide recovery in the tailings thickener or decant return water. Additional allowances for cyanide usage in elution and for the intensive cyanidation reactor will be made.

Cyanide leaching reagent consumption was consistently low. Cyanide consumption for both the fresh and oxide ores was 0.07 kg/t or 0.17 kg/t including the residual at 50% leach solids density.

Lime demand to maintain a pH of 10.5 will typically be low for the fresh ores, but moderate for the oxides. Consumption is based on the median usage in the laboratory variability tests using hydrated lime (nominally 60% CaO).

Lead nitrate will be added on an as required basis at the rate of 100 g/t of ore feed, but it is likely that this rate can be optimised downwards in practise.

No allowance is made for cyanide destruction reagents as cyanide recovery is proposed to reduce discharge levels instead.

Other reagent and consumable usage rates are estimated in the operating cost build-up, Section 17.

Table 13-47: Estimated Plant Leach Reagent Consumption

Ore Feed	NaCN kg/t	Lime kg/t (60% Cao Content)
Oxide	0.17	2.85
Fresh	0.17	0.32

13.14 Data Verification

The methodology applied to verifying/using the data presented in Section 13, and any limitations thereof, are discussed more fully in Section 12.

13.15 Comments for Section 13

The substantial quantity and quality of the comminution and metallurgical test work data developed from the Lafigué drill core samples has facilitated the development of a robust, energy efficient comminution circuit followed by a standard gold recovery process. The metallurgical investigation has provided key process engineering design data, operating cost inputs and gold recovery estimates with a high degree of confidence to support the financial evaluation of the project. The level of testwork and results generated are considered suitable for the requirements of this definitive feasibility study; and the QP considers that the development of the testwork programme and results are in accordance with the CIM's best practise guidelines for mineral processing[82].

13.16 Interpretations and Conclusions

Interpretations, conclusions, and risks for Section 13, are presented in Section 25 of this Report.

13.17 Recommendations

Recommendations for Section 13 are presented in Section 26 of this Report.

13.18 References

References cited in the preparation of Section 13 are presented in Section 27 of this Report.

[82] https://mrmr.cim.org/en/best-practices/mineral-processing/

14. MINERAL RESOURCE ESTIMATES

14.1 Introduction

The 2022 Lafigué MRE was prepared by SRK Consulting (UK) Ltd ('SRK'). SRK has collated the available exploration information for the Lafigué deposit on PE 58 and has prepared a Mineral Resource Estimate (MRE) in accordance with the CIM Definition Standards (2014). SRK is not aware of any environmental or social factors which would preclude the reporting of Mineral Resources at the present time. Table 14-1 summarises the available drilling data. The MRE and accompanying Statement is the responsibility of the Qualified Person, Dr Lucy Roberts (MAusIMM CP).

Table 14-1: Summary of Exploration Drilling Data for the Lafigué Deposit

Period	Type	Number	Total Length (m)
1997	DD	14	1447
	RC	37	1549
2002	RC	32	1281
	RAB*	94	1803
	DD	11	461
2010	RC	11	1109
	DD	4	396
2014	RC	54	4638
	DD	23	1864
2017	RC	179	12464
	DD	17	2197
2018	RC	105	14647
	DD	21	3861
	RC-DD	8	2662
	TRCH*	1	19
2019	RC	228	37 633
	DD	15	2543
	RC-DD	27	7804
	TRCH*	1	17
2020	RC	164	35 207
	RC-DD	126	41 144
2021	RC	412	61 762
	DD	1	207
	RC-DD	61	19 844
2022	RC	222	25 000
	RC-DD	7	1310

Table 14-1 notes:

- *Visually considered during modelling but not included in the MRE
- **Drillholes completed for geotechnical or hydrology purposes are excluded from the above totals
- DD = Diamond core drilling; RC = Reverse Circulation drilling; RC-DD = Reverse circulation with a diamond core tail; TRCH = Trench
- MRE database cut-off date: 15 May 2022

This section describes the methodology used to estimate the Mineral Resources and summarises the key assumptions considered by SRK. SRK considers that the MRE reported herein, is a sound representation of the grade and tonnage of the deposit at the current level of sampling.

Leapfrog Geo version 2021.2 was used to review and model the Mineral Resource estimation domains, prepare assay data for geostatistical analysis, construct the block model, estimate metal grades, and tabulate Mineral Resources.

SRK carried out the following steps to produce the MRE:

- database compilation and review;
- construction of wireframe geological models in Leapfrog Geo 2021.2 software;
- statistical analysis and definition of domains;
- geostatistical analysis (variography) within estimation domains;
- block modelling and grade interpolation using Leapfrog Edge software;
- model validation;
- Mineral Resource classification;
- consideration of reasonable prospects for eventual economic extraction (RPEEE); and
- reporting of Mineral Resources.

14.2 Data Adjustments

The database was directly exported from the Microsoft Access database managed by Endeavour geologists and Endeavour database managers. The following drillhole data was included:

- collars, including collar co-ordinates, drilling type, hole lengths;
- downhole surveys;
- sample assay intervals;
- lithology logging;
- density;
- mineralisation intervals;
- alteration logging;
- logged structures;
- weathering logging; and
- oxidation logging.

Minor adjustments to the database provided were discussed between SRK and Endeavour geologists and rectified prior to continuing with the MRE as part of the data review process; changes included:

- exclusion of drillholes completed for hydrology or geotechnical purposes, where these drillholes were not assayed; and
- where necessary, missing Au values were set to half of the limit of detection ('LOD'), 0.005 g/t.

SRK notes that samples from rotary air blast ('RAB') holes and exploration trenches were not used in the grade estimate but were considered during the generation of mineralisation wireframes.

14.3 Geology and Mineralisation Models

14.3.1 Lithological domains

In order to produce a simplified lithological model, SRK consolidated the logged lithology codes into a refined lithology field. Simplified lithological domains based on four refined lithology codes (intrusive felsic, extrusive felsic, intrusive mafic and extrusive mafic) were produced as intrusions in Leapfrog Geo, along with an overlying laterite domain (Figure 14-1). SRK notes that some discrepancies between the logging of intrusive and extrusive forms of each rock type has resulted in some localised inconsistencies in the lithological wireframes, and recommends these intervals are relogged and refined, for use in future iterations of the lithology modelling.

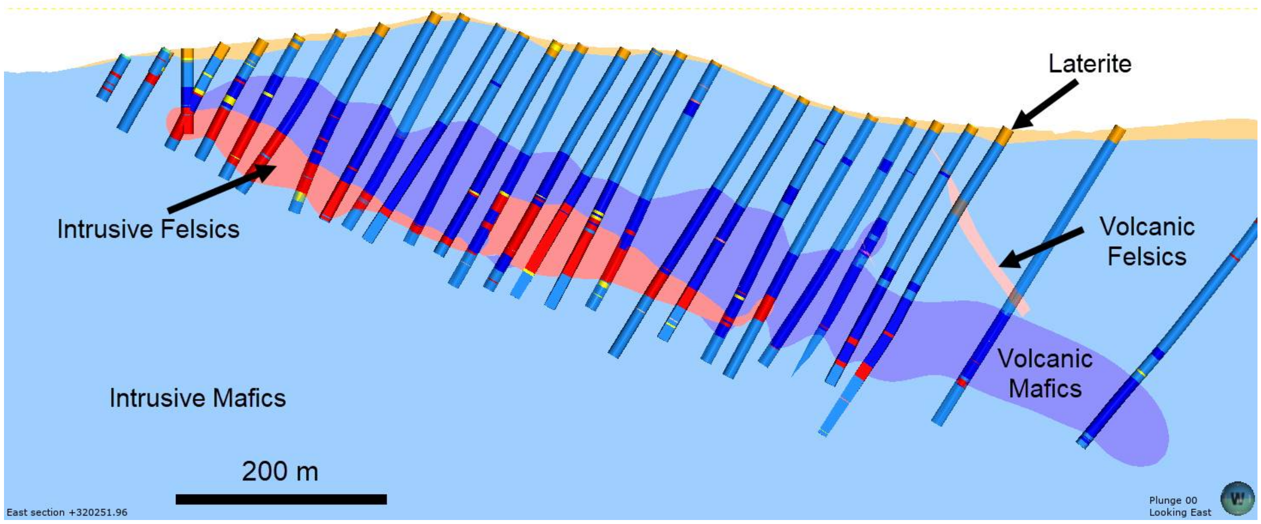


Figure 14-1: Simplified Lithological Model with the Associated Lithology Logging Cross-section (looking east)

14.3.2 Weathering Domains

Weathering surfaces were modelled on the basis of weathering logging, where the weathering profile reaches an average depth of approximately 15 to 25 m to fresh rock. Surfaces were produced for the base of the overburden/laterite, saprolite and saprock domains, with all material below the saprock footwall modelled as 'fresh' material (Figure 14-2).

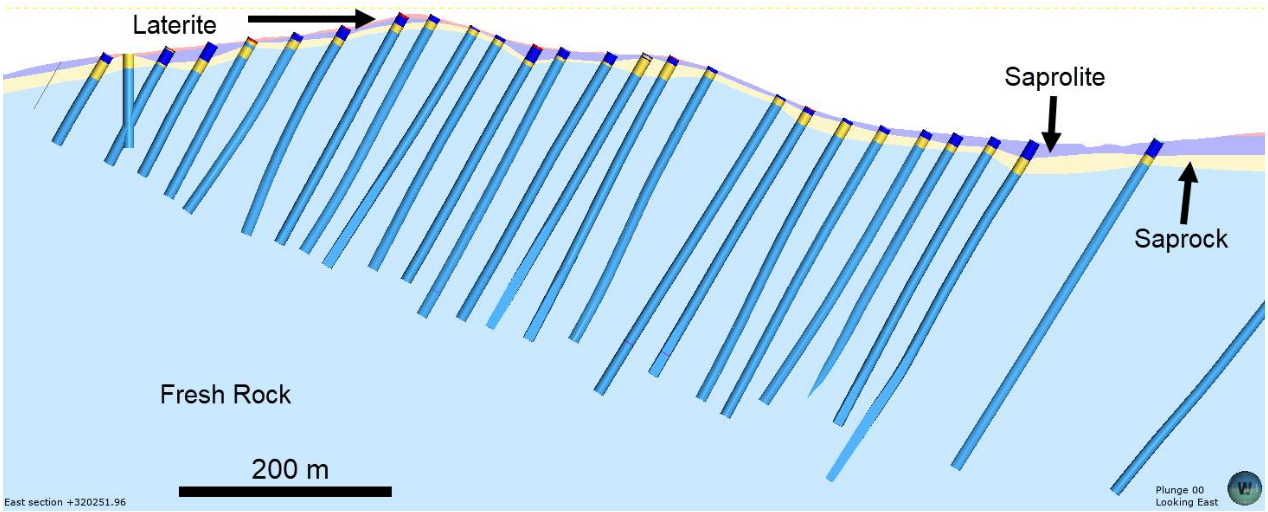


Figure 14-2: Cross-section (looking east) Showing the Four Modelled Weathering Domains with the Associated Logging

14.3.3 Mineralisation Domains

In the absence of a clear indication of an appropriate modelling cut-off from the Au grade distribution (Figure 14-3), SRK selected a modelling cut-off by assessing the extent and continuity of a series of indicator interpolant shells at different cut-off grades with respect to the assay grades of visually continuous mineralised structures.

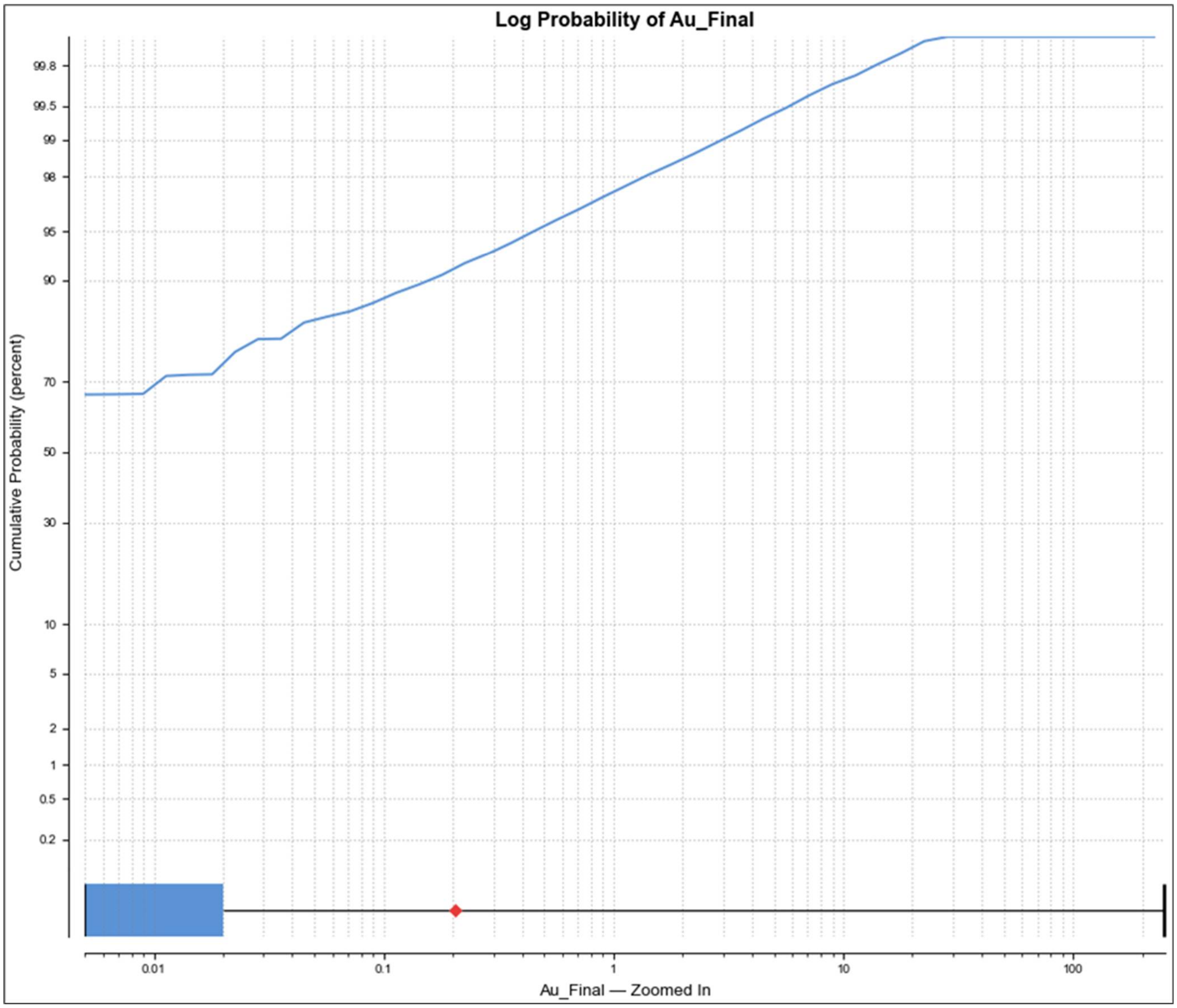


Figure 14-3: Log-probability Plot of Raw Gold Assays (Filtered to >0.005 g/t Au)

SRK selected a nominal modelling cut-off grade of 0.30 g/t Au for the modelling of Au mineralisation, using an indicator interpolant with a probability (called 'ISO value' in Leapfrog Geo software) of 0.4. Given the clear control of the lithological/rheological contacts on mineralisation, a series of surfaces were produced from the primary lithological contacts, such as the footwall of the intrusive felsic unit. These surfaces were used to produce a structural trend (Figure 14-4), where the trend and orientation of these surfaces influenced the trend and degree of continuity of the indicator interpolant volumes in each direction.

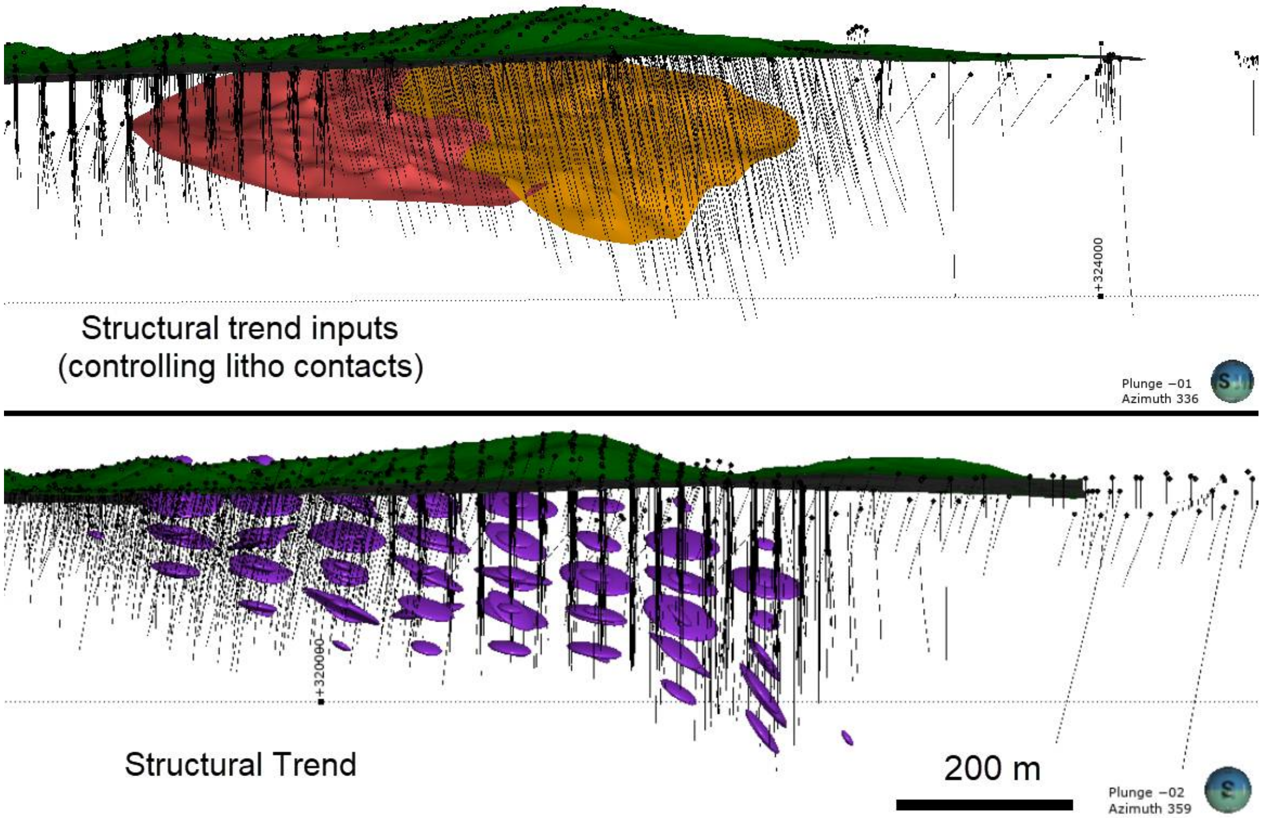


Figure 14-4: Isometric Views of the Showing the Structural Modelling Workflow Undertaken

Figure 14-4 notes:

- Upper Image = main lithological contact surfaces interpreted to be controlling mineralisation (IFEL Footwall Surface = Orange; IMAF – VMAF Contact = Red); and
- Lower Image = the resultant structural trend, represented by purple disks.

A single indicator interpolant volume was produced, including multiple manual adjustments using indicator polylines (Figure 14-5). Where mineralised structures were relatively thin, additional wireframes were produced based on sample selections in order to more accurately reflect the geometry and continuity of these structures. Vein wireframes based on sample selections were mainly utilised in Lafigué Centre, where mineralisation width and continuity is typically reduced. The final mineralisation domains used for grade and tonnage estimations are shown in Figure 14-6. The domain naming nomenclature is as follows:

- MMZ = Main Mineralisation Zone;
- WMZ1 = West Mineralisation Zone 1;
- V1-V32 = Vein domains; and,
- LAT = Laterite.

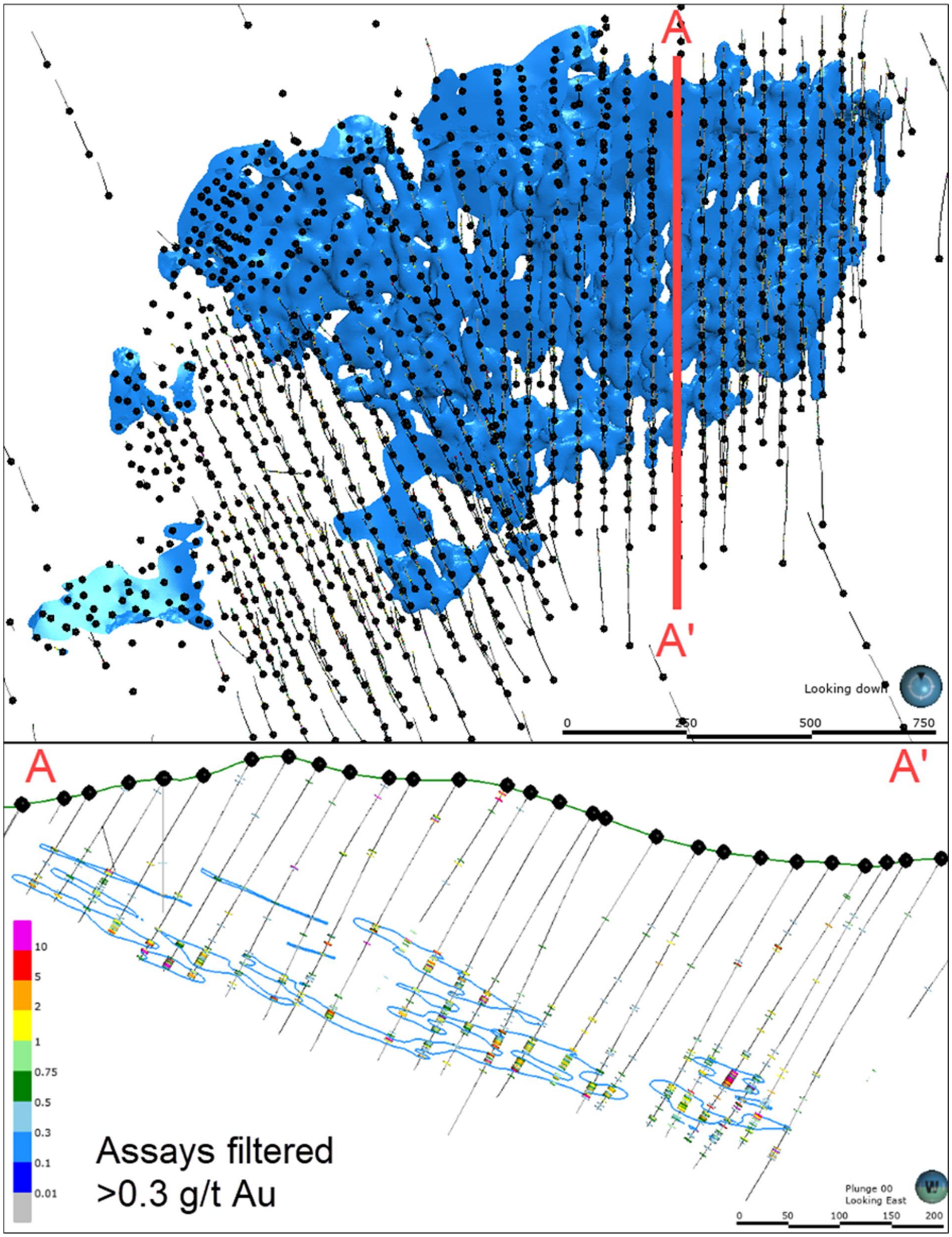


Figure 14-5: Indicator Interpolant Output Volumes Generated in Areas of Contiguous Mineralisation

Figure 14-5 notes:

- Upper Image: Plan view showing the extents of the mineralisation domains produced using an indicator interpolant.
- Lower Image: A-A' Cross-Section (looking East) showing the mineralisation domains modelled using A 0.30 g/t Au threshold, including those domains modelled as vein wireframes based on interval selections

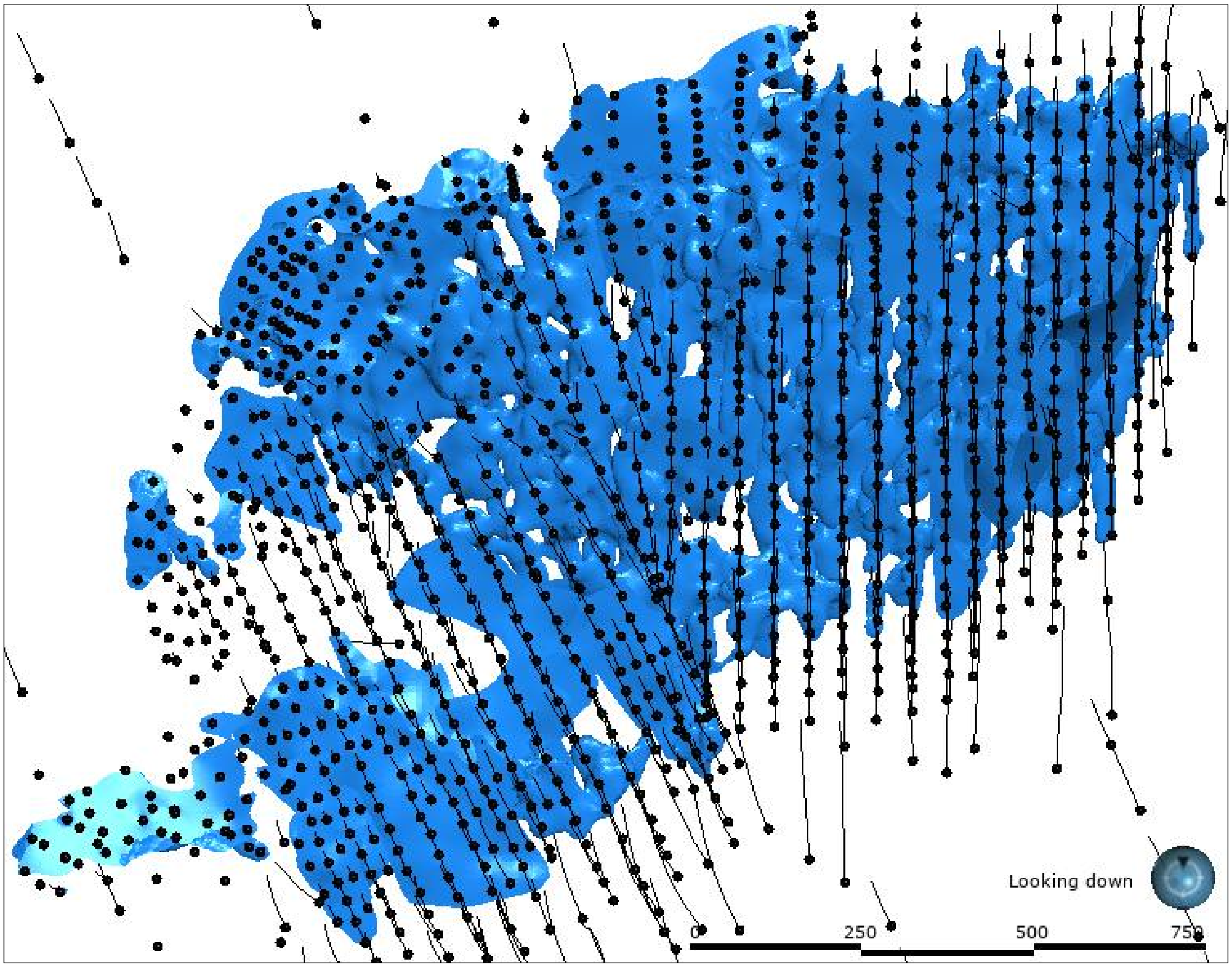


Figure 14-6: Plan View Showing all Mineralisation Domains Used for Grade and Tonnage Estimates

14.4 Post-domaining Statistical Analysis

A classical statistical study was undertaken on the domained gold assay data to assess its suitability for grade estimation. The statistics are used to confirm that appropriate estimation domains have been modelled and the statistics remain as constant (as possible) throughout the domain to allow for stationarity (constant grade distribution) to be assumed.

The average Au grades within the modelled mineralisation domains demonstrates a distinction of grade populations between the felsic and mafic host lithologies, with mafic units hosting mineralisation with a higher average grade (Figure 14-7 and Table 14-2).

Table 14-2: Summary Statistics for Raw Assay Grades within Modelled Mineralisation Domain, Split by Host Lithology

Domain	No. Samples	Min	Max	Mean	Median	Standard Deviation	CoV
LAT	1126	0.01	156.10	2.37	0.69	6.78	2.86
IFEL	6011	0.01	249.60	1.64	0.57	6.10	3.71
VFEL	212	0.01	101.80	1.62	0.53	7.79	4.80
VMAF	5651	0.01	163.60	2.77	0.58	9.75	3.51
IMAF	7292	0.01	186.50	2.77	0.64	8.79	3.17
All Felsic	6223	0.01	249.60	1.64	0.56	6.15	3.74
All Mafic	12 997	0.01	186.50	2.78	0.62	9.21	3.31

CoV = Coefficient of Variation

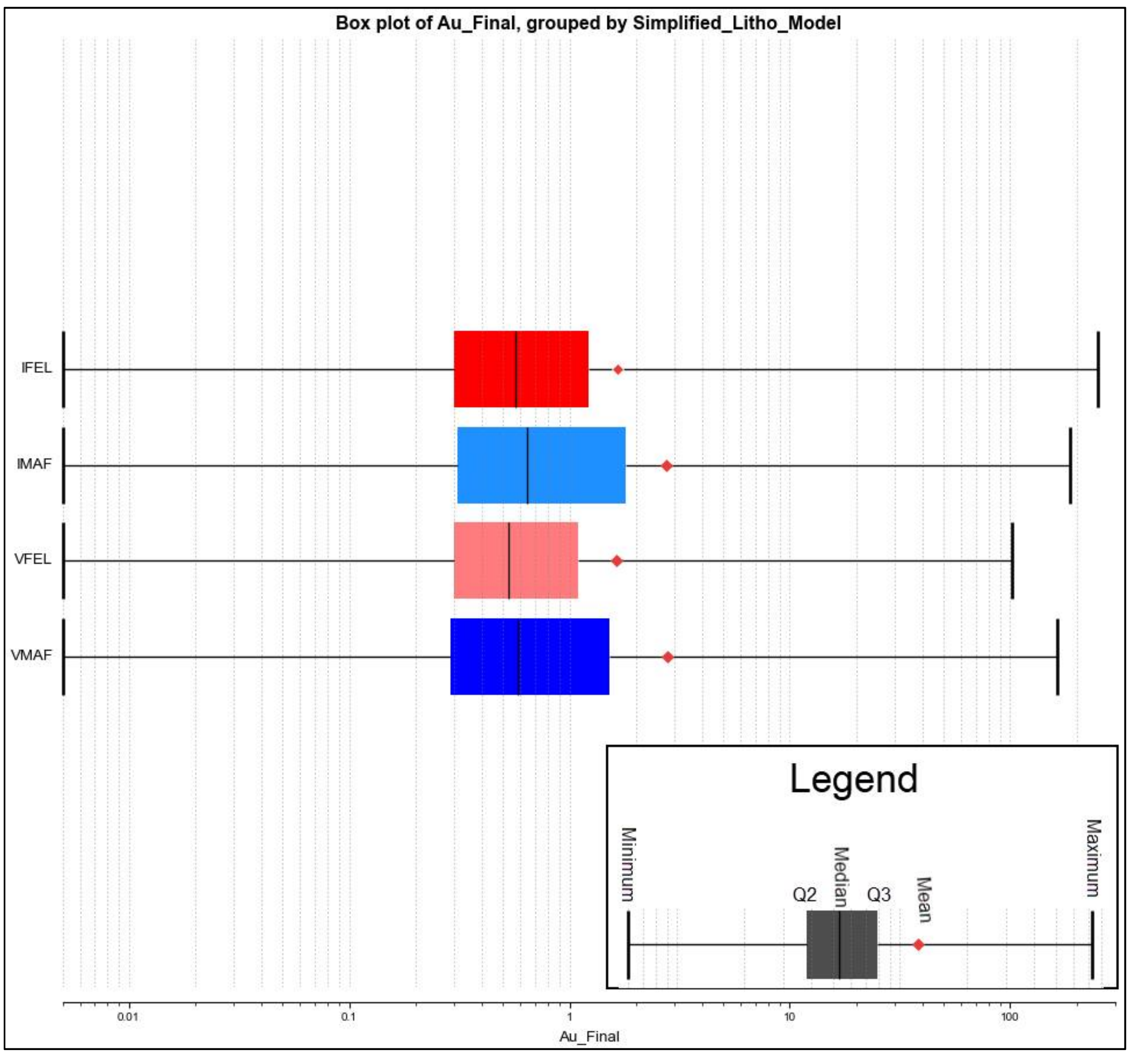


Figure 14-7: Box and Whisker Plot for Gold Grades, Split by Modelled Lithology Domains

Given the localisation of mineralisation along the lithological contacts, such as along the IFEL footwall (Figure 14-8) and at intrusive/volcanic mafic contacts (Figure 14-9), rather than the concentration of distinct mineralised structures and grade populations within each of the lithologies, SRK did not split the mineralisation/estimation domains on the basis of host lithology for grade estimation purposes.

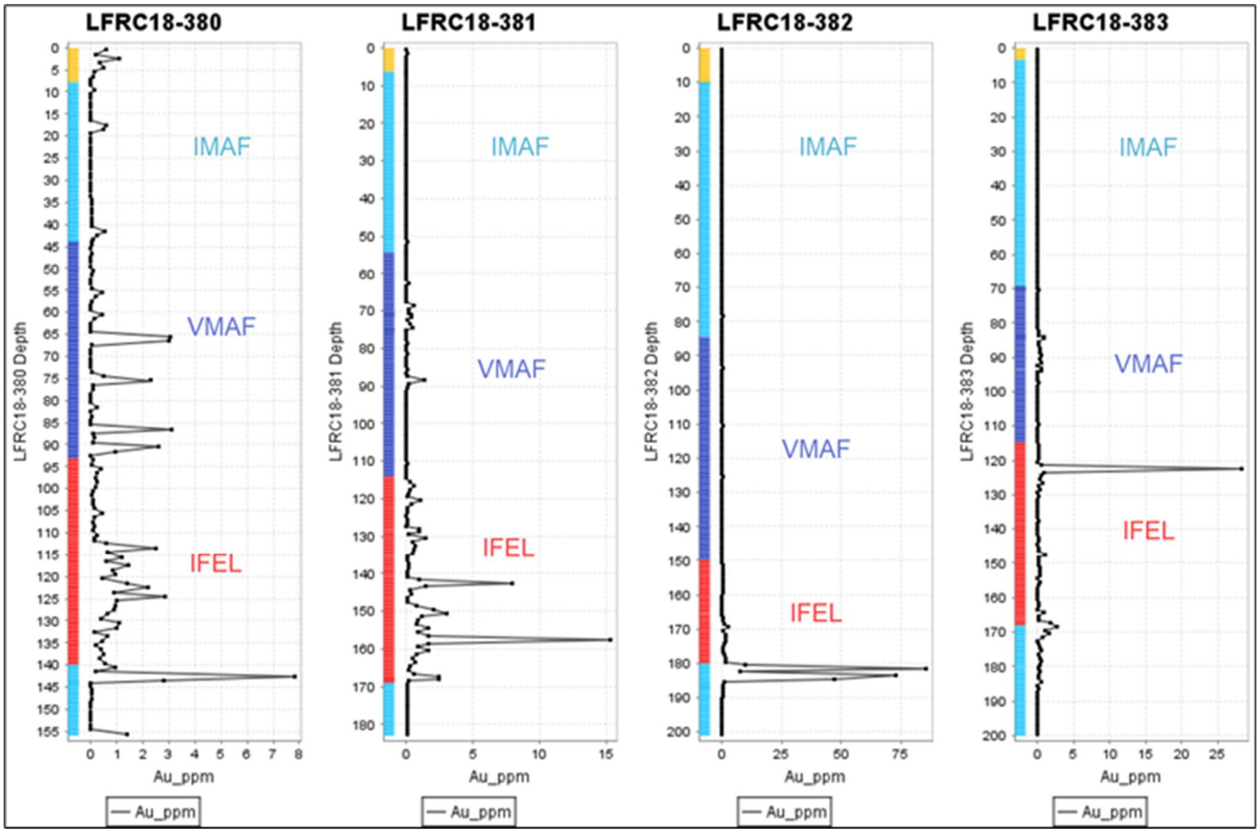


Figure 14-8: Downhole Logs Showing Logged Lithologies in a Series of Drillholes in Lafigué Centre

Figure 14-8 notes: Logged lithologies represented by coloured bars (left); Au Grades represented by black traces

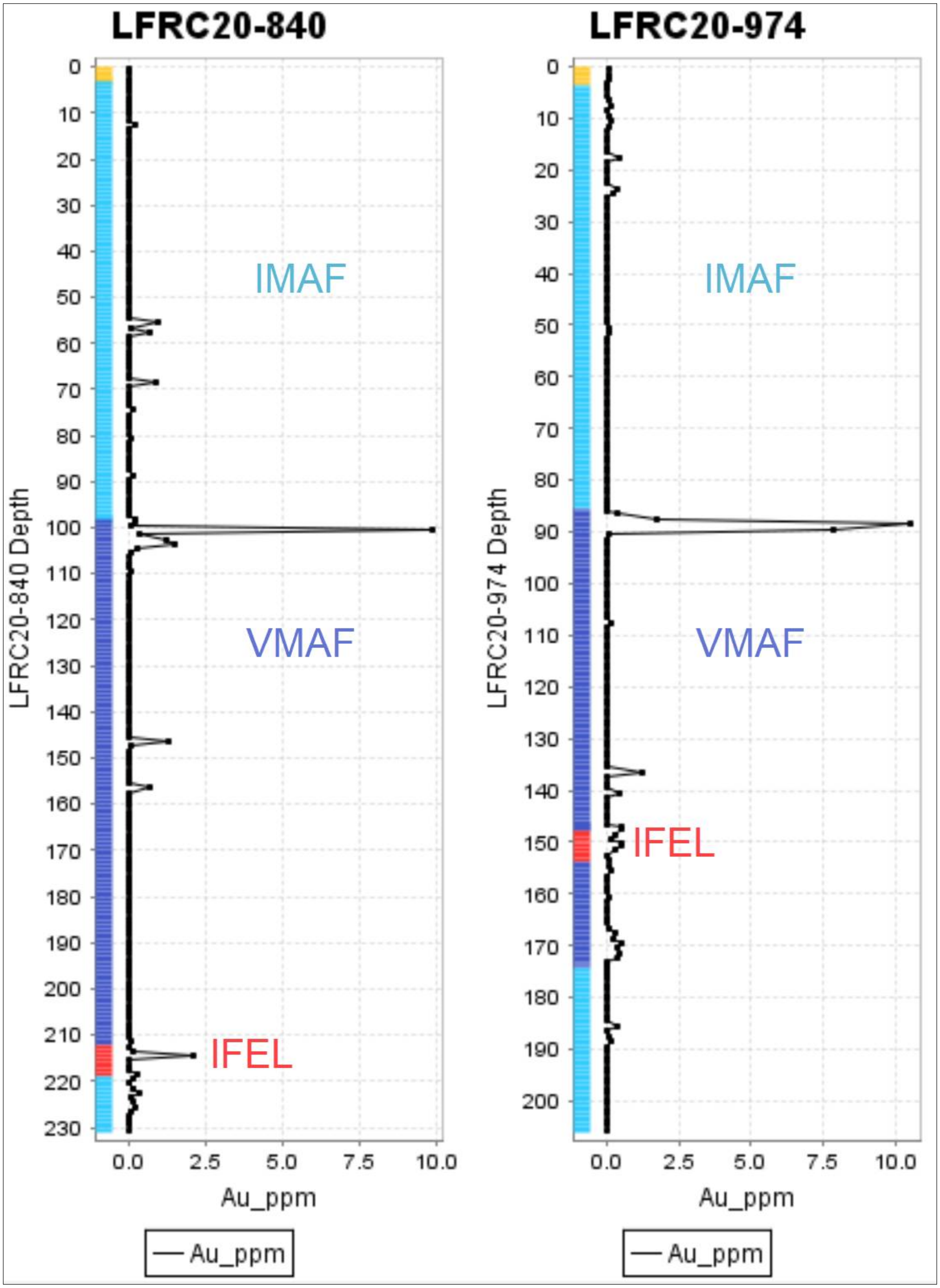


Figure 14-9: Downhole Logs Showing Logged Lithologies in Two Drillholes in Lafigué Centre/South

Figure 14-9 notes: Logged lithologies represented by coloured bars (left); Au Grades represented by black traces

Summary statistics for raw assay grades within each of the final estimation domains are presented in Table 14-3.

Table 14-3: Summary Statistics for Raw Assays, Split by Estimation Domain

Domain	No. Samples	Min (g/t Au)	Max (g/t Au)	Mean (g/t Au)	Median (g/t Au)	Standard Deviation	CoV
LAT	311	0.01	55	2.74	1.08	5.00	1.82
MMZ	17 961	0.01	250	2.28	0.59	7.89	3.45
V1	52	0.01	6	0.98	0.59	1.22	1.24
V2	30	0.01	102	4.76	0.45	19.33	4.06
V3	102	0.01	9	0.86	0.54	1.24	1.44
V7	44	0.01	9	1.06	0.39	1.90	1.80
V8	34	0.01	5	1.32	0.70	1.33	1.01
V9	23	0.01	8	1.47	0.48	2.15	1.47
V13	32	0.01	58	4.76	0.53	12.46	2.62
V16	79	0.01	52	2.05	0.42	7.11	3.48
V17	125	0.01	74	2.30	0.60	9.11	3.95
V20	111	0.01	60	2.62	0.65	8.20	3.13
V21	66	0.01	34	1.87	0.49	4.84	2.58
V22	348	0.01	156	4.63	0.97	14.37	3.10
V23	64	0.01	122	5.58	1.03	18.16	3.25
V25	120	0.01	119	3.59	0.83	12.39	3.45
V26	51	0.01	16	2.42	0.69	4.04	1.67
V27	18	0.01	34	6.87	1.31	10.94	1.59
V28	82	0.01	140	12.93	1.90	24.71	1.91
V29	86	0.01	46	3.16	0.88	6.77	2.14
V30	42	0.01	22	1.70	0.43	4.46	2.62
V31	168	0.01	34	1.72	0.69	3.15	1.83
V32	59	0.01	29	3.40	1.59	5.62	1.65
WMZ1	339	0.01	118	2.73	0.64	10.49	3.85

Table 14-3 notes: CoV = Coefficient of Variation

14.5 Compositing

Data compositing is undertaken to reduce the inherent variability that exists within the population, and to generate samples appropriate to the scale of the mining operation envisaged. It is also necessary for the estimation process that all samples are assumed to be of equal weighting and should therefore be of equal length.

Based on the sample interval length distribution (Figure 14-10), where >95% of samples are ≤1 m in length, a composite length of 1.0 m was selected for grade estimation. Using a 1.0 m compositing interval, mean Au (g/t) grades range from to 0.99 to 12.91 g/t across the 24 modelled domains. Composite statistics are summarised by estimation domain in Table 14-4.

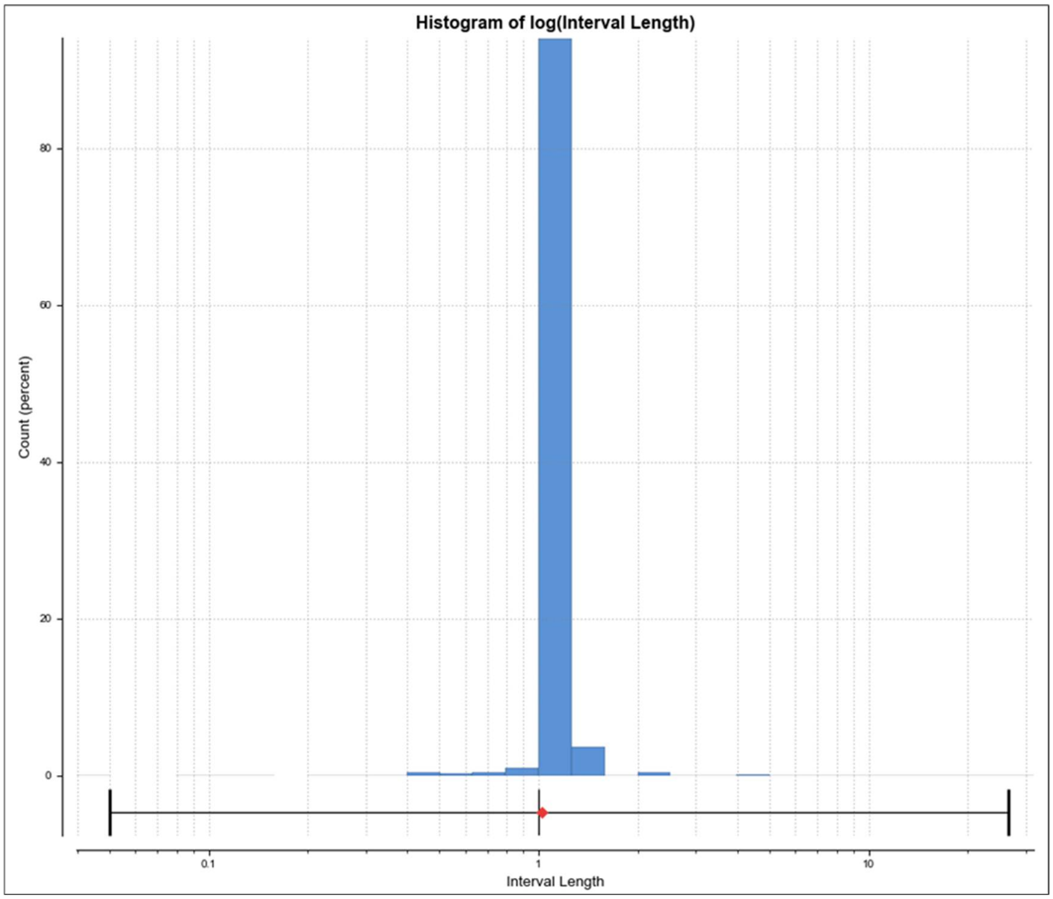


Figure 14-10: Log-histogram of Raw Sample Interval Lengths

Table 14-4: Summary Composite Statistics Split by Estimation Domain

Domain	No. Samples	Min (g/t Au)	Max (g/t Au)	Mean (g/t Au)	Median (g/t Au)	Standard Deviation	CoV
LAT	255	0.01	55.4	2.72	1.19	4.98	1.83
MMZ	15,745	0.00	249.6	2.27	0.68	6.61	2.91
V1	32	0.01	5.2	0.99	0.57	1.10	1.11
V2	28	0.01	101.8	4.70	0.39	19.21	4.09
V3	64	0.00	8.0	0.84	0.50	1.13	1.35
V7	31	0.08	8.3	1.05	0.47	1.73	1.65
V8	21	0.04	4.5	1.31	0.81	1.22	0.94
V9	16	0.11	7.7	1.46	0.48	2.13	1.46
V13	24	0.05	57.1	4.84	0.60	12.47	2.57
V16	58	0.00	49.4	1.99	0.52	6.78	3.40
V17	119	0.00	74.2	2.31	0.60	9.12	3.95
V20	81	0.05	58.7	2.62	0.68	7.93	3.03
V21	63	0.01	33.8	1.88	0.50	4.84	2.58
V22	274	0.01	151.2	4.61	1.13	13.24	2.87

Table 14-4: Summary Composite Statistics Split by Estimation Domain

Domain	No. Samples	Min (g/t Au)	Max (g/t Au)	Mean (g/t Au)	Median (g/t Au)	Standard Deviation	CoV
V23	47	0.01	119.4	5.52	1.21	17.75	3.22
V25	100	0.10	95.7	3.60	0.93	10.17	2.82
V26	37	0.01	16.1	2.54	0.76	3.78	1.49
V27	13	0.03	33.0	8.36	1.54	11.72	1.40
V28	72	0.10	91.2	12.64	3.58	19.90	1.57
V29	68	0.02	41.3	3.20	1.01	6.21	1.94
V30	33	0.12	21.1	1.76	0.43	4.23	2.40
V31	137	0.08	33.7	1.87	0.76	3.48	1.86
V32	46	0.12	20.2	3.45	1.68	4.76	1.38
WMZ1	298	0.00	113.6	2.68	0.78	9.60	3.58

CoV = Coefficient of Variation

14.6 Gold Grade Capping

The impact of isolated high-grade composites was assessed for each of the estimation domains. Caps or restricted searches can be used to reduce the impact of high grades throughout the entire domain. SRK investigated the presence of high-grade outliers by observing the grade distributions on log-histograms and log-probability plots for Au in each domain. SRK identified high-grade assays that could unduly affect the estimate based on population breaks indicated in both the log-histograms and log-probability plots. Example log-probability plots for the largest two domains are shown in Figure 14-11, with plots for all domains presented in Appendix A.

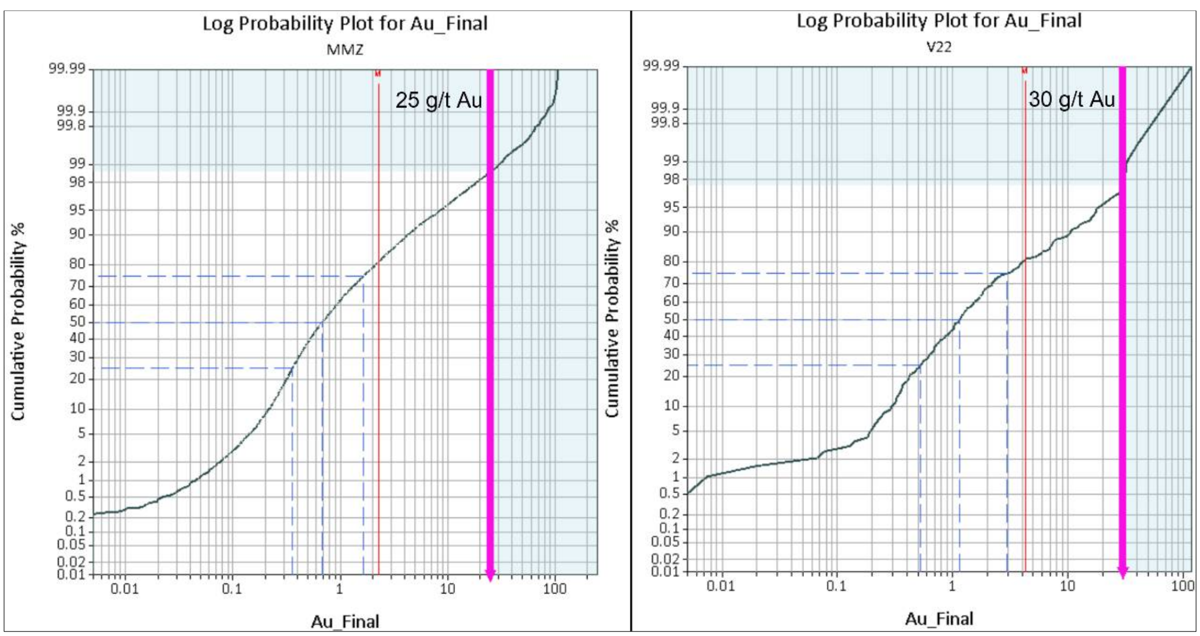


Figure 14-11: Log-probability Plot Showing Selected Capping Grades (Pink Lines) for the Main_Minzone and V22 Domains

Selected capping grades and the effects of these top cuts on the statistics of composites within each domain are shown in Table 14-5.

Table 14-5: Capping Levels and Summary Statistics for Capped Composites, Split by Estimation Domain

Domain	Cap (g/t Au)	Uncapped Mean (g/t Au)	Capped Mean (g/t Au)	% Change in Mean	Number of Samples Capped	% of Samples Capped	Standard Deviation	CoV
LAT	20	2.72	2.53	-7%	4	2%	3.61	1.43
MMZ	25	2.27	1.98	-13%	213	1%	3.96	2.00
WMZ1	20	2.68	1.90	-29%	8	3%	3.62	1.91
V1	-	0.99	0.99	0%	-	-	1.10	1.11
V2	15	4.70	1.60	-66%	1	4%	3.70	2.31
V3	-	0.84	0.84	0%	-	-	1.13	1.35
V7	-	1.05	1.05	0%	-	-	1.73	1.65
V8	-	1.31	1.31	0%	-	-	1.22	0.94
V9	-	1.46	1.46	0%	-	-	2.13	1.46
V13	20	4.86	3.00	-38%	2	8%	5.60	1.87
V16	10	2.02	1.21	-40%	2	3%	2.13	1.77
V17	10	2.31	1.23	-47%	3	3%	1.84	1.50
V20	10	2.62	1.56	-41%	3	4%	2.16	1.39
V21	10	1.88	1.36	-28%	3	5%	2.20	1.62
V22	22	4.61	3.29	-29%	16	6%	5.59	1.70
V23	16	5.52	3.06	-45%	4	9%	4.70	1.53
V25	15	3.57	2.70	-24%	2	2%	3.77	1.41
V26	10	2.37	2.18	-8%	2	5%	3.19	1.36
V27	15	6.85	5.90	-14%	3	23%	6.39	1.12
V28	27	12.91	8.59	-34%	13	18%	10.07	1.14
V29	12	3.16	2.54	-20%	6	9%	3.53	1.39
V30	10	1.63	1.39	-15%	2	6%	2.33	1.79
V31	15	1.82	1.69	-7%	1	1%	2.36	1.37
V32	20	3.42	3.25	-7%	1	2%	4.74	1.38

CoV = Coefficient of Variation

14.7 Boundary Analysis

In order to ascertain whether 'hard' or 'soft' boundaries between domains should be utilised during grade interpolation, a boundary analysis was undertaken. The process involves a statistical analysis of samples close to each domain (wireframe) boundary.

The Au grades decrease sharply across the boundary between each of the primary mineralisation domains, at spacings much less than the average drill spacing (Figure 14-12), which supports the differentiation of these zones during modelling and the implementation of hard boundary conditions during interpolation of Au grade into the block model.

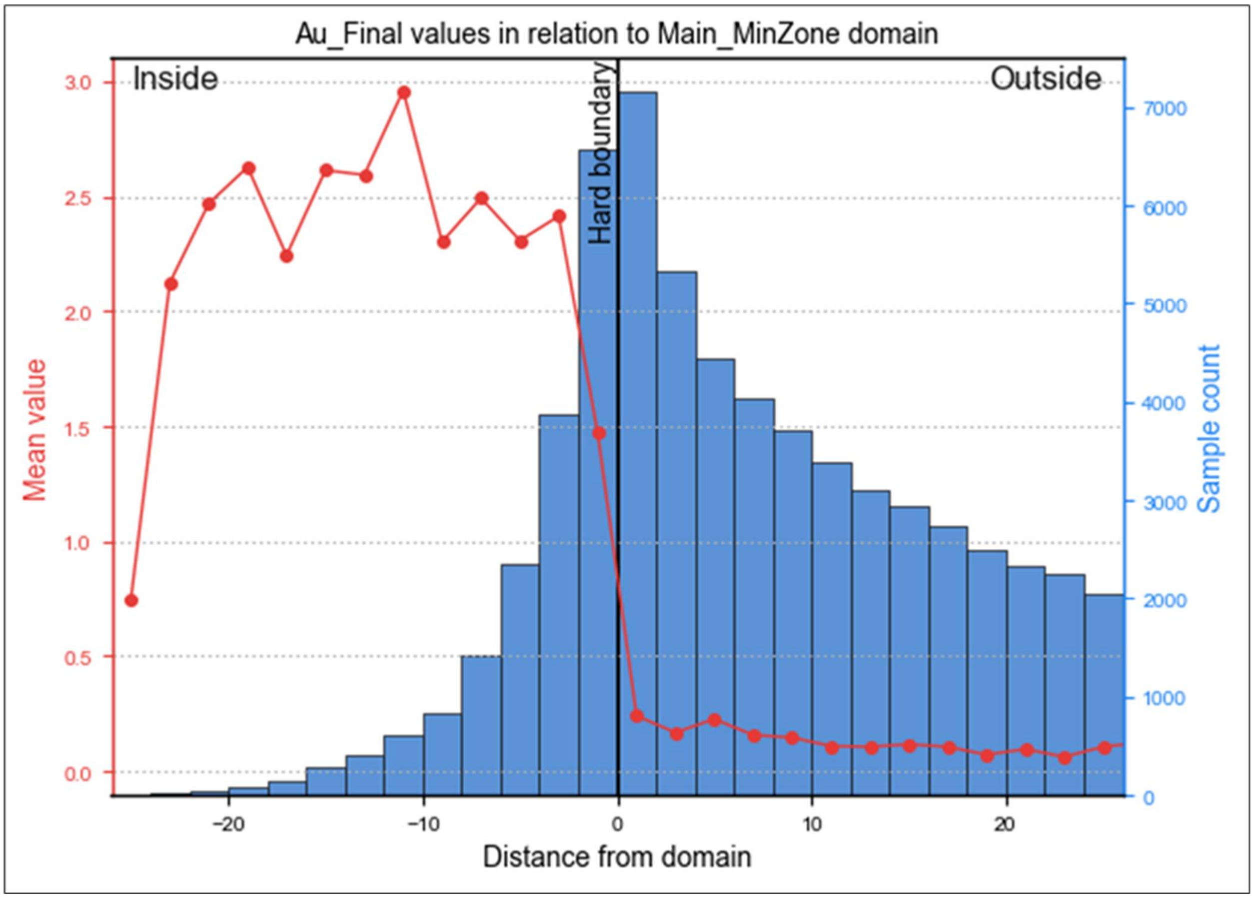


Figure 14-12: Domain Boundary Analysis for Au (g/t) in the MMZ

Where a boundary analysis was undertaken for the contact between the laterite and underlying primary mineralisation domains, no statistically significant distinction in average grades was apparent across the contact (Figure 14-13), and so soft boundaries with a range of 3 m were used.

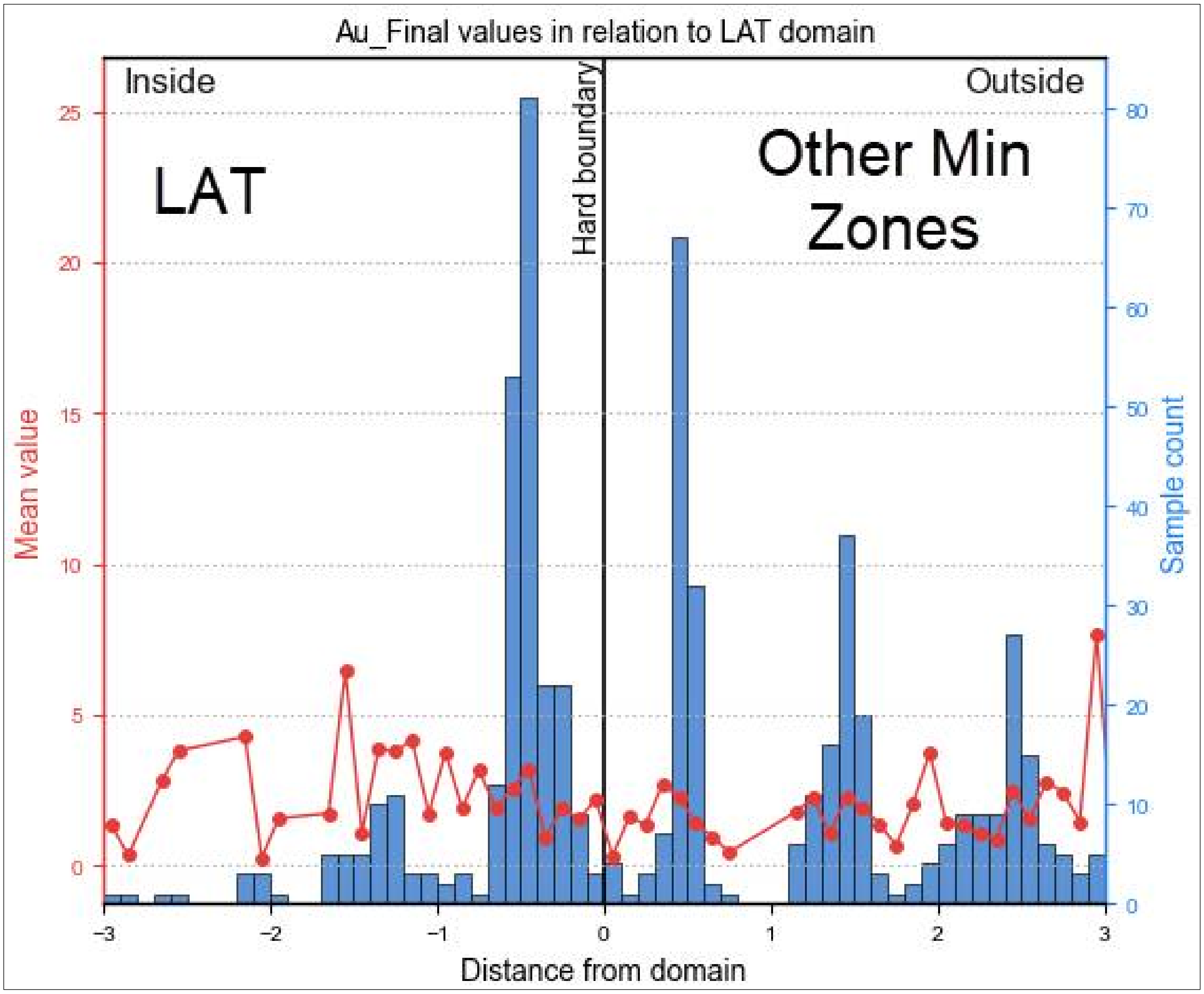


Figure 14-13: Contact Plot for Au (g/t) between the Laterite and All Other (Primary) Mineralisation Domains

14.8 Geostatistical Analysis

A geostatistical analysis (variography) of the composited Au assay grades was undertaken for each of the main estimation domains. The purpose of the study was to examine the 3D variability and spatial relationships between composite samples, and to derive appropriate variogram models to be used in block grade interpolation. Each domain was analysed separately, first using a variogram map to understand the principal directions of grade anisotropy and choose the major, semi-major and minor directions for analysis. After the directions were chosen, a down-hole variogram was generated to understand the grade variability at short-scales and define the nugget effect. Variograms for three directions were then modelled (using common sill values) to the variance of the data. An example of the directional variograms generated is shown in Figure 14-14.

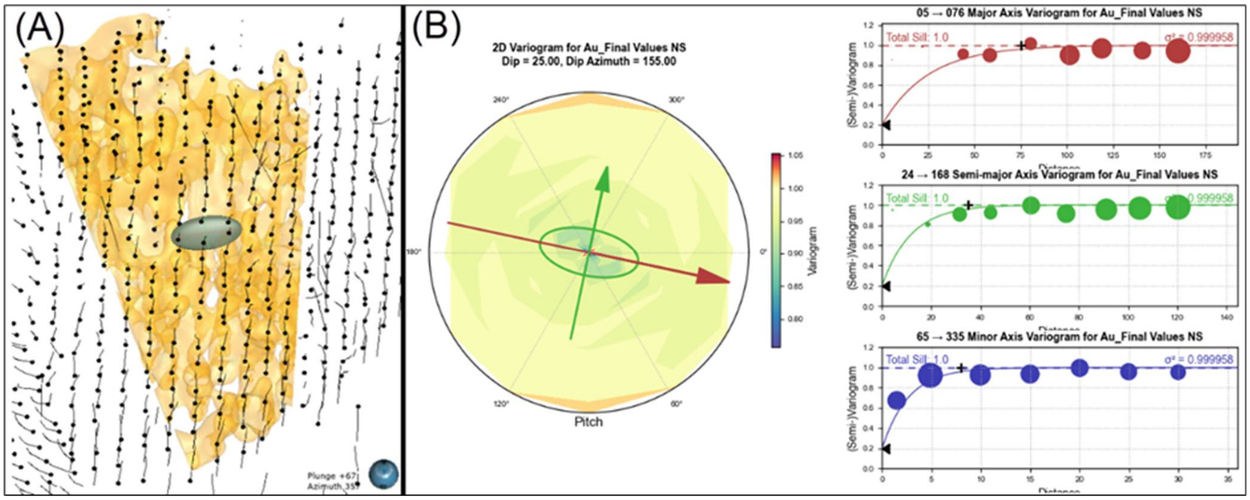


Figure 14-14: Overview of Modelled Variograms for the MMZ (Central) Domain

Figure 14-14 notes:

- (A) MMZ (central) domain represented by orange solid, drillholes by black traces and modelled variogram ranges displayed as an ellipsoid showing major, semi-major and minor axes directions;
- (B) Variogram map and normal scores transformed variograms for the major, semi-major and minor axes for Au (g/t) in the same domain

Given the high degree of litho-structural control on mineralisation in the central and eastern parts of the Lafigué deposit, the MMZ was sub-domained on the basis of three dominant structural trends (Figure 14-15). Variography and subsequent estimation was completed for each of the structural sub-domains, with full variogram parameters summarised in Table 14-6. The geostatistical analysis has produced adequate variograms to allow for Ordinary Kriging ('OK') to be utilised for grade interpolation. In smaller, less well-informed domains where there were significantly fewer samples (typically <50 samples), adequate quality variograms could not be produced and an inverse-distance-weighted estimation approach was adopted.

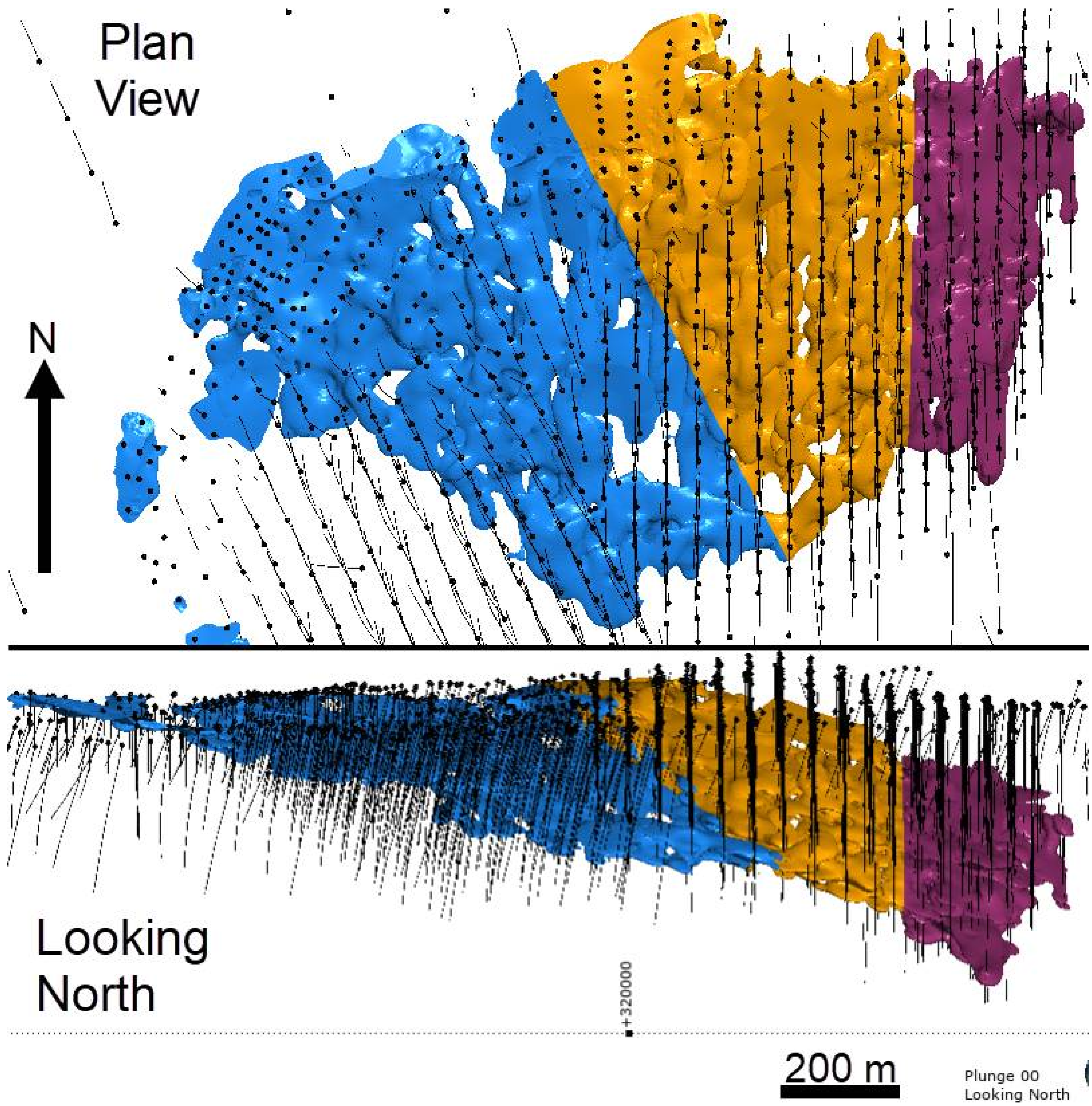


Figure 14-15: Plan and Isometric Views of the MMZ Structural Sub-Domains Used for Variography and Grade Estimation

Table 14-6: Variogram Parameters Used for Estimation in the MMZ, V17 and V22 Domains

Variogram Name	Direction							Structure 1					Structure 2				
	Dip	Dip Azimuth	Pitch	Model space	Variance	NE	Norm. Nugget	Sill	Norm. sill	Major	Semi-major	Minor	Sill	Norm. sill	Major	Semi-major	Minor
MMZ Central	25	155	12	Data	45.4	20.7	0.46	25.51	0.56	75	35	8					
MMZ Central	25	155	12	Normal score	1.0	0.25		0.75		75	35	8					
MMZ East	30	140	17	Data	67.5	29.7	0.44	29.18	0.43	58	29	2	8.7	0.13	75	40	12
MMZ East	30	140	17	Normal score	1.0	0.25		0.33		58	29	2	0.40		75	40	12
MMZ West	20	150	0	Data	33.0	12.5	0.38	21.11	0.64	45	20	6					
MMZ West	20	150	0	Normal score	1.0	0.20		0.80		45	20	6					
WMZ1	20	143	39	Normal score	1.0	0.25		0.75		60	45	5					
WMZ1	20	143	39	Data	92.1	45.7	0.50	46.44	0.50	60	45	5					
V17	20	150	13	Normal score	1.0	0.40		0.65		55	55	6					
V17	20	150	13	Data	81.3	53.0	0.65	29.13	0.36	55	55	6					
V22	20	145	173	Normal score	1.0	0.30		0.70		55	45	3					
V22	20	145	173	Data	175.3	88.1	0.50	87.13	0.50	55	45	3					
V25	25	140	75	Normal score	1.0	0.30		0.75		70	50	3					
V25	25	140	75	Data	75.0	33.0	0.44	41.98	0.56	70	50	3					
V28	10	160	10	Normal score	1.0	0.30		0.70		50	50	6					
V28	10	160	10	Data	492.9	196.6	0.40	296.43	0.60	50	50	6					
V31	20.41	144	55	Normal score	1.0	0.40		0.65		38	35	2					
V31	20.41	144	55	Data	11.3	6.1	0.54	5.23	0.46	38	35	2					

NE = Nugget Effect

14.9 Block Modelling and Grade Estimation

14.9.1 Block Model Definition

The block model covered an area encompassing all modelled mineralised zones. The geometry and extents of the block model are summarised in Table 14-7. Parent block dimensions are 20 x 20 x 10 m and are sub-blocked to 2.5 x 2.5 x 1.25 m. No rotation was applied to the block model.

Table 14-7: Lafigué Block Model Dimensions

Dimension	Origin	Block Size (m)	Number of Blocks	Minimum Sub-blocking (m)
X	318950	20	106	2.50
Y	913150	20	83	2.50
Z	-180	10	65	1.25

14.9.2 Grade Interpolation

Search ellipsoid parameters were tailored to consider the number of drillholes to be used, based on the average drillhole spacing, with the search orientation aligned with the model variograms obtained for each domain. In the MMZ, dynamic anisotropy was utilised due to account for the variable orientation of each structural sub-zone (Figure 14-16). The variable orientation was informed by surfaces representing the primary mineralisation-controlling structures/lithology contacts as described in Section 14.3.3. Individual domains were estimated separately using hard boundaries (with the exception of the Laterite domain and MMZ sub-domains) in order to prevent drillhole data from one domain affecting block grades in a neighbouring domain.

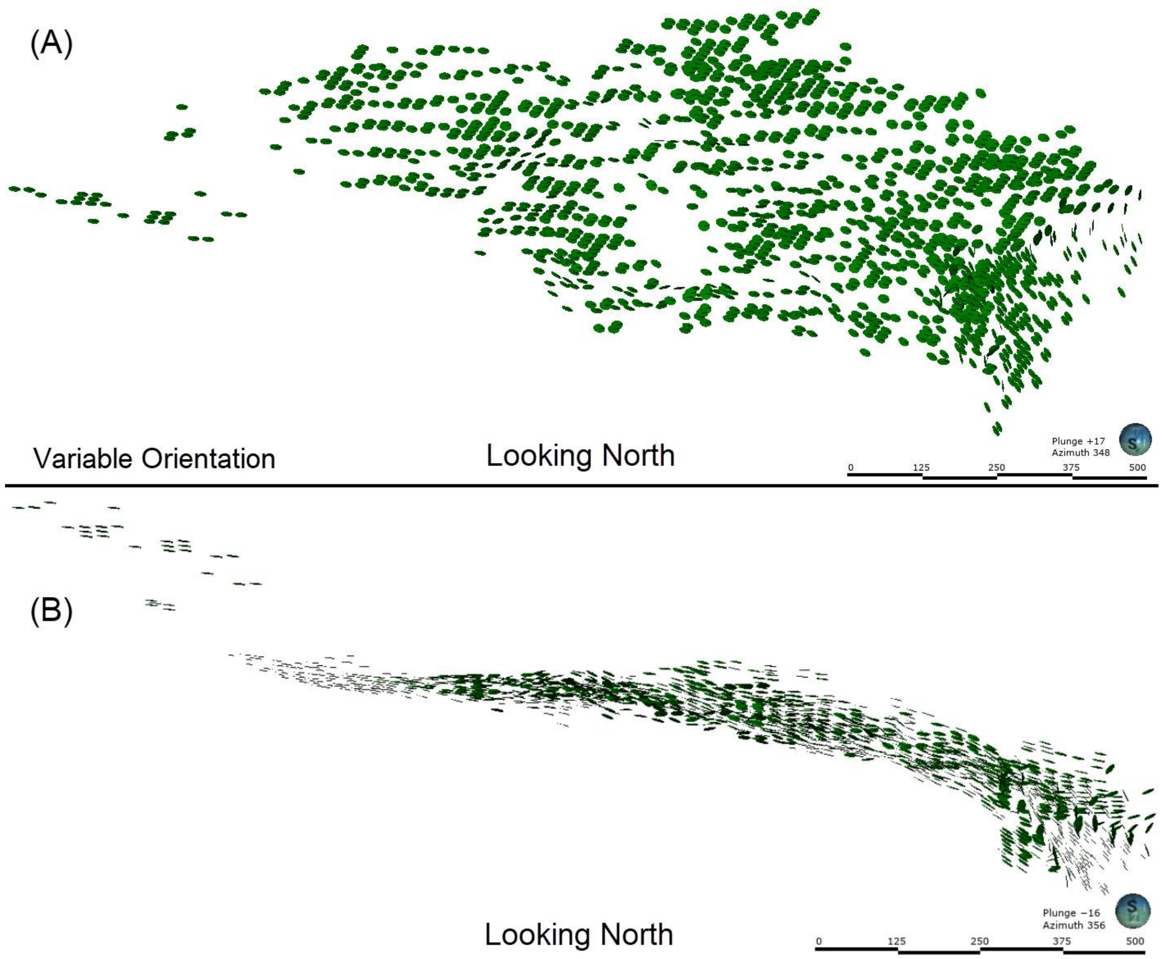


Figure 14-16: Isometric Views Showing the Variable Orientation (Represented by Green Disks) of the MMZ

14.9.3 Neighbourhood Analysis

Kriging neighbourhood analysis ('KNA') was undertaken in order to optimise the block size, sample selection criteria and discretisation used during grade interpolation. The initial KNA process was based on comparisons of kriging efficiency ('KE') and slope of regression ('SoR'), when varying each of the above parameters independently. The KE estimates the degree of correspondence between the estimated block histogram and that of the true block grades, where a KE of 100% would represent a perfect match between the two (Coombes, 2008).

The SoR is a measure of conditional bias. That is, the tendency for higher grades to be under-estimated and lower grades to be over-estimated, where the slope of regression equation compares the estimated and theoretical true block grades (Coombes, 2008). A 1:1 relationship between theoretical true and estimated block grades would produce a slope of 1, meaning that the estimated high grades and estimated low grades correspond accurately to the respective theoretical true high and low grades. The flatter the slope (and therefore over-estimation of low grades and under-estimation of high grades), the lower the SoR. Figure 14-17 shows example plots for the MMZ (Central domain), where KE and SoR are plotted as a function of selected block sizes and min/max samples selected. Overall, the KNA undertaken showed that the estimates were relatively insensitive to changing parent block size or min/max sample selection criteria (within reasonable ranges).

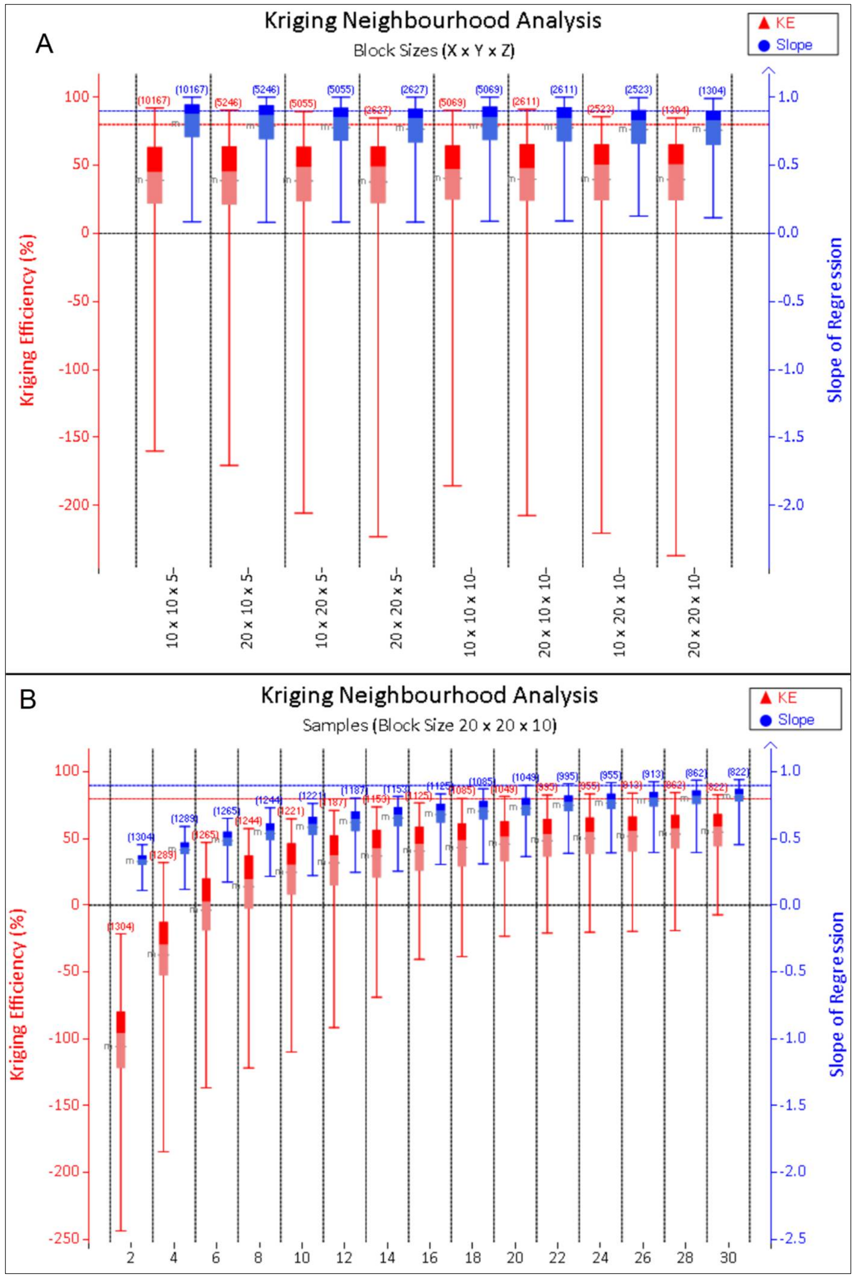


Figure 14-17: Kriging Neighbourhood Analysis Box and Whisker Plots

Figure 14-17 illustrates a series of box and whisker plots illustrating Kriging Efficiency (KE) and Slope of Regression (SoR) as a function of changing:

- (A) Parent block sizes, at increments of 5 m between (10 X 10 X 5) m and (20 X 20 X 10) m; and,
- (B) Number of samples used to inform block estimates between a minimum of two and a maximum of 30 Samples.

Additional sensitivity analyses were undertaken assessing the influence of changes to search ellipsoid dimensions by running a series of estimation runs and comparing a range of search ellipsoid dimensions appropriate to the drill spacing and variogram ranges. Defaults were selected for all other search parameters and remained constant for each sensitivity run, with only the search ellipsoid dimensions adjusted.

A discretisation level of 4 x 4 x 2 was set for all estimates. The final grade interpolation parameters used for each domain are detailed in Table 14-8.

Table 14-8: Summary of Lafigué Estimation Parameters

Domain	Estimation Method	Ellipsoid Directions			Ellipsoid Ranges			Number of Samples		Drillhole Limit	Sector Search		
		Dip	Dip Az.	Pitch	Maximum	Intermediate	Minimum	Minimum	Maximum	Max Samples per Hole	Method	Max Samples	Max Empty Sectors
MMZ_Central - Pass 1	OK	Variable Orientation			75	35	6	7	12	3	Quadrant	3	1
MMZ_Central - Pass 2	OK	Variable Orientation			100	45	10	7	12	3	Quadrant	3	1
MMZ_Central - Pass 3	OK	Variable Orientation			200	100	25	4	12	3	None	-	-
MMZ_East - Pass 1	OK	32	125	135	60	45	8	6	12	3	Quadrant	2	1
MMZ_East - Pass 2	OK	32	125	135	80	60	10	6	12	3	Quadrant	2	1
MMZ_East - Pass 3	OK	32	125	135	200	100	30	4	12	3	None	-	-
MMZ_West - Pass 1	OK	Variable Orientation			60	35	8	7	12	3	Quadrant	3	1
MMZ_West - Pass 2	OK	Variable Orientation			75	45	12	7	12	3	Quadrant	3	1
MMZ_West - Pass 3	OK	Variable Orientation			200	100	40	4	12	3	None	-	-
LAT - Pass 1	IDW^2	0	50	90	60	40	4	7	12	3	None		
LAT - Pass 2	IDW^2	0	50	90	75	50	6	5	12	3	None		
LAT - Pass 3	IDW^2	0	50	90	150	150	20	3	5	-	None		
V1 - Pass 1	IDW^2	25	165	0	60	60	60	8	20	7	None		
V1 - Pass 2	IDW^2	25	165	0	150	150	100	6	20	5	None		
V2 - Pass 1	IDW^2	25	175	0	60	60	60	7	20	6	None		
V2 - Pass 2	IDW^2	25	175	0	150	150	150	9	20	8	None		
V3 - Pass 1	IDW^2	55	135	150	60	60	60	9	20	8	None		
V3 - Pass 2	IDW^2	55	135	150	130	130	130	8	20	7	None		
V7 - Pass 1	IDW^2	30	190	5	50	50	20	7	14	6	None		
V7 - Pass 2	IDW^2	30	190	5	150	150	50	4	7	3	None		
V8 - Pass 1	IDW^2	25	175	10	45	45	20	6	10	5	None		

Table 14-8: Summary of Lafigué Estimation Parameters

Domain	Estimation Method	Ellipsoid Directions			Ellipsoid Ranges			Number of Samples		Drillhole Limit	Sector Search		
		Dip	Dip Az.	Pitch	Maximum	Intermediate	Minimum	Minimum	Maximum	Max Samples per Hole	Method	Max Samples	Max Empty Sectors
V8 - Pass 2	IDW²	25	175	10	100	100	50	4	7	3	None		
V9 - Pass 1	IDW²	25	175	20	60	60	60	11	20	8	None		
V9 - Pass 2	IDW²	25	175	20	150	150	150	4	7	3	None		
V13 - Pass 1	IDW²	25	180	10	60	60	60	9	14	8	None		
V13 - Pass 2	IDW²	25	180	10	75	75	50	4	7	3	None		
V16 - Pass 1	IDW²	20	145	0	40	40	25	9	20	8	None		
V16 - Pass 2	IDW²	20	145	0	100	100	50	7	12	6	None		
V17 - Pass 1	OK	20	150	13	55	50	6	8	14	-	None		
V17 - Pass 2	OK	15	140	20	80	70	20	8	14	-	Quadrant	5	1
V17 - Pass 3	OK	15	140	20	200	150	30	3	12	-	Quadrant	5	1
V20 - Pass 1	IDW²	25	100	15	40	40	15	7	12	6	None		
V20 - Pass 2	IDW²	25	100	15	75	75	50	4	7	3	None		
V21 - Pass 1	IDW²	25	100	15	40	40	20	9	20	8	None		
V21 - Pass 2	IDW²	25	100	15	100	100	60	7	14	6	None		
V22 - Pass 1	OK	20	145	170	55	45	6	7	12	-	None		
V22 - Pass 2	OK	20	145	170	80	70	10	7	12	-	Quadrant	4	1
V22 - Pass 3	OK	20	145	170	150	150	25	3	14	-	Quadrant	4	1
V23 - Pass 1	IDW²	23	140	25	75	75	75	8	20	7	None		
V23 - Pass 2	IDW²	23	140	25	125	125	125	8	20	7	None		
V25 - Pass 1	OK	25	140	75	45	40	10	6	10	3	None		
V25 - Pass 2	OK	25	140	75	80	70	20	6	10	3	Quadrant	4	1

Table 14-8: Summary of Lafigué Estimation Parameters

Domain	Estimation Method	Ellipsoid Directions			Ellipsoid Ranges			Number of Samples		Drillhole Limit	Sector Search		
		Dip	Dip Az.	Pitch	Maximum	Intermediate	Minimum	Minimum	Maximum	Max Samples per Hole	Method	Max Samples	Max Empty Sectors
V25 - Pass 3	OK	25	140	75	150	150	25	3	12	-	Quadrant	4	1
V26 - Pass 1	IDW²	25	180	10	60	60	60	9	14	8	None		
V26 - Pass 2	IDW²	25	180	10	120	120	120	9	14	8	None		
V27 - Pass 1	IDW²	25	180	10	55	55	20	7	12	6	None		
V27 - Pass 2	IDW²	25	180	10	100	100	50	4	7	3	None		
V28 - Pass 1	OK	10	160	10	60	60	10	7	12	-	None		
V28 - Pass 2	OK	10	160	10	75	75	12	7	12	-	Quadrant	4	2
V28 - Pass 3	OK	10	160	10	150	150	25	3	12	2	Quadrant	4	2
V29 - Pass 1	IDW²	15	80	165	40	40	20	9	14	8	None		
V29 - Pass 2	IDW²	15	80	165	100	100	50	7	12	6	None		
V30 - Pass 1	IDW²	15	140	140	40	40	20	7	12	6	None		
V30 - Pass 2	IDW²	15	140	140	100	100	50	4	7	3	None		
V31 - Pass 1	OK	20	145	55	40	35	8	7	12	-	None		
V31 - Pass 2	OK	20	145	55	75	70	12	7	12	-	Quadrant	4	1
V31 - Pass 3	OK	20	145	55	100	100	25	3	4	-	Quadrant	4	1
V32 - Pass 1	IDW²	25	145	20	40	40	20	7	12	6	None		
V32 - Pass 2	IDW²	25	145	20	100	100	50	4	7	3	None		
WMZ1 - Pass 1	OK	20	140	40	50	40	8	9	12	4	None		
WMZ1 - Pass 2	OK	20	140	40	100	80	12	9	15	4	Quadrant	4	1
WMZ1 - Pass 3	OK	20	140	40	200	150	25	3	10	-	Quadrant	4	1

14.10 Tonnage Estimation

The density database provided includes a total of 2214 samples with logged lithology and weathering attributes and located within the area covered by the resource model. The samples are distributed across the Project area, with a slight spatial bias towards the centre and east of the deposit (Figure 14-18). Each lithology is represented in the density database; however, the laterite material is only represented by a single density measurement from a sample outside of the extents of mineralisation modelled by SRK. Although this represents a risk in terms of the representivity and accuracy of the density value applied to lateritic material, SRK considers the risk to be minimised by the limited remaining tonnage within this domain (see Section 14.12). SRK has coded the block model with average density values, split by lithology/material type. These values are listed in Table 14-9. SRK notes that a number of density samples (e.g., from hydrological drillholes) do not have accompanying lithology and weathering logging. With reduced certainty of what material type these samples represent, SRK excluded these samples from the statistics presented in Table 14-9.

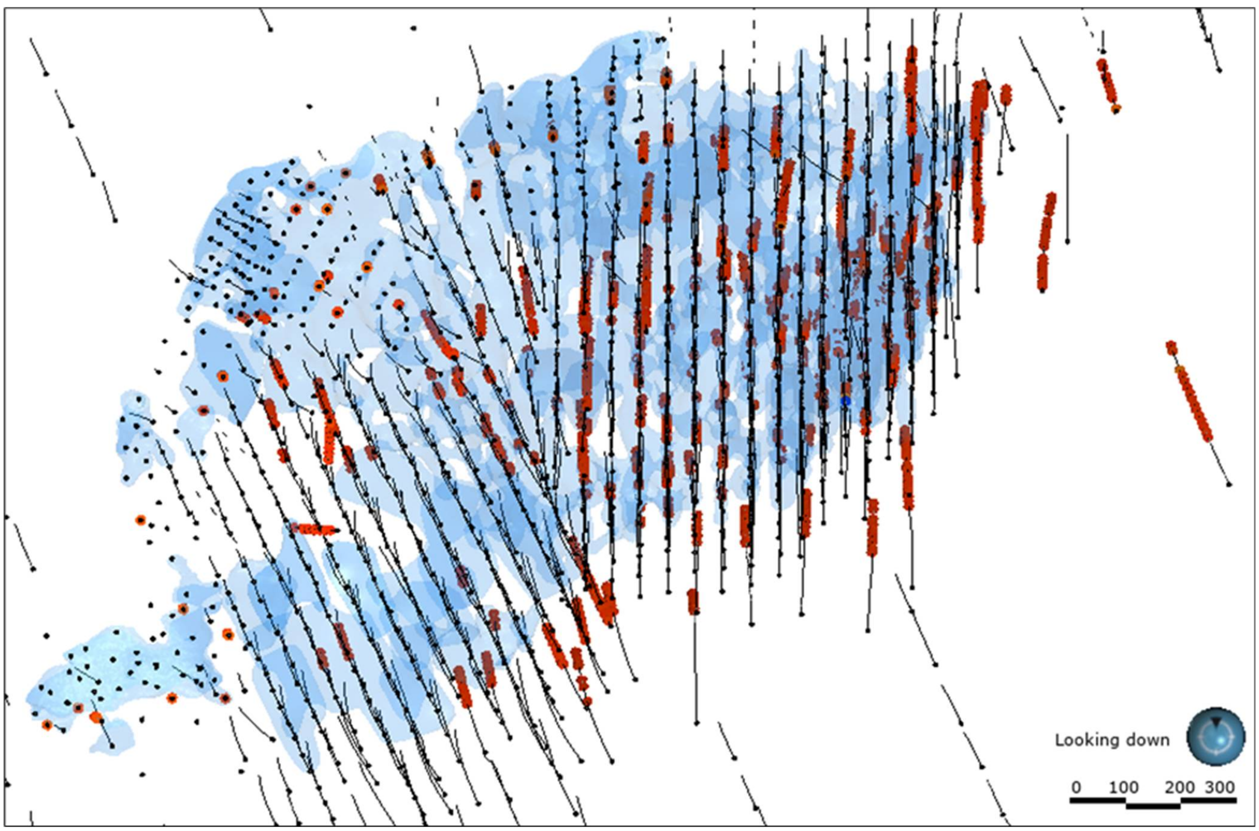


Figure 14-18: Plan View Showing the Spatial Distribution of Density Measurements (Red Disks)

Table 14-9: Average Density Values, Split by Lithology

Lithology	Material Type	Number of Measurements	Mean Value (g/cm^3)	Notes
Laterite	Oxide	1	2.00	Single measurement outside of modelled mineralisation extents
Saprolite	Oxide	9	1.66	Excludes one anomalous measurement (2.6)
Saprock	Transition	17	2.51	Excludes one anomalous measurement (3.1)
Fresh - Mafic	Fresh	1205	2.86	
Fresh - Felsic	Fresh	628	2.72	

14.11 Model Validation

14.11.1 Overview

SRK validated the block model through the following checks:

- local validation using visual inspections on sections and plans, viewing composites versus block estimates;
- global validation by comparison of de-clustered composite statistics versus block estimates; and
- local validation by comparison of average assay grades with average block estimates along different directions, through the generation of swath plots.

SRK considers that the block model reflects the current understanding of the distribution of mineralisation and is an acceptable basis for a Mineral Resource statement.

14.11.2 Visual Validation

Visual validation provides a comparison of the interpolated block model on a local scale. A thorough visual inspection has been undertaken in 3D, demonstrating a good degree of correspondence between the block estimates and nearby composites (Figure 14-19 and Figure 14-20).

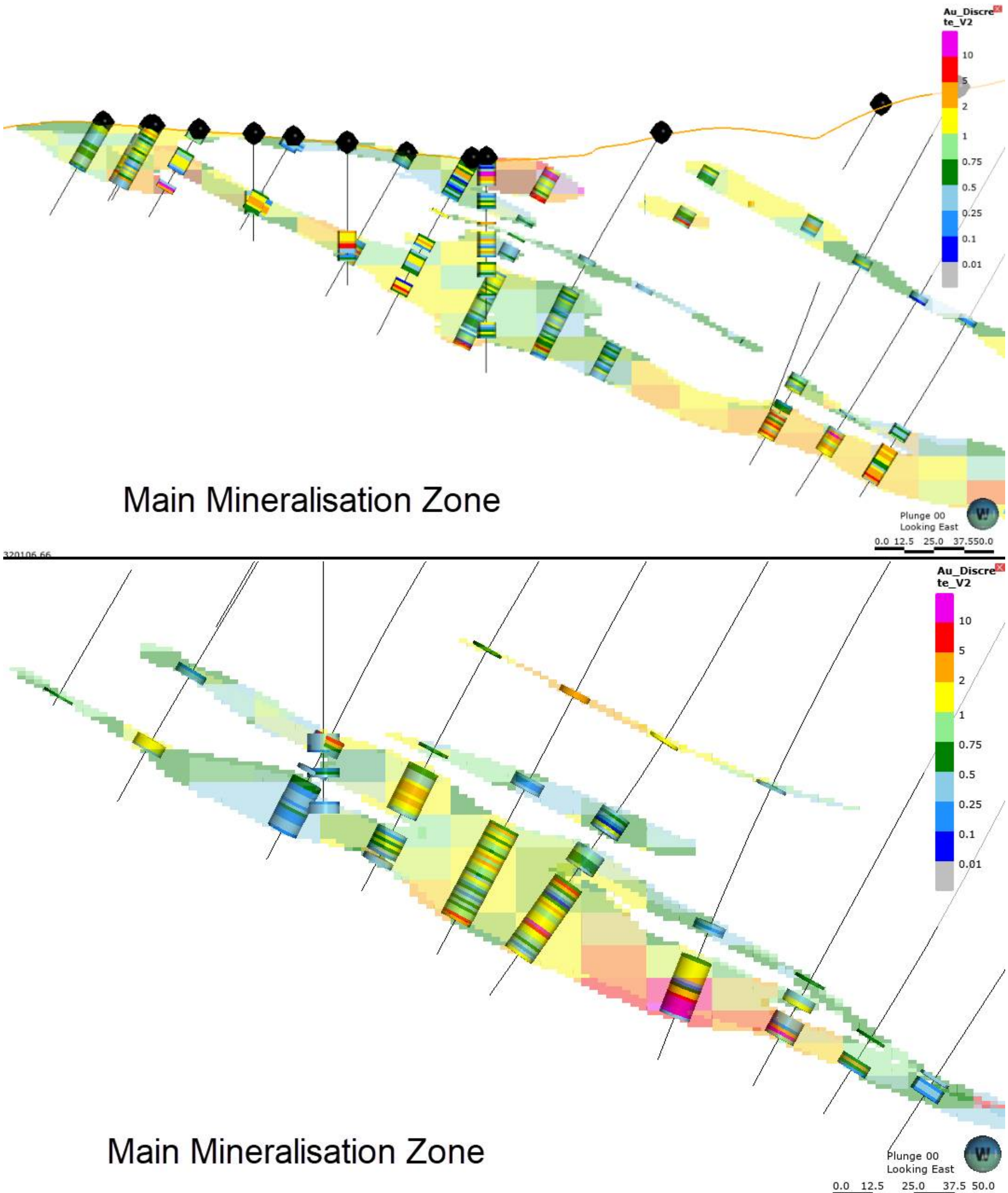


Figure 14-19: Estimated Block Grades (Au g/t) Versus Input Composite Grades in MMZ (Cross-sections looking east)

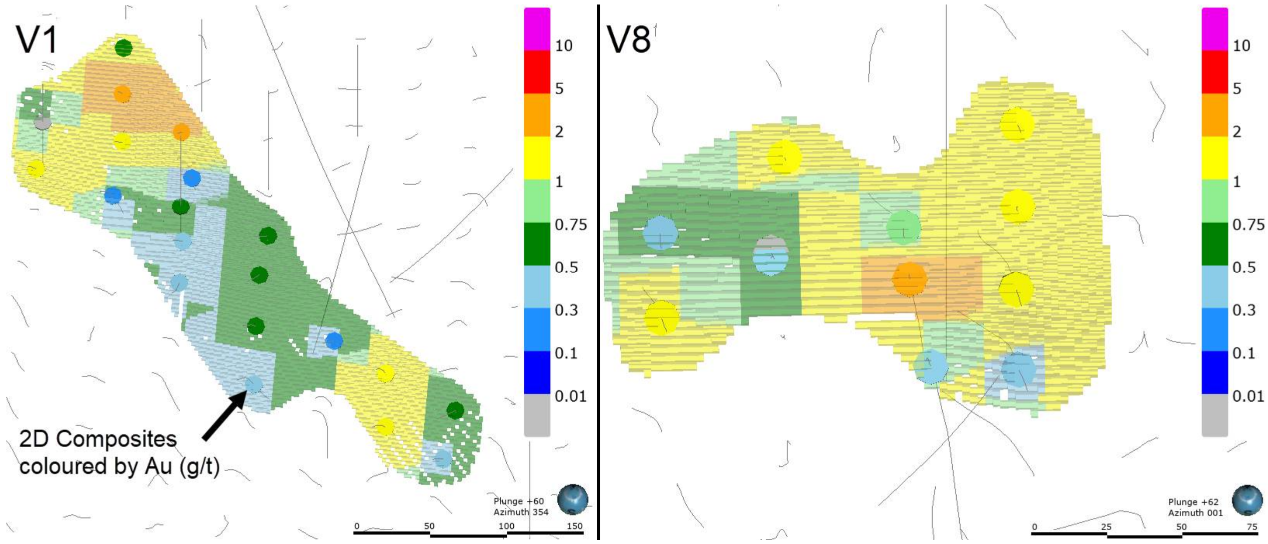


Figure 14-20: Isometric Views Showing Estimated Block Grades Versus Input Composite Grades (Domains V1 and V8)

14.11.3 Swath Plots

As part of the validation process swath plots were generated in the X (easting), Y (northing), and Z (vertical) coordinate directions. Average grades for input samples and estimated blocks are calculated along a series of vertical and horizontal slices (swaths) and plotted on graphs. In effect, a moving average is calculated for blocks and samples along three coordinate axes; this enables the fit of the block model to the underlying data to be assessed. The number of samples per swath are plotted as bars.

Examples of swath plots for Au within the MMZ and the V17 domains are shown in Figure 14-21 and Figure 14-22. Each of the mineralisation domains shows a good degree of correspondence between block model grades and composite grades in three dimensions, with the block model displaying a more smoothed profile, as anticipated with the Ordinary Kriging interpolation method used. Where the Inverse Distance (squared) interpolation method was used for the estimates in smaller, relatively poorly supported vein domains, the block estimates are typically slightly less smoothed.

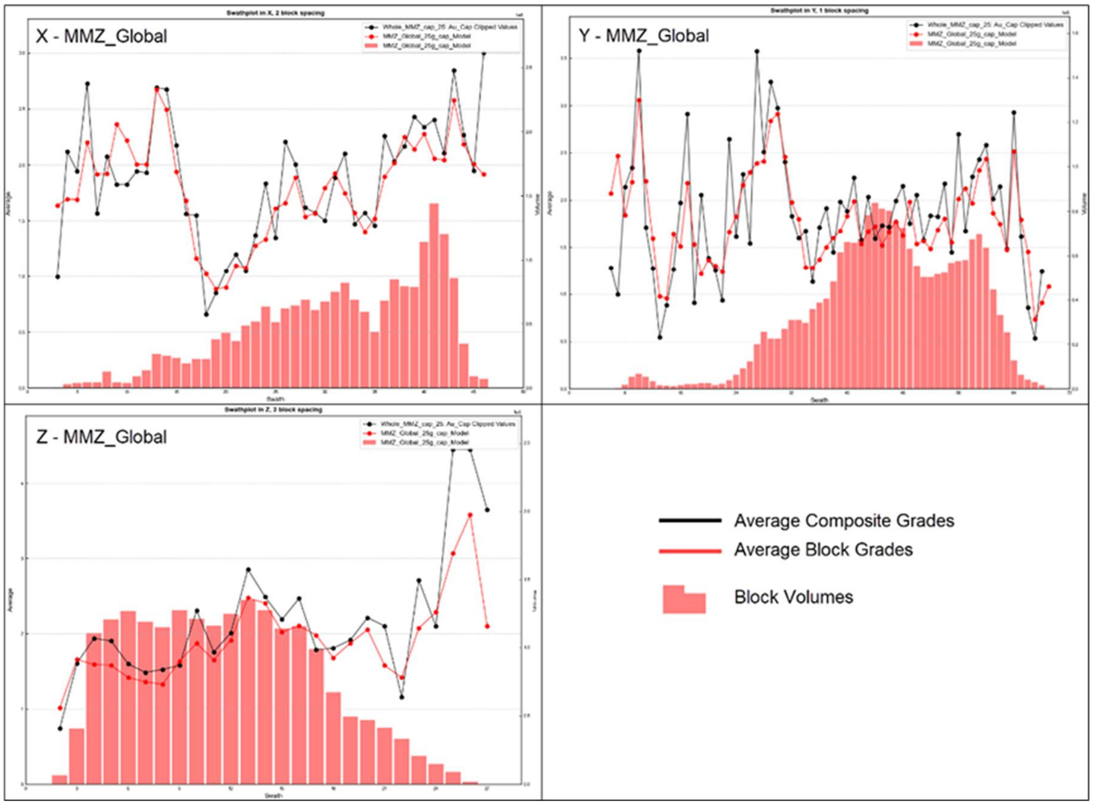


Figure 14-21: X, Y & Z Swath Plots Showing Estimated Au Grades Versus Input Composite Grades (MMZ Domain)

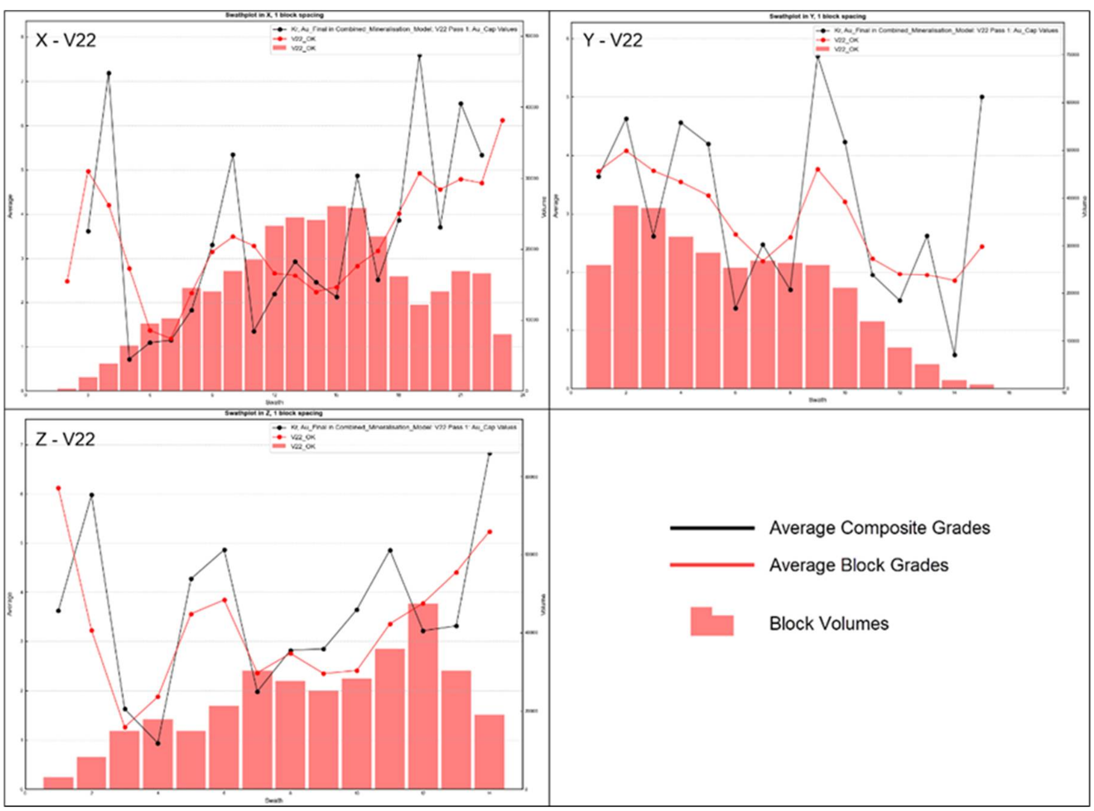


Figure 14-22: X, Y & Z Swath Plots Showing Estimated Au Grades Versus Input Composite Grades (Vein 22 Domain)

14.11.4 Statistical Validation

To globally validate the estimates, the mean of the capped composite grades were compared to the mean of the estimated block grades, on a domain-by-domain basis (Table 14-10). Estimated Au grades generally correspond well with average input composite values, being within ±4% for each of the MMZ structural sub-domains, and within 1% globally. There are larger discrepancies between estimated block mean and input composite means grades for some of the much smaller vein domains supported by relatively few composites (typically <50 composites). The mean composite grades of the smallest vein domains are sometimes significantly skewed by a small number of capped high-grade composites, however SRK is satisfied that through visual checks and reviews of swath plots, the estimated block grades of these domains are broadly representative of the input sample grades. The largest discrepancy between the mean composite and block grades is for domain V32, where two high grade intercepts influence a relatively large volume of blocks with high estimated grades.

Table 14-10: Mean Composite Grades Compared to Mean Estimated Block Grades

Domain	Number of Composites	Capped Comp. Mean (g/t Au)	Decl. Capped Comp. Mean (g/t Au)	Block Mean (g/t Au)	% Diff.	Decl. Window Size (x,y,z in m)
MMZ	15,745	1.98	1.81	1.81	0%	20x20x10
V1	32	0.99	0.96	0.94	-2%	20x20x10
V2	28	1.60	1.76	1.79	2%	20x20x15
V3	64	0.84	0.86	0.84	-2%	20x20x10
V7	31	1.05	0.98	1.00	2%	20x20x10
V8	21	1.31	1.25	1.31	5%	10x10x5
V9	16	1.46	1.27	1.22	-4%	20x20x15
V13	24	3.00	2.27	2.32	2%	20x20x10
V16	57	1.21	1.26	1.16	-8%	20x20x10
V17	119	1.23	1.11	1.12	1%	25x25x15
V20	81	1.56	1.60	1.52	-5%	20x20x10
V21	63	1.36	1.20	1.16	-3%	25x20x10
V22	274	3.30	3.12	3.21	3%	20x20x10
V23	47	3.06	3.14	3.17	1%	25x20x10
V25	99	2.70	2.44	2.28	-7%	15x15x5
V26	37	2.18	2.29	2.23	-3%	20x20x5
V27	13	5.90	5.74	5.60	-2%	20x20x10
V28	72	8.59	7.20	7.49	4%	20x20x10
V29	68	2.54	2.39	2.44	2%	20x20x10
V30	33	1.39	1.37	1.22	-11%	20x20x5
V31	139	1.69	1.59	1.66	4%	15x15x5
V32	46	3.25	3.29	2.81	-15%	20x20x10
LAT	255	2.53	2.41	2.30	-5%	20x20x15
WMZ1	298	1.92	1.98	1.93	-3%	20x20x10

14.11.5 Global Change of Support Analysis

In order to assess the degree of smoothing introduced into the estimated block model, SRK conducted a global change of support ('CoS') analysis using the Discrete Gaussian ('DG') method, whereby the estimated grade distribution is compared to a theoretical grade-tonnage curve for a range of parent block sizes.

The DG approach to Global CoS analyses is a relatively robust model which de-skews a grade distribution, providing a reasonable indication of a theoretical unsmoothed grade tonnage curve. This grade tonnage curve can then be compared to the OK estimate at the parent block size, which provides an indication of the level of smoothing within the OK model. The DG model is a useful methodology to indicate whether the kriging process has over or under smoothed the composite data. Large variations in the OK model from the DG model indicate that the kriging process would require review.

Figure 14-23 shows global CoS grade-tonnage curves for parent block sizes of 10 x 10 x 5 m and 20 x 20 x 10 m, alongside the corresponding OK model curves for the MMZ. The grade tonnage curves indicate how the OK estimates have greater smoothing than the theoretical DG grade tonnage curves, as indicated by the gradient of the curves. The steeper the grade curve, the less smoothing is present in the model. This is to be expected with a smoothed OK model, as compared to the un-smoothed composite data. Figure 14-23 indicates that the OK estimates are relatively insensitive to parent block size around the reporting cut-off grade, though the global grade profile of the 20 x 20 x 10 m OK model shows a gradient/profile closer to the theoretical grade-tonnage profile and is therefore considered more appropriately smoothed than the OK model with 10 x 10 x 5 m parent blocks.

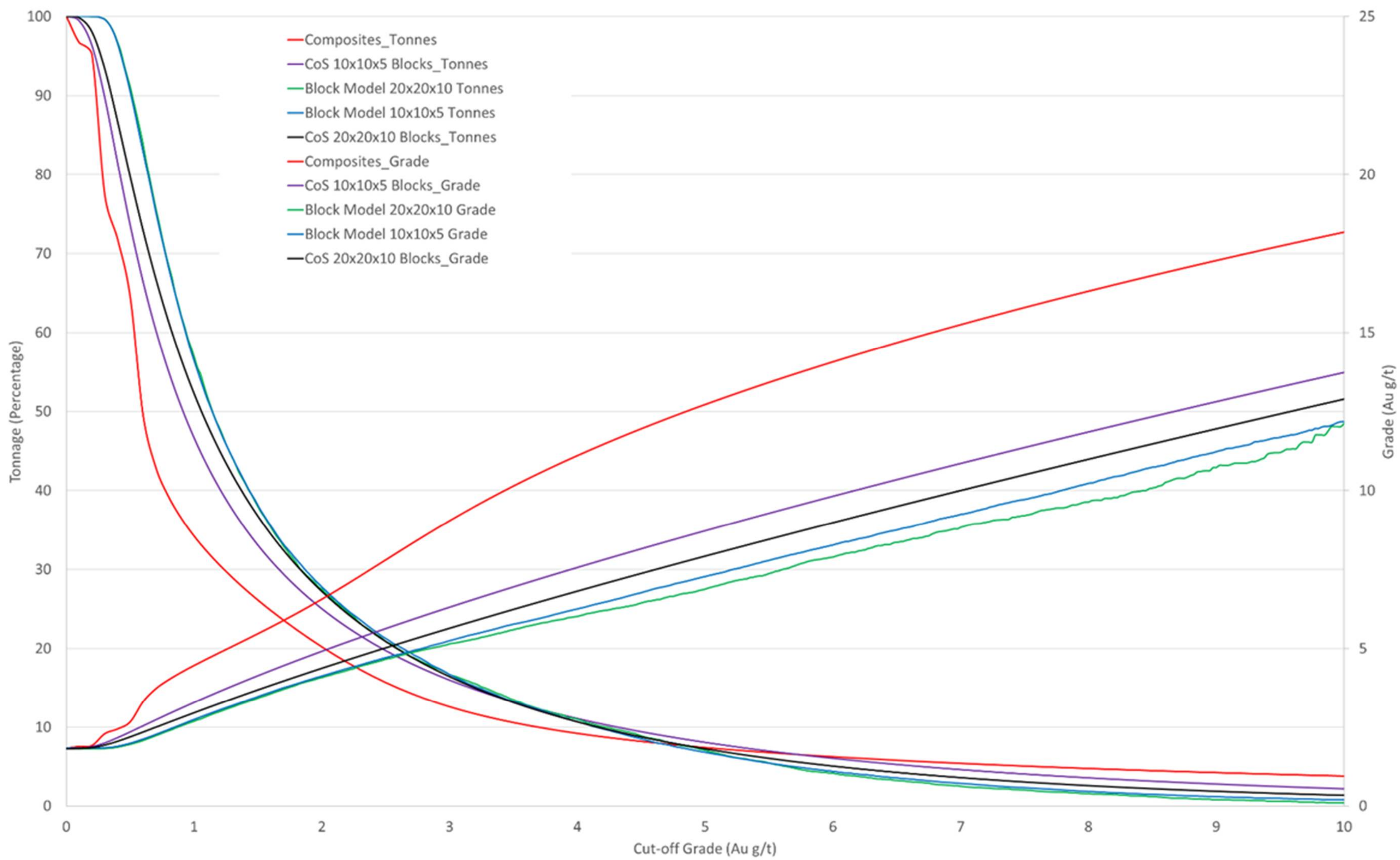


Figure 14-23: Global Change of Support Grade-tonnage Curves for the MMZ Compared with the Corresponding OK Block Model Grade-Tonnage Curves

Overall, SRK considers that the Au block estimates at the selected parent block size 20 x 20 x 10 m reflect the current understanding of the distribution of mineralisation and is an acceptable basis for reporting the Mineral Resource statement.

14.12 Mining Depletion

To date, the Lafigué Deposit has not been mined on a commercial scale, but has been subject to substantial artisanal mining (Figure 14-24 A). During SRK's site visit in May 2021 several thousand artisanal miners were active on the site, including some uncontrolled blasting activities. Concerns around safety inhibited the completion of a detailed survey of the artisanal workings at the time, however an aerial drone survey was completed on 17 August 2021 to assess the surface expression of the activities. The survey was of sufficient resolution to resolve the main open pit working areas which were typically on the scale of 10s of metres at surface, and less than 10 m deep (Figure 14-25). Both SRK and Endeavour also observed deeper, and more laterally extensive trenches and access to underground workings, which were not resolvable from the drone survey (Figure 14-24B).

Figure 14-24: Photographs Showing Some of the PE 58 Artisanal Workings Observed (SRK Site Visit 15/05/2021)

Since September 2021, Endeavour with the Dabakala Gendarmes, have been undertaking an eviction exercise, whereby the artisanal miners are being removed from site. Endeavour have stated in correspondence with SRK (Appendix A) that within the Resource area, which is to be fenced at Lafigué, approximately 30% of artisanal miners were removed by the end of September 2021, with this proportion increasing to 60% by mid-December 2021 and 98% by late January 2022. SRK understands that these figures are estimates. In addition, the site was bulldozed in Q1 2022 by Endeavour, with no survey of the underground workings having been conducted. SRK understands that at the time of writing, the fence was still not completed, and the process of moving the artisanal miners from site was continuing.

In order to account for artisanal mining depletion in the reporting of the current Mineral Resource Statement, and in the absence of a more recent survey since August 2021 and before the site was bulldozed in 2022, SRK has used the available height data obtained from the August 2021 drone survey to deplete the tonnes and grade from the main artisanal open pit excavations across the Lafigué deposit. In these areas, the density and grade fields in the block model have both been set to zero. In addition, in order to account for the depletion of the so-far poorly quantified volume of material mined from smaller trenches and underground workings, SRK has set all block grades to zero to a depth of 5 m below the pre-mining Lidar topographic surface, within a defined set of boundaries considered to reflect the approximate lateral extent of artisanal mining activities (Figure 14-25 and Figure 14-26). Where the drone survey height data indicates that individual pits reach a depth greater than 5 m below the pre-mining topography, these volumes of the model have been depleted to the maximum depth extents surveyed. SRK considers this approach represents a reasonable approximation of the understanding of the average depth of workings across the deposit area. SRK stresses that some localised areas of mining are known to have reached significantly greater depths, including areas of up to 20 m below the pre-mining surface. SRK also highlights that the approach outlined above is based upon the last reliable survey of the site in August 2021, and that artisanal mining is known to have continued beyond this date. However, it has not been possible to improve on the estimate of the artisanal mining which occurred after the August 2021 drone survey due to the levelling of the site. SRK considers that this represents a risk to the Mineral Resource Statement presented herein and, in particular, the early stages of the mine plan.

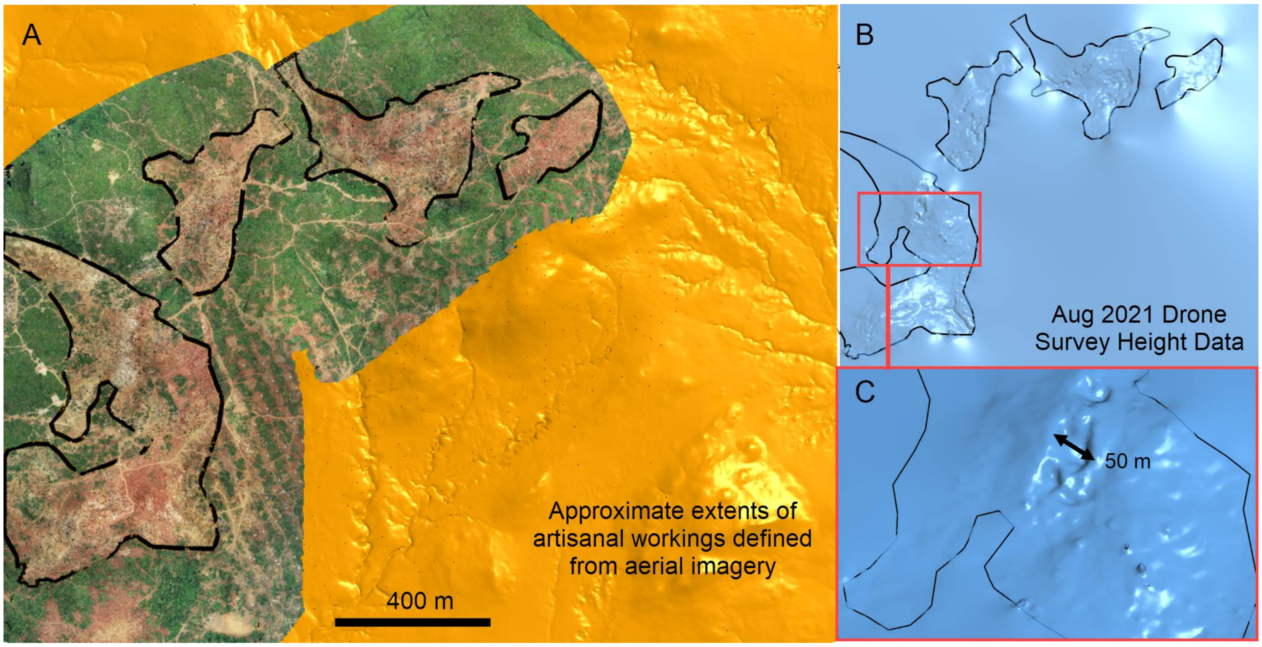


Figure 14-25: Plan Images Showing Aerial Imagery Draped on the Pre-Mining Topography

Figure 14-25 notes:

- Figure (A) illustrates the interpreted approximate extent of artisanal mining activity, as delineated by the black lines.
- Figure (B) and (C) illustrate the drone survey height data (DTM) used to deplete the main open pit excavations.

In order to assess the sensitivity of the stated Mineral Resources to the potential for more extensive or deeper artisanal depletion, SRK has completed an analysis to quantify the material currently present in the block model to depths of (10, 15 and 20) m below the pre-mining/pre-levelled topography (Figure 14-26 and Table 14-11). In each case, SRK has set the block model grades to zero but retained the block density so as to assume all the Au metal has been depleted without significant mining of waste rock. In all scenarios both the density (and as such, the tonnage) and grade have been set to zero for all of the open pit volumes surveyed in August 2021. The table is reported on a global grade-tonnage inventory basis (Table 14-11, for illustrative purposes only). Table 14-11 shows that depletion to a 5 m depth across the deposit, results in a reduction of the oxide material inventory by 64%, with the majority of oxide material depleted at a depth of 10 m below the pre-mining topography. Transitional material is minimally impacted by depletion to a depth of 5 m, and only significantly impacted by depletion to a depth of 15 m or greater. SRK notes that there is limited impact on the material currently classified as Inferred Mineral Resources as the majority of this material is located in the down-dip areas of the deposit.

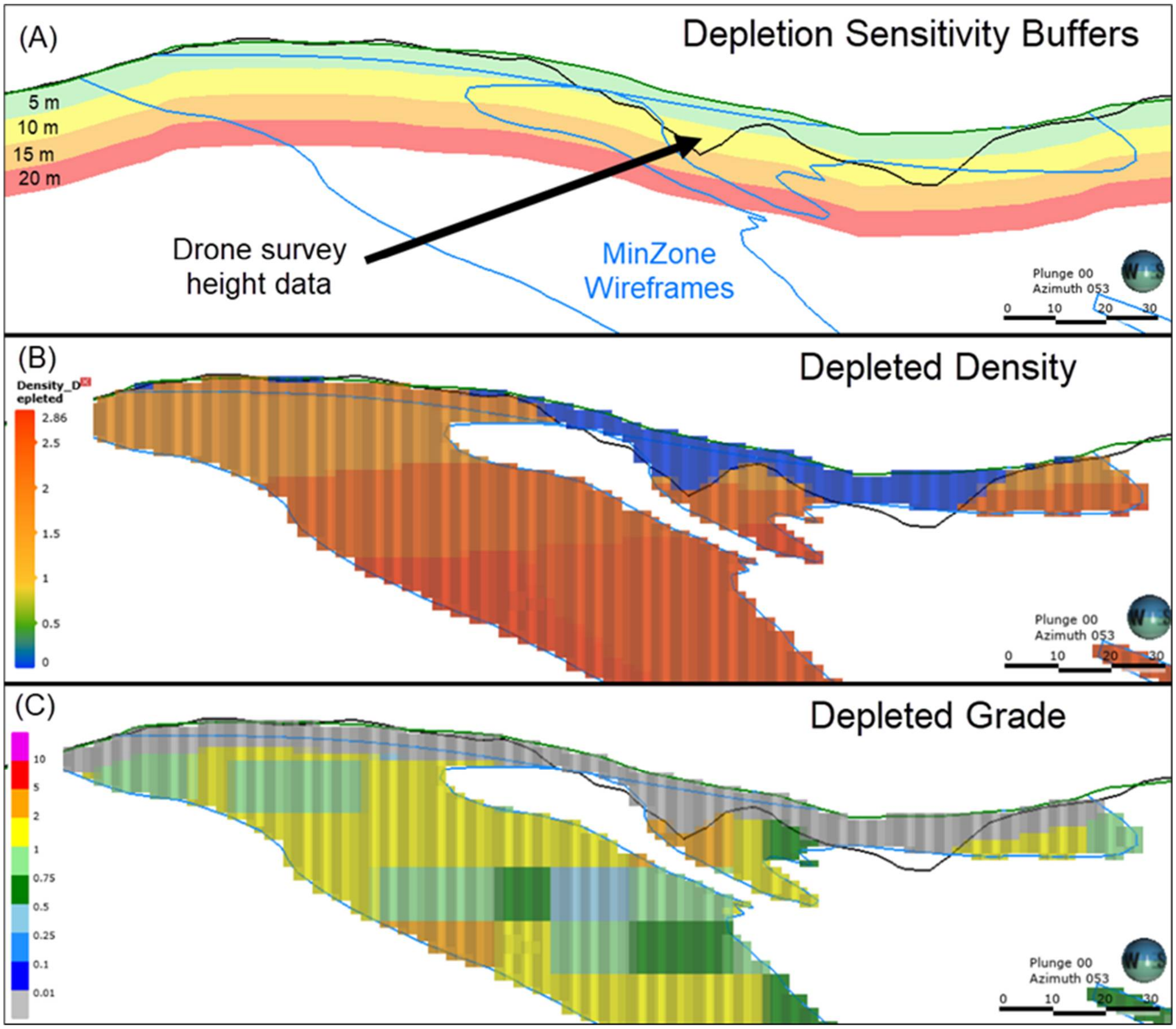


Figure 14-26: Cross-sections Showing Depletion Sensitivity Buffers and Approach to Depleting Tonnes and Grade

Figure 14-26 notes:

- Figure (A) illustrates the drone depletion survey surface (black line) and the pre-mining topographic buffer volumes used for the depletion sensitivity Analysis (5, 10, 15 and 20 m).
- Figure (B) illustrates the block model density depletion, accounting for the volume of material mined from the drone surveyed pits.
- Figure (C) illustrates the block model Au grade depletion, accounting for both the surveyed pits and also to a uniform depth of 5 m below the pre-mining topography outside of the surveyed pits.

Table 14-11: Block Model Inventory Inclusive of a Sensitivity to Artisanal Depletion - Indicated and Inferred Material

	Material Type	Tonnes (Mt)	Average Grade Au (g/t)	Contained Metal (koz)	% Difference Metal
Undepleted	Oxide	1.8	1.9	111	-
	Transition	1.9	1.7	103	-
	Fresh	45.2	2.1	2989	-
	Total	**48.8**	**2.0**	**3203**	-
After application of 5 m depletion	Oxide	0.8	1.5	40	-64%
	Transition	1.8	1.7	99	-4%
	Fresh	45.2	2.1	2989	0%
	Total	**47.8**	**2.0**	**3128**	**-2%**
After application of 10 m depletion	Oxide	0.4	1.3	16	-86%
	Transition	1.6	1.7	86	-16%
	Fresh	45.2	2.1	2989	0%
	Total	**47.2**	**2.0**	**3091**	**-3%**
After application of 15 m depletion	Oxide	0.2	1.2	7	-93%
	Transition	1.2	1.6	62	-39%
	Fresh	45.1	2.1	2985	0%
	Total	**46.5**	**2.0**	**3055**	**-5%**
After application of 20 m depletion	Oxide	0.1	1.1	5	-96%
	Transition	0.7	1.6	37	-64%
	Fresh	45.1	2.1	2979	0%
	Total	**45.9**	**2.0**	**3021**	**-6%**

Table 14-11 notes:

- Indicated and Inferred material - for illustrative ourposes only
- *In all depletion scenarios, block grades and density values were set to zero above the provided drone survey height data. Below this level, only grade was set to zero to the specified depths.

In the absence of a detailed survey of all of the artisanal workings across the Project area, SRK highlights that the artisanal workings at Lafigué present a potentially significant risk to the early stages of the mine plan.

14.13 Mineral Resource Classification

The Mineral Resource estimate for the Lafigué deposit has been classified in accordance with the CIM Definition Standards and includes Indicated and Inferred Mineral Resources. In addition to the quality and quantity of exploration data supporting the estimates, SRK has considered the confidence in the geological continuity of the mineralised structures and the confidence in the tonnage and grade estimates, specifically:

- Grade data for the drilling campaigns has generally been collected and analysed using industry best practise. Where documentation of operating procedures is not available for historical drilling, this drilling has in most places been supported by the close spaced 2017 to 2019 drilling programme. Adequate quality control measures are in place to monitor laboratory performance, drillhole collars have been surveyed using a differential GPS, and downhole surveys have been collected appropriately.

- The QAQC analyses presented is generally of a sufficient quality to support the subsequent geological modelling and grade and tonnage estimate.
- The current geological model is based on a combination of litho-structural and assay data derived from DD and RC exploration drilling. Where stacked E-W-trending shear structures and lithology contacts act as the primary mineralisation-controlling features, confidence in the modelled grade continuity is relatively high for the main mineralised structures (MMZ) but somewhat reduced for some of the smaller, less continuous veins domains, particularly in Lafigué Centre.
- The quality of the grade estimations has been reviewed on a global and local basis using various validation techniques and is considered a reasonable representation of the input sample grades.

SRK considers that the quality and spatial distribution of the data used, the geological continuity of the mineralisation and the quality of the estimated block model for Lafigué is sufficient for the reporting of Indicated and Inferred Mineral Resources, in accordance with the CIM Definition Standards. Isometric and cross-sectional views of the classified block model are shown in Figure 14-27. A summary of the specific criteria used to classify the block model is provided in Sections 14.13.1 and 14.13.2.

14.13.1 Indicated Mineral Resources:

Where exploration drillholes used for the grade estimation are typically spaced at 20 to 40 m along sections, and 40 to 50 m between sections, providing a reasonable level of confidence in geological and grade continuity, SRK has classified this material as Indicated Mineral Resources. These areas of the model show a reasonable degree of grade continuity and typically coincide with modelled lithological contacts or lie within the intrusive felsic unit. These areas are also typically estimated by search passes 1 or 2 (see Section 14.9.2).

14.13.2 Inferred Mineral Resources:

Where exploration drillholes used for the grade estimation are typically spaced at 50 to 75 m in areas along-strike and down-dip from areas classified as Indicated Mineral Resources, SRK has classified this material as Inferred Mineral Resources. Additionally, areas drilled at closer spacings but where mineralisation controls are less well understood, and continuity is typically reduced, such as observed in some parts of Lafigué Centre, are also classified as Inferred Mineral Resources. These areas of the block model were primarily estimated in search passes 2 or 3.

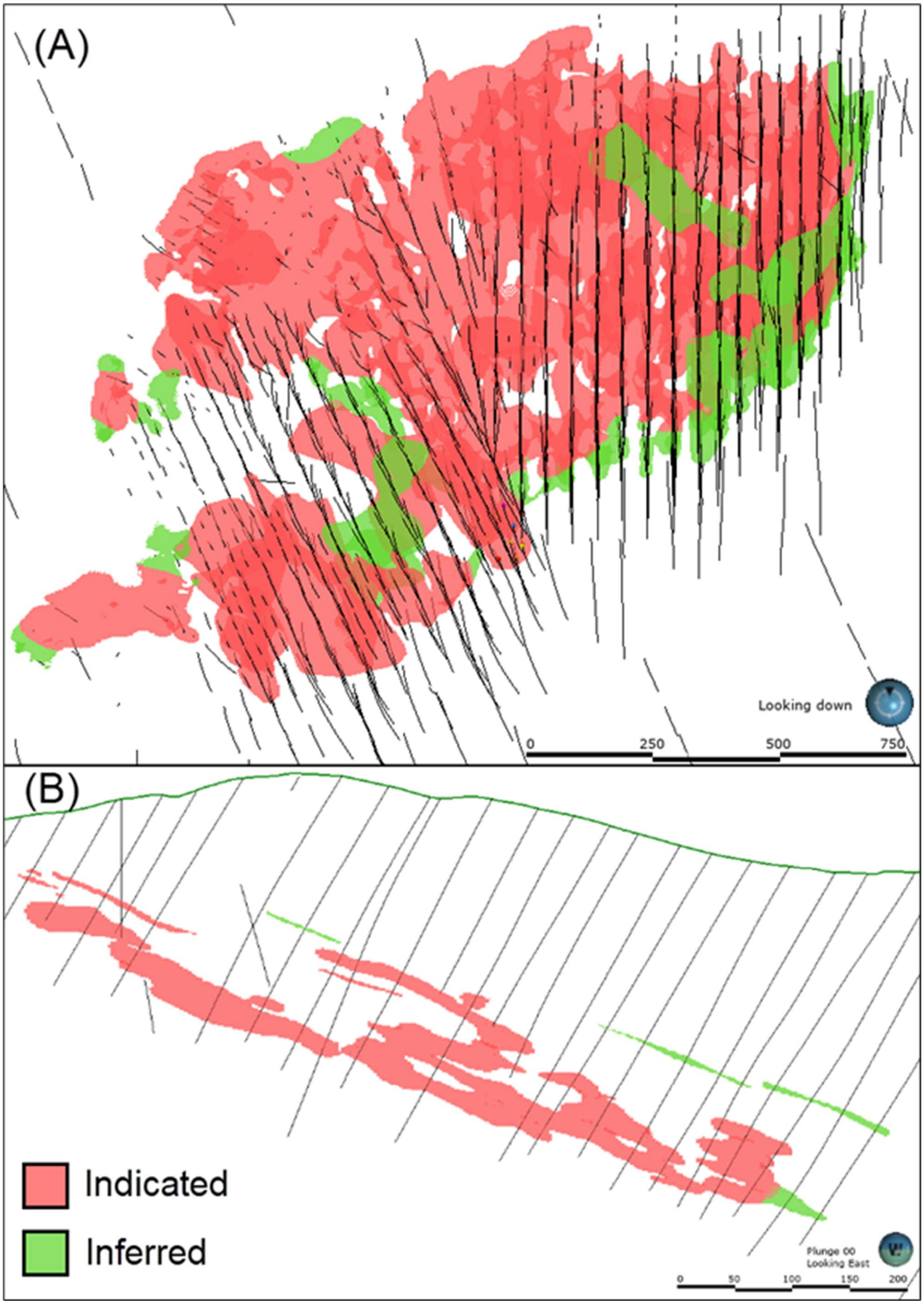


Figure 14-27: Isometric (A) and Cross-section (B) Views of the Classified Lafigué Block Model

14.14 Assessment of Reasonable Prospects for Eventual Economic Extraction ('RPEEE')

14.14.1 Economic and Technical Input Parameters

In order to determine which portion of the block model has reasonable prospects for eventual economic extraction by open-pit mining methods, SRK has applied basic economic considerations based on previous technical studies in order to generate an optimised pit shell within which the Mineral Resource is to be reported. The pit optimisation study has been carried out on the Mineral Resource based on a long-term gold price and technical parameters established as part of previous/current studies. The pit optimisation identifies material within the model with potential for open pit mining above a suitable gold cut-off grade. SRK has reviewed the economic and technical parameters used in the pit optimisation exercise and considers them appropriate for the purpose of indicating the proportion of the block model that demonstrates reasonable prospects for eventual economic extraction ('RPEEE').

The parameters used for the pit optimisation exercise are summarised in Table 14-12.

Table 14-12: Key Technical/Economic Parameters for the Lafigué Conceptual Pit Optimisation and CoG Calculation

Parameters	Units	Value	Source/Basis
Production Rate	(Mt/a db)	4.00	
Geotechnical			
Max Surface Elevation	(Z Elevation)	420	
Laterite & Saprolite (Oxide) & Transition	(Z Elevation)	Above weathering wireframe	
		OSA	
	(Degrees)	33.0	END-011-GD DRAFT Report.pdf
Fresh	(Z Elevation)	Below weathering wireframe	
		IOSA	
	(Degrees)	33 to 51	END-011-GD DRAFT Report.pdf
Mining Factors			
Dilution	(%)	9	Regularisation - block size (5 x 5 x 2.5) m
Recovery	(%)	98	Regularisation- block size (5 x 5 x 2.5) m
Processing			
Recovery - Au (oxide)	(%)	94.9	IF(Au≥34,99.7,IF(Au≤0.2,(68*Au+82.2),(0.769*LN(Au)+97)))/100)-0.5%
Recovery - Au (transition)	(%)	94.9	
Recovery - Au (fresh)	(%)	95.1	Lycopodium 2021
Operating Costs			
Wavg Waste Mining Cost	(USD/t_{rock})	2.65	2021 PFS Financial Model
Incremental Mining Cost	(USD/m bench)	0.0022	2021 PFS Financial Model
Reference Level	(Z Elevation)	350.00	2021 PFS Financial Model
Wavg Ore Mining Cost	(USD/t_{ore})	2.12	2021 PFS Financial Model
Rehandle Cost	(USD/t_{ore})	0.37	2021 PFS Financial Model
OFF ROM Rehandle	(USD/t_{ore})	0.79	2021 PFS Financial Model
Rehabilitation Cost	(USD/t_{rock})	0.06	2021 PFS Financial Model
CIL – Oxide	(USD/t_{ore})	7.47	Lycopodium 2021

Table 14-12: Key Technical/Economic Parameters for the Lafigué Conceptual Pit Optimisation and CoG Calculation

Parameters	Units	Value	Source/Basis
CIL – Transition	(USD/t_{ore})	7.47	Lycopodium 2021
CIL – Fresh	(USD/t_{ore})	9.13	Lycopodium 2021
G&A	(USD/t_{ore})	5.60	2021 PFS Financial Model
Sustaining Capital (SIB)	(USD/t_{ore})	1.87	2021 PFS Financial Model
Selling Cost Au (@USD1500/oz)	(USD/oz)	71.8	EDV 2021 Assumptions
Metal Price			
Gold	(USD/oz)	1500	EDV 2021 Assumptions
Gold	(USD/g)	49.83	
Discount Rate	(%)	5%	EDV 2021 Assumptions
Cut-Off Grade			
Marginal Operating Costs (oxide)	(USD/t_{ore})	16.71	
Marginal Operating Costs (transition)	(USD/t_{ore})	16.86	
Marginal Operating Costs (fresh)	(USD/t_{ore})	19.11	
Marginal Cut-Off Grade (oxide)	(g/t Au)	0.4	
Marginal Cut-Off Grade (transition)	(g/t Au)	0.4	
Marginal Cut-Off Grade (fresh)	(g/t Au)	0.4	
IS Marginal Cut-Off Grade (oxide)	(g/t Au)	0.4	
IS Marginal Cut-Off Grade (transition)	(g/t Au)	0.4	
IS Marginal Cut-Off Grade (fresh)	(g/t Au)	0.5	

Table 14-12 Notes: t_{ore} = ore tonnes; t_{rock} = total rock tonnes; IS = In situ; Wavg = Weighted average; OSA = Overall Slope Angle

14.14.2 Cut-off Grade

Based on the above pit optimisation study and associated technical and economic input parameters, the in situ marginal cut-off grades determined for reporting the Mineral Resource are given below:

- 0.4 g/t Au for oxide;
- 0.5 g/t Au for transition; and
- 0.5 g/t Au for fresh.

14.15 Mineral Resource Statement

The 2021 Mineral Resource statement for the Lafigué gold deposit is shown in Table 14-13.

Table 14-13: Mineral Resource Statement for the Lafigué Gold Project, Effective Of 15 May 2022*

Classification	Material Type	Tonnes Mt	Au Grade g/t	Contained Metal (Au) koz
Indicated	Oxide	0.7	1.55	36
	Transition	1.7	1.71	94
	Fresh	43.8	2.06	2896
	Total	**46.3**	**2.03**	**3027**
Inferred	Oxide	0.1	1.22	4
	Transition	0.1	2.05	4
	Fresh	1.4	2.11	94
	Total	**1.5**	**2.05**	**102**

**In reporting the Mineral Resource Statement, SRK notes the following:*

- The reported Mineral Resources are depleted to a drone survey provided to SRK by Endeavour. The survey was conducted on 17 August 2021 and only accounts for artisanal open pit development at surface. SRK understands that there were further artisanal mining workings underground, but these could not be captured by the drone survey. To account for this, outside of (and below, where necessary) the artisanal open pit workings, to a depth of 5 m below the pre-mining topography, the grades have been reduced to zero. In the absence of any underground survey, and to reflect the uncertainty for these areas, SRK has not depleted the tonnages.
- Since September 2021, Endeavour have been undertaking an eviction exercise whereby the artisanal miners are being removed from site. Endeavour have stated in correspondence with SRK that as of late January 2022, 98% of the artisanal miners were removed. In the absence of an updated survey and groundworks completed at site, SRK highlights the risk associated with more extensive depletion due to ongoing artisanal mining activity in the intervening period and or more extensive workings in the prior period, than is accounted for in this Mineral Resource Statement. A sensitivity analysis is provided in the accompanying report to inform the reader of the associated risks.
- The reported Mineral Resources have an 'Effective Date' of 15 May 2022. The Competent Person for the declaration of Mineral Resources is Dr Lucy Roberts, MAusIMM(CP), of SRK Consulting (UK) Ltd. The Mineral Resource estimate was authored by Dr James Davey, also of SRK;
- Technical and economic assumptions were agreed between SRK and LMCI/Endeavour for mining factors (mining and selling costs, mining recovery and dilution, pit slope angles) and processing factors (gold recovery, processing costs), which were used to run a pit optimisation exercise. These factors were developed as part of the ongoing Feasibility Study for the Lafigué project, as stated below:
  - Mining cost: 2.12 (USD/tore)
  - Waste mining cost: 2.65 (USD/trock)
  - Processing cost: Oxide/Transition: 7.47 (USD/tore); Fresh: 9.13 (USD/tore)
  - Selling cost: 71.8 (USD/oz Au)
  - Mining recovery: 98%
  - Mining dilution: 9%
  - Processing recovery: Oxide = 94.87%; Transition = 94.92%; Fresh = 95.08%
  - Average slope angles: (33 to 51)°, dependent on geotechnical domain
  - G&A cost: 5.60 (USD/tore)
  - Discount rate: 5%
- SRK considers there to be reasonable prospects for eventual economic extraction by constraining the Mineral Resources within an optimised open pit shell using a gold price of USD 1500/oz.
- Mineral Resources are reported within the optimised pit shell using cut-off of grades of 0.4 g/t Au (oxide); 0.5 g/t Au (transition) and 0.5 g/t Au (fresh), which are the marginal cut-off grades for CIL processing determined during the pit optimisation.
- Mineral Resources are reported as in situ and undiluted, with no mining recovery applied in the Statement. All tonnages are reported on a dry basis.
- Mineral Resources are not Ore Reserves and do not have demonstrated economic viability, nor have any mining modifying factors been applied;
- Tonnages are reported in metric units, grades in grams per tonne (g/t), and the contained metal in kilo troy ounces. Tonnages, grades, and contained metal totals are rounded appropriately. 1 troy ounce is assumed to be the equivalent of 31.1034 g
- Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content. Where these occur, SRK does not consider these to be material.

14.16 Grade-tonnage Curves

The results of grade-tonnage sensitivity analysis completed for Indicated Mineral Resources (given the very limited total contribution of Inferred Mineral Resources) at Lafigué are shown in Figure 14-28 to Figure 14-30 following, split by material type. This is to show the continuity of the grade estimates at various cut-off increments and the sensitivity of the Mineral Resource to changes in Au (g/t) cut-off. The tonnages and grades in these charts, however, should not be interpreted as Mineral Resource statements.

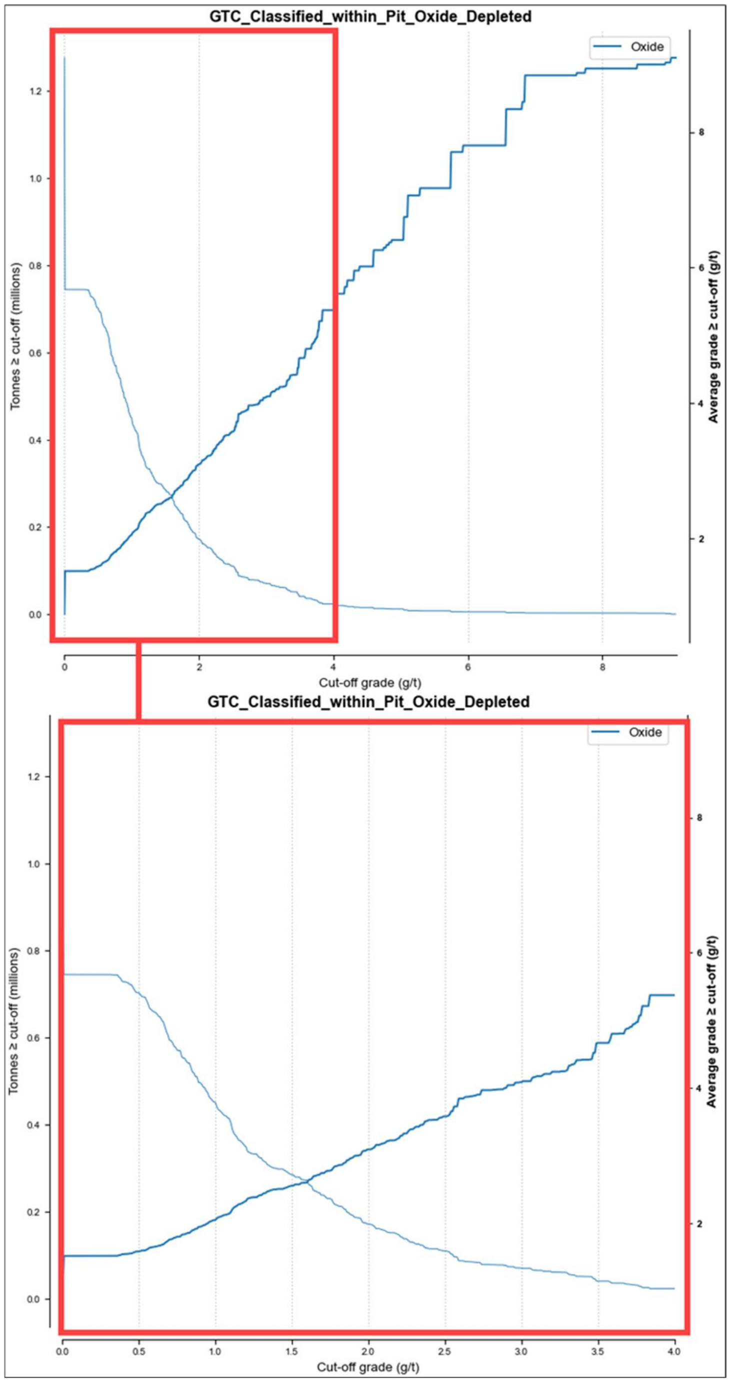


Figure 14-28: Grade-tonnage Curve for Indicated Oxide Material within the USD 1500 Optimised Pit Shell After Depletion

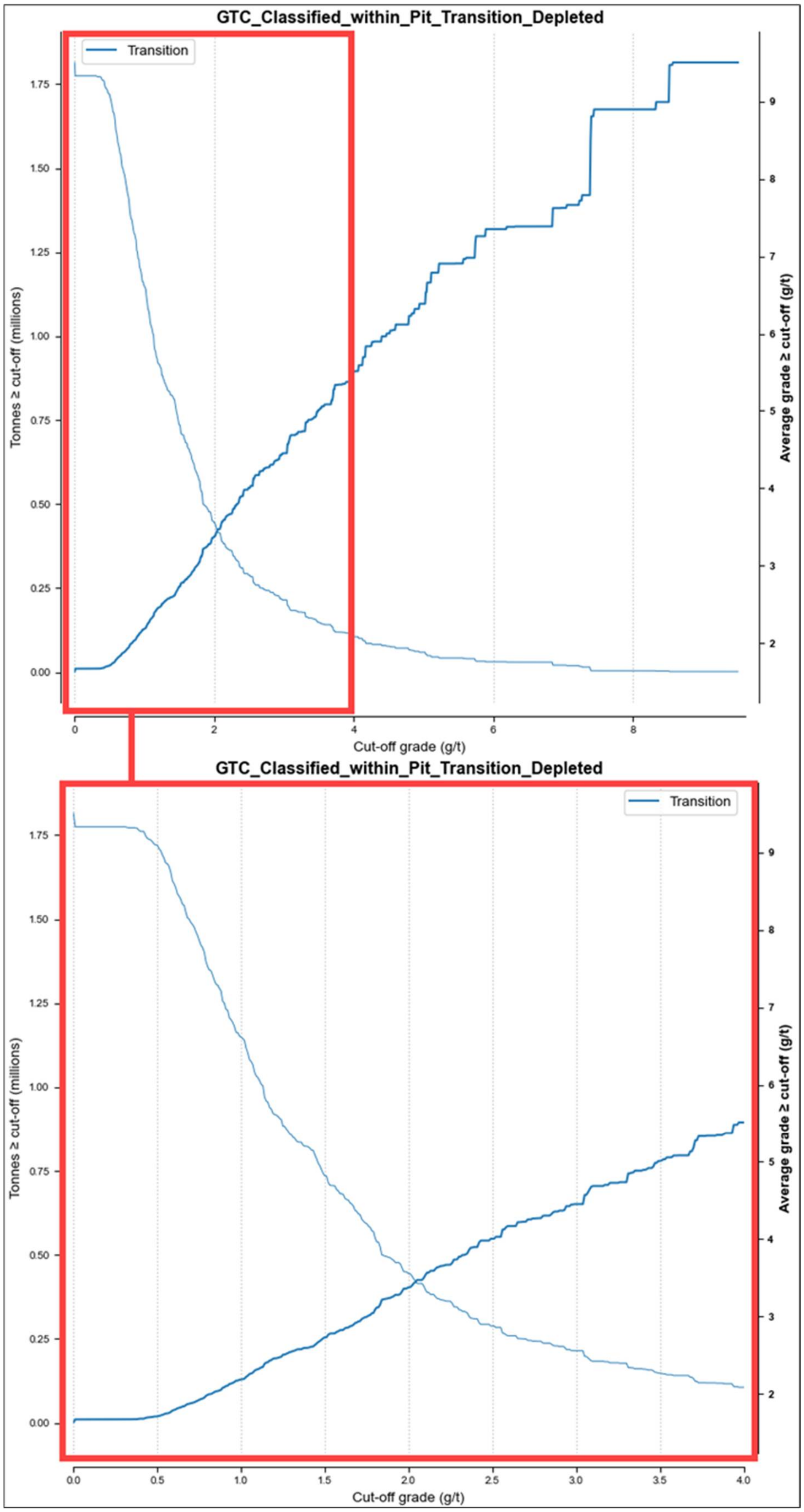


Figure 14-29: Grade-tonnage Curve (Indicated Transition Material Within the USD 1500 Optimised Pit Shell After Depletion)

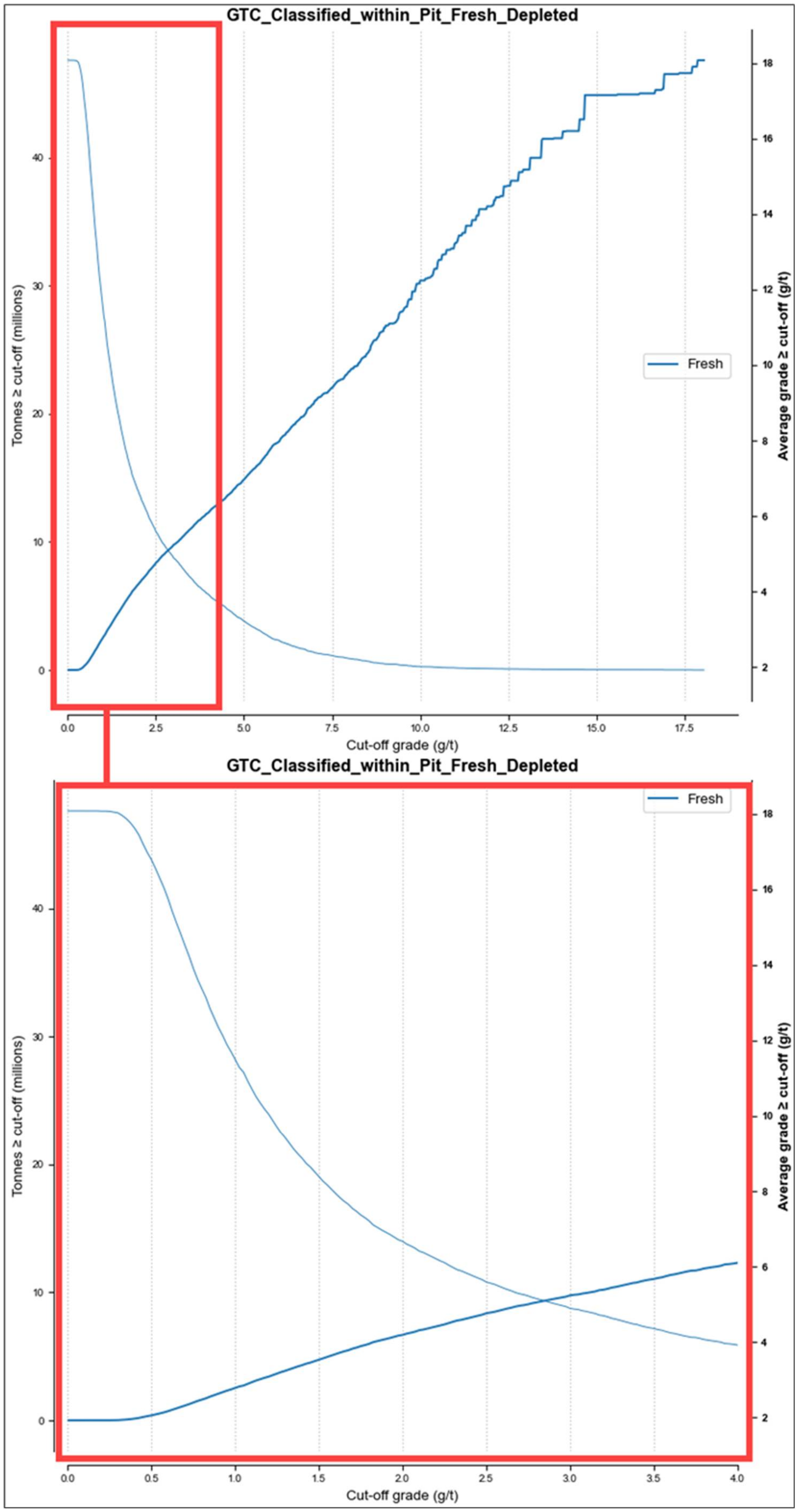


Figure 14-30: Grade-tonnage Curve (Indicated Fresh Material within the USD 1500 Optimised Pit Shell After Depletion)

14.17 Comparison with Previous Estimates

The previous Mineral Resource Statement for Lafigué was effective as of 21 September 2021, based on a previous version of the Mineral Resource model produced by SRK. A comparison of this Mineral Resource Statement with the 2022 SRK Resource Statement is provided in Table 14-14 following.

Table 14-14: Comparison of the 2021 and 2022 Resource Statements (based on the respective SRK models)

Model	Reporting Pit	Cut-off Grade (g/t Au)	Classification	Tonnes (Mt)	Grade Au (g/t)	Contained Metal Au (koz)
SRK (2021)	2021 (Au price: USD1500/oz)	Oxide: 0.4 Transition: 0.5 Fresh: 0.5 (IS MCOG)	Measured	-	-	-
			Indicated	44.8	2.0	2917
			Inferred	3.6	2.4	270
			Total	**48.4**	**2.0**	**3186**
SRK (2022)	2022 (Au price: USD1500/oz)	Oxide: 0.4 Transition: 0.5 Fresh: 0.5 (IS MCOG)	Measured	-	-	-
			Indicated	46.3	2.03	3027
			Inferred	1.5	2.05	102
			Total	**47.8**	**2.04**	**3128**

**Rounding may result in apparent summation differences between tonnes, grade and contained metal content*

Since the SRK 2021 model was produced, approximately 245 additional infill drillholes have been completed during late 2021 and early 2022 in the Lafigué Mining Licence , primarily in Lafigué Centre and around the periphery of the deposit. In the Lafigué Centre area, the modelled mineralised structures were refined in the updated (2022) model to account for shorter-scale variability that was not apparent based on the previously available, wider-spaced drillholes, and some additional mineralised structures were also modelled. On a local basis some areas of the model reduced in volume whereas others increased (Figure 14-31), with intercepted grades generally remaining aligned with the global grade distribution already established from previous drilling and sampling at the deposit. Both the 2021 and 2022 SRK models define relatively broad packages of mineralisation above the natural cut-off of the available sample population. As a result, both models incorporated a greater proportion of samples within the 0.3 to 0.5 g/t Au grade range than pre-2021 models produced for Lafigué.

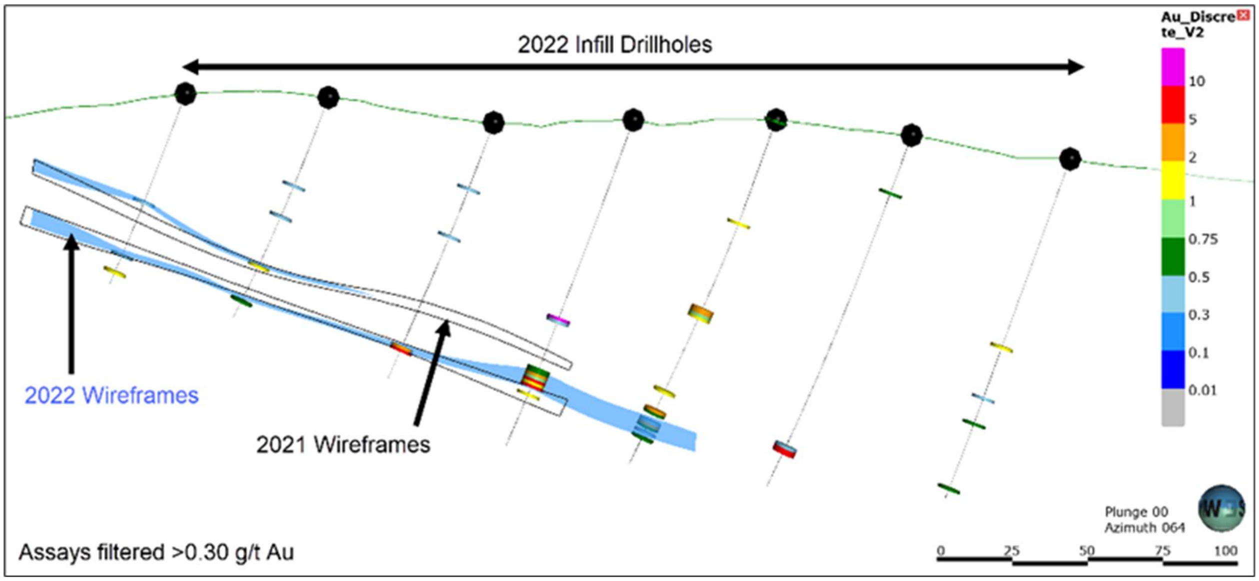


Figure 14-31: Cross-section (Looking East) in Lafigué Centre Comparing the Extents of the 2021 and 2022 Wireframes

The majority of the technical and economic parameters used during the pit optimisation process which were completed for the purposes of satisfying the RPEEE test, have not been modified or updated since the reporting of the 2021 Mineral Resource Statement for the Project.

14.18 Data Verification

The MRE presented herein is based upon geological observations, measurements and sample data described in earlier sections. The verification methodology applied to the supporting data is described more fully in Section 12.

14.19 Comments on Section 14

SRK considers that the geological model and subsequent grade and tonnage estimates developed for Lafigué are a reasonable representation of the in-situ mineralisation based on the available supporting data, and that the approaches and methods adopted are aligned with the CIM Definition Standards.

14.20 Interpretations and Conclusions

Interpretations, conclusions and risks pertaining to Section 14 are summarised in Section 25 of this Report.

14.21 Recommendations

Recommendations pertaining to Section 14 are summarised in Section 26 of this Report.

14.22 References

References cited in the preparation of Section 14, are detailed in Section 27 of this Report.

15. MINERAL RESERVE ESTIMATES

15.1 Introduction

Section 15 discusses the process followed to derive the Mineral Reserves for the Lafigué Project (the 'Project') in accordance with the Canadian National Instrument 43-101, and adhering to the CIM Definition Standards guidelines (CIM, 2014). The Section will specifically focus on the following essential items:

- Mineral Reserve estimation approach and methodology;
- hydrological conditions;
- mining geotechnics;
- pit optimisation;
- mine designs;
- waste rock dump design;
- Mineral Reserve statements;
- mine production schedule;
- mining strategy;
- mining cost; and
- opportunities and risks.

15.2 Mineral Reserve Estimation Approach

As reported herein, the Mineral Reserve statement for the Project is supported by the engineering designs and modifying factors discussed in Section 15.3.2. A site layout in Figure 15-1 shows the location of the pit relative to the process plant, run of mine ('RoM') stockpiles and waste rock dumps ('WRD').

The open pit is designed with various stage pushbacks, with smaller pits southeast of the Main Pit. The life of mine plan ('LoMp') for the Project includes the following key data inputs and activities:

- Resource block model modified to generate the mining block model through re-blocking, which introduces a degree of dilution.
- The pre-mining topographic surface.
- Open Pit optimisation analysis to include:
  - derivation of pit optimisation parameters, among other things: dilution, diluting grades and losses, metallurgical recovery and refining factors, commodity price and operating expenditure assumptions;
  - for unit mining operating expenditures, mining costs were derived from the final financial analysis done during the Pre-feasibility Study ('PFS') (Snowden, 2021) based on actual cost data from similar operations provided by Endeavour Mining plc ('Endeavour'). Costs include both waste and ore mining costs relationships with reduced level ('RL') elevations coded into the block models as a specific attribute;
  - derivation of marginal economic ore cut-off grades ('COG'), as appropriate; and
  - ultimate pit and staged pit selections.

- Engineering pit design assumptions inclusive of:
  - open pit access, including haul road designs;
  - geotechnical design considerations for pit slopes (inter ramp angles, overall slope angles, batter angles, stacked berm configurations and berm widths), which vary as appropriate with azimuth and depth to reflect the geotechnical domains as established for each deposit;
  - mine planning and production scheduling inclusive of production rates; stockpiling strategies; grade bin selection criteria; production capacities for mining; and processing activities; and,
  - Mineral Reserve reporting is based on aggregating all Measured and Indicated Mineral Resource blocks incorporated within the LoMp and reported inclusive of all appropriate dilution, diluted grade, and losses to enable the reporting of Mineral Reserves.

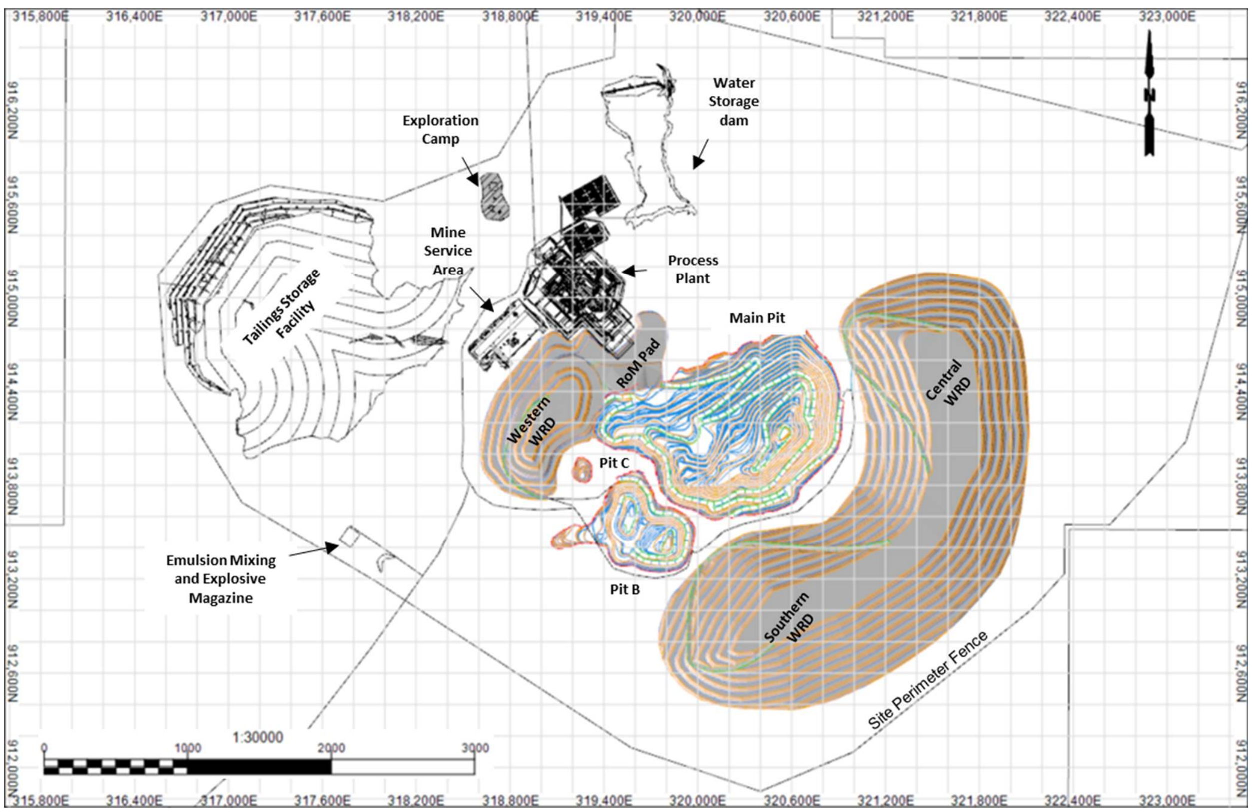


Figure 15-1: Project Site Layout (SRK,2022)

15.3 Mineral Reserve Assumption

15.3.1 Block Models and Surfaces

The geologic block models used in the Mineral Reserve estimation are:

- Resource Model – Data Mine Model - Lafigué _ResMod_May2022.dm
- Reserve Model – Geovia Surpac™ Mine Model - Lafigué _resmod_may2022_reg_5x5x25.mdl

The block model incorporates all typical attributes inclusive of gold grades, rock type (facies), weathering (oxidation) status, 'resource' confidence categories (Measured, Indicated, and Inferred) and density values. The block model was depleted with the pre-mining topography, including current artisanal mining operations. More background is available in the mine depletion section under the Resource Geology chapter in Section 14.

15.3.1.1 Artisanal Mining Activities

The height data obtained from the August 2021 drone survey was used to deplete the tonnes and grade from the main artisanal open pit excavations across the Lafigué deposit to account for the artisanal mining depletion in the current Mineral Reserve Statement. Gold depletion, which is not visible from the surface topography, was accounted for by removing all the gold grades within the active areas (Figure 15-2) to a depth of 5 m below the surface. Also, to account for dilution due to backfilling or bulldozing the artisanal area, an additional 10 % dilution was added within the mining schedule to all the oxide ore tonnes within the vicinity of artisanal activity.

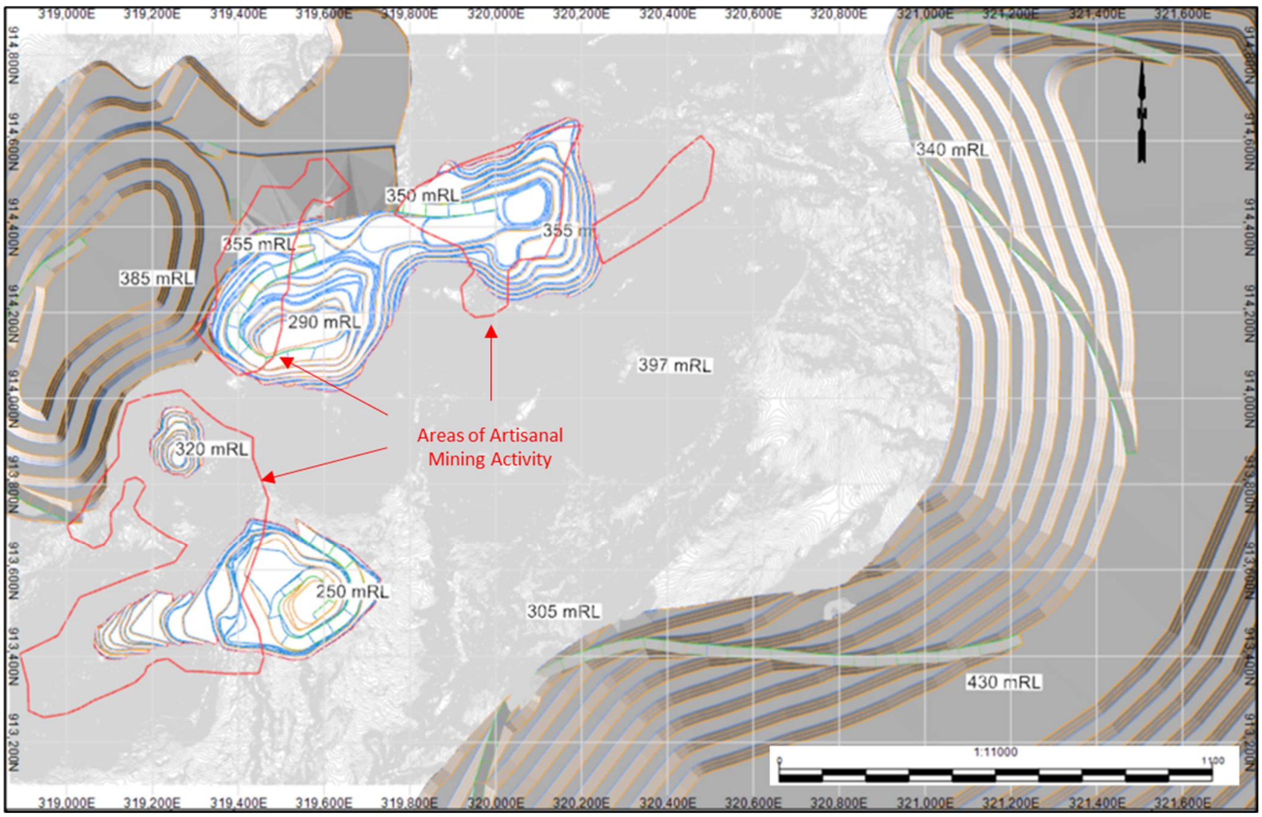


Figure 15-2: Areas Impacted by Artisanal Mining (SRK,2022)

15.3.2 Geotechnical Slope Design

BG has used inputs from the Hydrogeological Study undertaken by (Endeavour Management Service Abidjan, 24 May 2021) within their slope stability analyses. BG notes that whilst pore pressure distribution is a very significant factor in the influence on slope stability, the footwall slope's driving factor is the foliation planes' orientation and shear strength. More details on hydrogeology and open pit geotechnical engineering review can be found in Section 16, Mining Methods.

Table 15-1: Bastion Geotechnical Pit Geotechnical Design Criteria (Bastion, 2021)

Design Sector	Units	Oxide	DS2	DS3	DS4	DS5	DS6	DS7	DS8	DS9
Batter Face Angle	(°)	60.0	55.0	85.0	85.0	85.0	55.0	85.0	85.0	85.0
Batter Hight	(m)	10.0	10.0	20.0	20.0	20.0	20.0	20.0	20.0	20.0
Berm Width	(m)	6.0	6.0	12.3	12.3	13.9	11.6	13.9	13.9	13.9
Inter-Ramp Angle (IRA)	(°)	40.3	40.3	55.0	55.0	52.0	38.0	52.0	52.0	52.0
Bench Stack Height	(m)	50.0	50.0	100.0	100.0	100.0	50.0	100.0	100.0	100.0
Decoupling Berm Width	(m)	50.0	50.0	100.0	100.0	100.0	50.0	100.0	100.0	100.0
Overall Slope Angle (OSA)	(m)	33.0	33.0	51.0	51.0	48.0	35.0	48.0	48.0	48.0

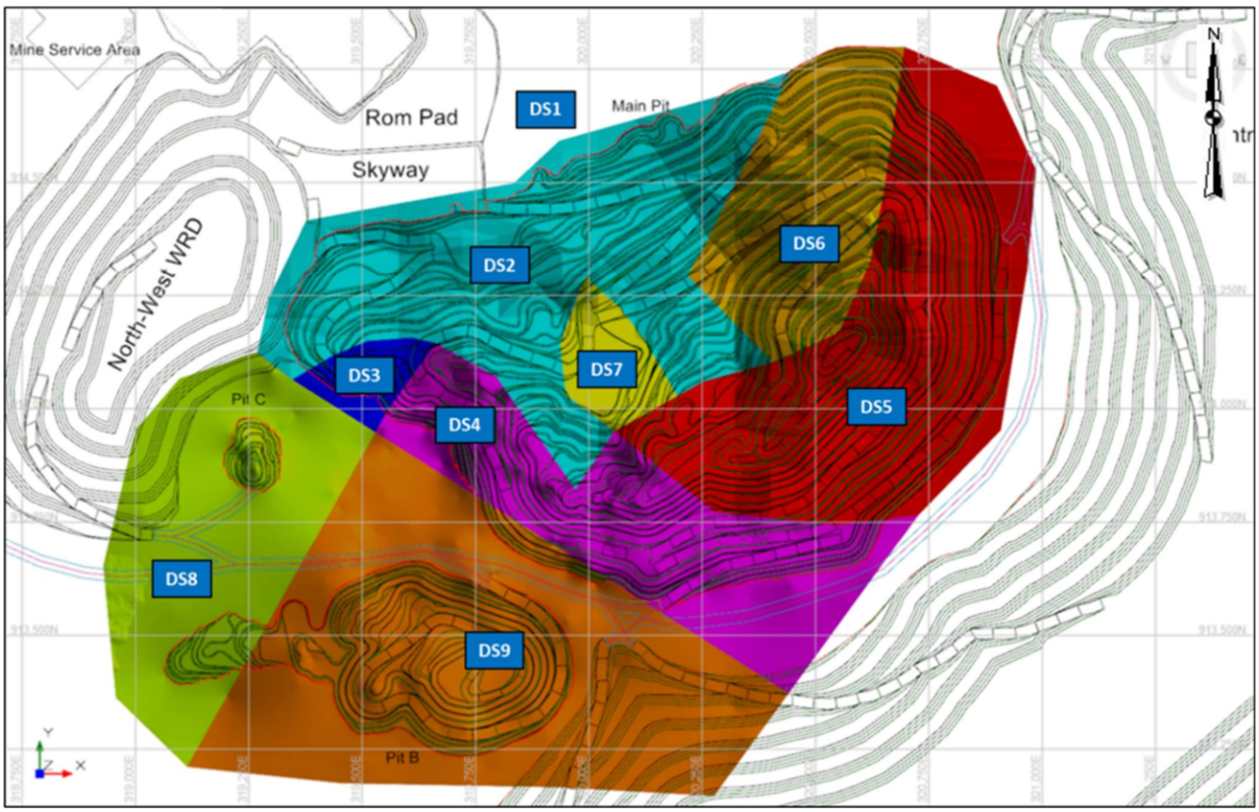


Figure 15-3: Bastion Geotechnical Pit Geotechnical Design Sectors (DS) (SRK,2022)

15.3.3 Re-blocking, Dilution and Ore Loss

Table 15-2, following, summarises the effective ore dilution and metal losses resulting from the re-blocking of the resource block models to generate the mining block models used to support the mine planning and scheduling process and the reporting of Ore Reserves. The estimates reported in this table are derived by applying the relevant deposit-specific cut-off grades based on Marginal Ore ('MO') assumptions to each block model within the final pit designs.

The resource block model was sub-blocked with the parent block size and smallest sub-block sizes 20 m x 20 m x 10 m and 2.5 m x 2.5 m x 1.25 m along the X-direction, Y-direction, and Z-direction, respectively. In mine planning, the optimisation process assumes that the smallest size sub-blocks can be selectively mined and processed; however, it is impossible to mine small sub-blocks selectively without causing dilution in mining ore blocks with the large excavators envisioned for operation at Lafigué, such as a Komatsu PC2000 or PC3000, with bucket sizes of 2.8 m and 3.0 m respectively. Therefore, as discussed in this section, the re-blocking process is seen as an industry best practise and one of the most elegant ways of applying proper dilution to the resource models containing too small a block size to be mined selectively.

The resource model was re-blocked to a selective mining unit ('SMU'), size 5 m x 5 m x 2.5 m along the X-direction, Y-direction, and Z-direction respectively, to create regularised mine planning models. The SMU size was based on ore loading units corresponding in size to a PC2000 and smaller. Therefore, if larger excavators are to be used, the SMU size will need to increase.

During the re-blocking process, smaller size blocks were added together to form an SMU size block. If some of these small blocks are ore and others are waste containing no gold, the resultant block would have a lower grade than the grade of the smaller ore blocks due to the addition of the waste. Similarly, if higher grade blocks are merged with neighbouring lower grade blocks, the overall grade would be the same as the average grade of the blocks if they all had the same density.

The effects of the re-blocking process for Lafigué deposit as percentages of change on the Mineral Reserve ore tonnes, gold content and grade above the set cut-off grades through conversion of the resource model with sub-blocks to a regularised re-blocked model are shown in Table 15-2 and Figure 15-4. In the estimation of the Mineral Reserves, the cut-off grades vary by weathering type 0.4 g/t Au for oxide and transition and 0.5 g/t Au and fresh ore-forming the majority of the deposit. An average approximate line was used in the graph to demonstrate the impact of the re-blocking process.

Figure 15-4, following, shows that the impact on tonnes and gold content increases as the cut-off grade increases. Around the marginal ore cut-off grade, shown as a dotted line on the graph, the ore tonnage increased 8.5% compared to the ore tonnes at the same cut-off grade within the resource model. Due to dilution, the gold content of the pit decreased by 1.5% and the overall average grade reduced by 7.8%.

Table 15-3 summarised the potential percentages change in the Mineral Reserve ore tonnes, gold content and grade above the set CoG through conversion of the resource model to various dimensions on a horizontal and vertical plain. The data shows that on a horizontal plain, the effect on tonnes and grade is slightly higher than on the vertical, but for both horizontal and vertical, the loss in content increases significantly at 0.5 g/t Au CoG but is limited at 0.4 g/t Au CoG.

Table 15-2: Modifying Factors Associated with the Re-blocking Process

Cut-Off Grade	(g/t Au)	0.4	0.5	0.6	0.8	1.0
In Situ Tonnage	(Mt)	68.0	58.8	51.9	41.6	34.5
	(g/t Au)	1.7	1.9	2.1	2.4	2.7
	(koz Au)	3700	3567	3447	3217	3014
Diluted Tonnage	(Mt)	74.5	62.9	54.4	42.4	34.5
	(g/t Au)	1.5	1.7	1.9	2.3	2.6
	(koz Au)	3682	3513	3365	3097	2871
Dilution	(%)	110.0	108.5	107.2	105.7	104.7
Recovery	(%)	99.5	98.5	97.6	96.3	95.3
or						
Tonnes	(%)	10.0	8.5	7.2	5.7	4.7
Grade	(%)	-9.1	-7.8	-6.7	-5.4	-4.5
Content	(%)	-0.5	-1.5	-2.4	-3.7	-4.7

Table 15-2 notes: Unconstrained Block Model

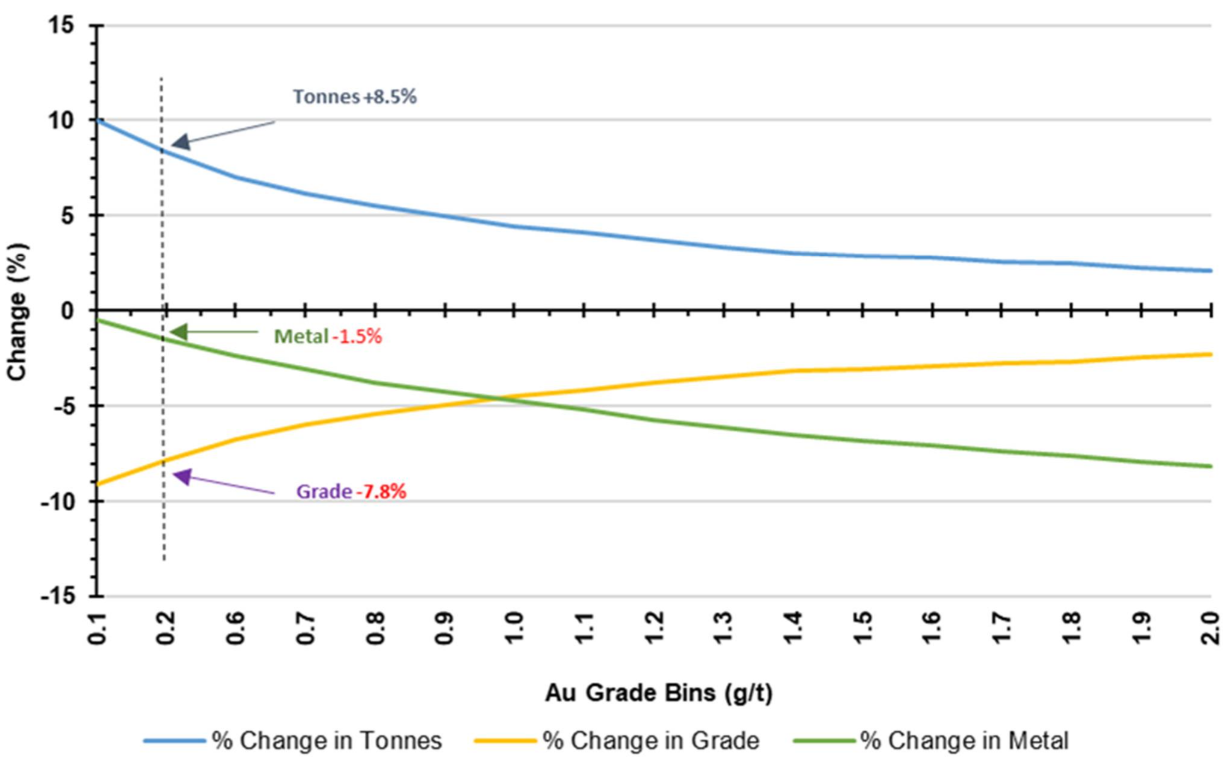


Figure 15-4: Effects of Re-blocking

Figure 15-5 illustrates the effects of the re-blocking process as a percentage in variation between the original model and re-blocked models within the pit, based on; tonnes, gold, and grade of indicated material as a function of cut-off grade for the Lafigué Deposit.

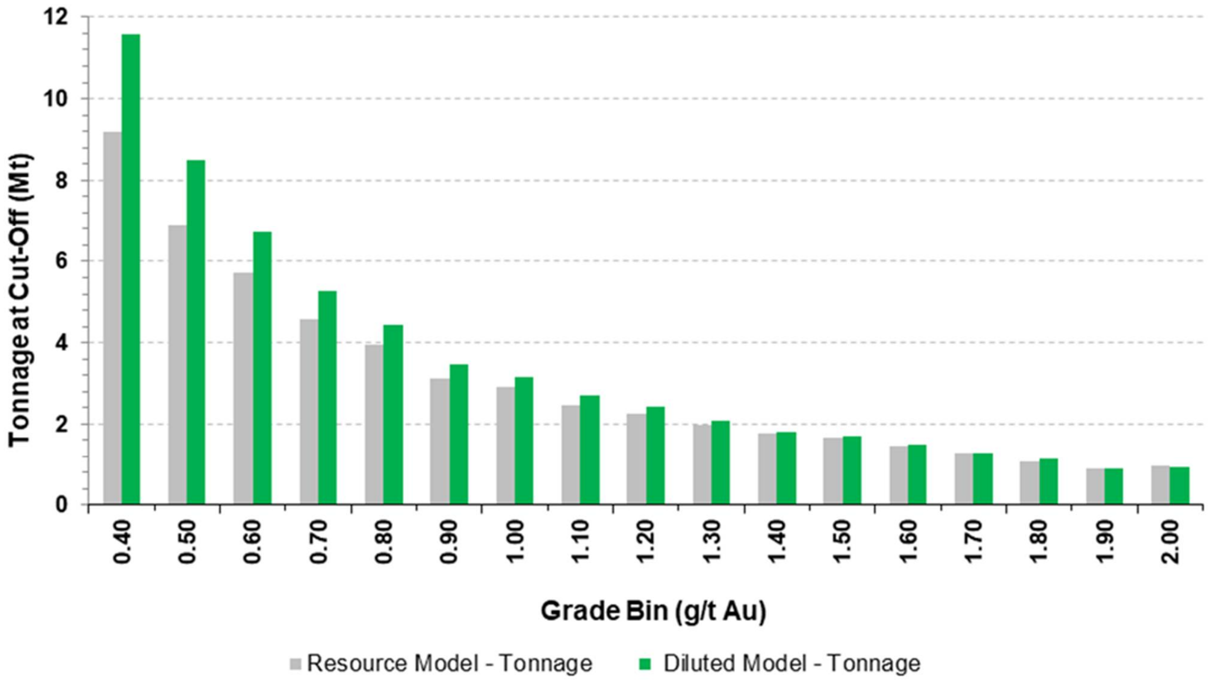


Figure 15-5: Diluted Model Grade Histogram

Table 15-3: Change in the Mineral Reserve at various SMU block sizes

Block Size (xyz)	(m)	5x5x2.5	5x5x5	10x10x2.5	10x10x5	10x10x10	20x20x2.5	20x20x5	20x20x10
0.5 g/t Au CoG									
Tonnes	(%)	8.5	9.14	9.72	11.78	17.30	17.37	18.20	21.94
Grade	(%)	-7.8	-8.37	-8.86	-10.53	-14.75	-14.80	-15.40	-17.99
Content	(%)	-1.5	-3.04	-3.25	-4.20	-6.11	-5.72	-6.36	-8.52
0.4 g/t Au CoG									
Tonnes	(%)	10.0	11.39	12.12	14.96	21.80	21.37	22.67	27.20
Grade	(%)	-9.1	-10.23	-10.81	-13.02	-17.90	-17.61	-18.48	-21.38
Content	(%)	-0.5	-1.87	-2.03	-2.63	-3.99	-3.69	-4.14	-5.86

On the basis that that the Lafigué deposit has not been mined historically, an additional 5% dilution was added on a block-by-block basis. The additional factor is applied in addition to the modifying factors incurred during the regularisation process. The 5% dilution was only applied during scheduling and not incorporated during the pit optimisation.

Mineralogy and by association higher and lower gold grades in the quartz vein versus rock/shear-hosted mineralisation, may be visually discernible at the Lafigué deposit. However, the relationship did not prove to be consistent within the drill core. In addition, the ore/waste contacts will be challenging to visually distinguish, as the deposit incorporates diffuse packages of mineralisation where the grade slowly drops off. Good grade and ore control practises will be required that is supported by internal and external training on dig polygon design and adherence to acceptable dilution levels.

Total applied modifying factors, including factors associated with re-blocking, artisanal mining and the additional 5% dilution on a block-by-block basis, are summarised in Table 15-4 following.

Table 15-4: Total applied Modifying Factors, including additional block-by-block dilution

Cut-Off	(g/t Au)	0.4	0.5	0.6	0.8	1.0
Dilution	(%)	112.7	110.8	109.3	107.8	107.1
Recovery	(%)	93.8	92.6	91.5	90.0	89.0
or						
Tonnes	(%)	12.7	10.8	9.3	7.8	7.1
Grade	(%)	-11.3	-9.8	-8.5	-7.2	-6.6
Content	(%)	-6.2	-7.4	-8.5	-10.0	-11.0

Table 15-4 notes: Total applied Modifying Factors on 5 m x5 m x2.5 m (x,y,z) model (Unconstrained)

15.3.4 Gold Price and Revenue Related Assumptions

For the calculation of cut-off grades and supporting the economic viability of the Mineral Reserves, a gold price of USD 1300/oz was used. The other factors that derive the net sales revenue are the individual government royalty of 4%[83] with an additional community levy of 0.5%, and a transport, vaulting and refining cost of USD 4.0/oz payable. This results in a net gold price of USD 1237/oz payable (USD 39.79/g), and a total selling cost of USD 62.5/oz.

15.3.5 Mining Cost

The mining costs were derived from the inputs used in the PFS study as the best source. At the same time, contractors were approached to provide updated mining costs for the Lafigué Project, including equipment and labour numbers. The PFS cost was based on actual operating data from similar mines in the area. The costs were based on contractor mining operations and inflated by 5%. The resulting mining costs address all key mining-related activities: drilling and blasting, excavation, load and haul, ancillary support, dewatering, grade control, stockpile re-handling and ore over-haul (Off RoM re-handle) costs. An additional provision was made for Rehabilitation at USD 0.06/t of rock. Subsequent comparisons to preliminary contractor quotes in January 2022, showed little difference in the overall mining unit rate.

The Off RoM re-handle cost of USD 0.79/t of ore was not applied in the pit optimisation as the assumption was that this cost is associated with sub-ore stockpiles a distance from the pit. Sub-ore will not be classified as ore in the pit optimisation, as it will be at a lower CoG and thus, will unnecessarily penalise the total ore cost. This cost will be appropriately applied in the financial model.

[83] Corresponding to a gold price of> USD 1300/ozt and ≤ USD 1600/ozt;

Table 15-5: Mining Operation Cost Assumptions for Pit Optimisation

Description	Units	Value
Reference Level	(Z Elevation)	350.00
Laterite (LT)	(Z Elevation)	380.00
Saprolite (SP)	(Z Elevation)	370.00
Transitional (TR)	(Z Elevation)	310.00
Fresh (FR)	(Z Elevation)	210.00
Waste Mining Cost (LT)	(USD/t_{rock})	1.78
Waste Mining Cost (SP)	(USD/t_{rock})	1.81
Waste Mining Cost (TR)	(USD/t_{rock})	1.96
Waste Mining Cost (FR)	(USD/t_{rock})	2.75
Wavg Waste Mining Cost	(USD/t_{rock})	2.65
Incremental Mining Cost	(USD/m bench)	0.0022
Ore Mining Cost		
Ore/Waste Differential (LT)	(USD/t_{ore})	0.16
Ore/Waste Differential (SP)	(USD/t_{ore})	0.09
Ore/Waste Differential (TR)	(USD/t_{ore})	0.24
Ore/Waste Differential (FR)	(USD/t_{ore})	0.83
Wavg Ore Mining Cost	(USD/t_{ore})	0.81
Grade Control (LT)	(USD/t_{ore})	1.31
Grade Control (SP)	(USD/t_{ore})	1.31
Grade Control (TR)	(USD/t_{ore})	1.31
Grade Control (FR)	(USD/t_{ore})	1.31
Rehandle Cost (LT)	(USD/t_{ore})	0.37
Rehandle Cost (SP)	(USD/t_{ore})	0.37
Rehandle Cost (TR)	(USD/t_{ore})	0.37
Rehandle Cost (FR)	(USD/t_{ore})	0.37
Off RoM Rehandle	(USD/t_{ore})	0.79
Rehabilitation Cost	(USD/t_{rock})	0.06

15.3.6 Processing Cost and Recoveries

The processing cost and Recoveries summarised in Table 15-6 were revised in May 2021 by Lycopodium, based on updated throughput rates and cost information available at the time. As a result, these values may differ from the latest cost estimates in the final financial model, with impacts evaluated in the sensitivity analysis.

Process recoveries were revised based on updated laboratory test work done during 2021. The recovery formula applied was:

Equation 15-1: $\boldsymbol{Au\ Rec\ \% = 0.769\ x\ LN(\ Head\ Grade\ ) +\ 97}$

The use of Equation 15-1 is subject to the following constraints:

- Above a 34 g/t Au head grade, a constant maximum recovery of 99.7% was applied;
- Below a 0.2 g/t Au head grade, a linear relationship applies Au% Rec = 68* Head grade+82.2;
- 0.5% should be deducted from the above recoveries to account for soluble gold loss; and
- Guidance from Endeavour was to subtract an additional 2% to limit the recovery from exceeding 97%.

The formula applied in the Whittle Pit Optimisation Software is illustrated in Equation 15-2 following:

Equation 15-2: if(au.G>=34,99.7,IF(au.G<=0.2,(68* (Head Grade) +82.2),0.769*(LOG(Head Grade)/LOG(2.71828))+97)))/100-0.025)

* Whittle does not contain natural log function and was thus converted to a Log function.

Table 15-6: Processing cost and Recoveries

Description	Units	Value
Throughput Rate	(Mt/a db)	4.00
Recovery Au		
• Laterite (LT)	(%)	94.87
• Saprolite (SP)	(%)	94.87
• Transitional (TR)	(%)	94.92
• Fresh (FR)	(%)	95.08
Processing Cost		
Fixed		
• CIL (LT)	(USD/t_{ore})	3.41
• CIL (SP)	(USD/t_{ore})	3.41
• CIL (TR)	(USD/t_{ore})	3.41
• CIL (FR)	(USD/t_{ore})	4.06
Variable		
• CIL (LT)	(USD/t_{ore})	4.06
• CIL (SP)	(USD/t_{ore})	4.06
• CIL (TR)	(USD/t_{ore})	4.06
• CIL (FR)	(USD/t_{ore})	5.07

15.3.7 Other Parameters

Other costs included in the pit optimisation not mentioned previously are:

- General and Administration:USD 5.60/t_{ore}; and,
- Sustaining Capital:USD 1.87/t_{ore}

Except for Sustaining Capital, which was applied as part of the processing unit cost, no capital or taxation costs were included.

15.3.8 Inputs and Parameters Summary

Table 15-7, following, summarises the optimisation input parameters as discussed in the previous section.

Table 15-7: Pit Optimisation Input Assumptions Summary

Parameters	Units	Value	Source/Basis
Production Rate - Ore	(Mt/a)	4.00	
Geotechnical			
• Max Surface Elevation	(Z Elevation)	420	
Laterite & Saprolite (Oxide) & Transition		IRA / OSA	
• DS 1	(Deg)	40.3 / 33.0	END-011-GD DRAFT Report.pdf
Fresh		IRA/OSA	
• Footwall Zone DS2	(Deg)	35.0/33.0	END-011-GD DRAFT Report.pdf
• Footwall Zone DS6	(Deg)	38.0/35.0	
• Footwall Zone DS7	(Deg)	52.0/48.0	
• Hanging Wall Zone DS3	(Deg)	55.0/51.0	
• Hanging Wall Zone DS4	(Deg)	55.0/51.0	
• Hanging Wall Zone DS5	(Deg)	52.0/48.0	
• Hanging Wall Zone DS8	(Deg)	52.0/48.0	Inputs extended from Zone 5
• Hanging Wall Zone DS9	(Deg)	52.0/48.0	Inputs extended from Zone 5
Mining Factors			
• Dilution	(%)	107%	From Regularisation 5x5x2.5
• Recovery	(%)	99%	
Processing			
• Recovery - Au (oxide)	(%)	94.87%	Lycopodium 2021
• Recovery - Au (transition)	(%)	94.92%	
• Recovery - Au (fresh)	(%)	95.08%	
Operating Costs			
• Wavg Waste Mining Cost	(USD/t_{rock})	2.65	PFS Financial Model
• Incremental Mining Cost	(USD/m bench)	0.0022	
• Reference Level	(Z Elevation)	350.00	
• Wavg Ore/Waste Differential	(USD/t_{ore})	0.81	
• Grade Control	(USD/t_{ore})	1.31	
• Rehandle Cost	(USD/t_{ore})	0.37	
• Off RoM Rehandle	(USD/t_{ore})	0.79	
• Rehabilitation Cost	(USD/t_{rock})	0.06	
• CIL - Oxide	(USD/t_{ore})	7.47	Lycopodium 2021
• CIL - Transition	(USD/t_{ore})	7.47	
• CIL - Fresh	(USD/t_{ore})	9.13	
• G&A	(USD/t_{ore})	5.60	PFS Financial Model

Table 15-7: Pit Optimisation Input Assumptions Summary

Parameters	Units	Value	Source/Basis
• Sustaining Capital (SIB)	(USD/t_{ore})	1.87	PFS Financial Model
• Selling Cost Au (@USD 1300/oz)	(USD/oz)	62.5	EDV 2021 Assumptions
Metal Price			
• Gold	(USD/oz)	1300	EDV 2021 Assumptions
• Gold	(USD/g)	41.80	
Discount Rate	(%)	5%	EDV 2021 Assumptions

15.3.9 Cut-off Grade analysis

The cut-off grades for the economic and Marginal ore by deposit and weathering types are given in Table 15-8, following. The cut-off grade analysis completed to support the end-2020 Mineral Reserve statements incorporated the various assumptions reported in Table 15-7, inclusive of:

- Processing operating expenditures.
- General and administration operating expenditures are assumed at 100% for the economic cut-off grade determination and 60% for the Marginal-Ore ('MO') grade category.
- Deposit-specific metallurgical recovery assumptions distinguishing between oxide, transitional and fresh ore.
- Transportation, vaulting, and refining charges expressed in USD/oz.
- Long-term gold price assumption of USD 1300/oz for reserves.
- Government royalty of 4% with an additional community levy of 0.5%.

Table 15-8: CoG for LG and MO by Weathering Type

Description	Units	Value
Economic Operating Costs (Oxide)	(USD/t_{ore})	16.78
Economic Operating Costs (Transition)	(USD/t_{ore})	16.71
Economic Operating Costs (Fresh)	(USD/t_{ore})	16.86
Low Grade (LG) Cut Off Grade		
• Economic Cut-Off Grade (Oxide)	(g/t Au)	0.4
• Economic Cut-Off Grade (Transition)	(g/t Au)	0.4
• Economic Cut-Off Grade (Fresh)	(g/t Au)	0.5
• In Situ Economic Cut-Off Grade (Oxide)	(g/t Au)	0.5
• In Situ Economic Cut-Off Grade (Transition)	(g/t Au)	0.5
• In Situ Economic Cut-Off Grade (Fresh)	(g/t Au)	0.6
Marginally Economic Ore (MO) Cut-off Grade		
• Marginal Cut-Off Grade (Oxide)	(g/t Au)	0.4
• Marginal Cut-Off Grade (Transition)	(g/t Au)	0.4
• Marginal Cut-Off Grade (Fresh)	(g/t Au)	0.4

15.4 Pit Optimisation

The objective of the open pit optimisation process is to determine a generalised open-pit shape (shell) that provides the highest economic value for a deposit. Pit optimisations were carried out using the Whittle Four-X (Whittle) pit optimisation software. Whittle software is considered the leader and widely used in the mining industry for open pit optimisation and consequently was selected for use in the Lafigué DFS.

For a given block model, cost, recovery and slope data, the Whittle software calculates a series of incremental pit shells, and within which each shell, is an optimum for a slightly higher commodity price factor. Then, the Lerchs Grossman (LG) algorithm determines the optimal pit shape at specific techno-economic and slope criteria.

The algorithm progressively constructs lists of related blocks that should or should not be mined. The final pit shell list defines a pit outline with the highest possible economic value, subject to the required pit slope angles. This outline includes every block that adds economic value when waste stripping is considered and excludes every block that does not add economic value. The process considers all revenues and costs and includes mining and processing parameters. The resulting pit shells are not necessarily practical and do not incorporate ramps, catchment berms etc. From an analysis of all the nested shells generated in the optimisation process, a single shell will be selected as the guide for a practical ultimate pit design. The Whittle pit shell results are used to assess the project's sensitivity to changes in the input parameters and to guide the pit design process.

The final pit design defines the ore reserve, and subsequently, the LoM production schedule/cashflows. Hence, pit optimisation is the first step in developing any LoM plan. In addition to defining the ultimate size of the open pit, the pit optimisation process also indicates possible mining pushbacks. These intermediate mining stages allow the pit to be developed practically and incrementally while at the same time targeting high-grade ore and deferring waste stripping.

Two optimisation scenarios were run, with the results discussed in this section:

- Measured and indicated classified material ('MI') case. Inferred classified material is treated as waste.

The MI case pit optimisation has been run based solely on the Measured and Indicated classified material to define the optimal pit shell and inventory and supports the estimation of Ore Reserves.

- Measured, Indicated, and Inferred classified material treated as ore ('MIF') case.

The MIF case pit optimisation has been used to evaluate if there is any inferred material in the near-term that might add value or be sterilised by the current MI pit. The MIF case also assists in better understanding the long-term potential.

15.4.1 Measured, Indicated (MI) Pit Optimisation Results

The calculation of a Whittle optimisation NPV, the usual criteria for selecting an optimal pit, largely depends on the discount rate and the high-level scheduling methodology applied in Whittle. Whittle then produces nested pit shells with a relative undiscounted cash flow ('CF') and discounted cash flow ('DCF') for each nested pit. Three relative DCF are presented based on three different scheduling methodologies applied by Whittle:

- Best: The best cash flow is achieved when the nested pit shells are mined in sequence. Although optimal for cash flow, such a sequence is mostly impractical since nested pit shells are often closely layered (like the layers of an onion) and would imply that thin pushbacks could be mined.

- Worst: The worst cash flow is achieved when the selected final pit shell is mined from top to bottom without consideration for nested pit shells or pushbacks. This approach would undoubtedly be practical but usually presents the lowest economic scheduling option.
- Specified: The specified case lies somewhere between the best and the worst. The user selects which cutbacks to mine to gain the advantages of higher cash flow of the Best Case, whilst still being practical and feasible.

With respect to Figure 15-6 to Figure 15-9, the following may be noted:

- Figure 15-6 shows the pit-by-pit metal price sensitivity results from the MI case pit optimisation.
- Figure 15-7 to Figure 15-9 shows the recovered ounces, ore tonnes and total tonnes, respectively, to cash flow and incremental cost per ounce.

Detailed pit optimisation results are provided in Table 15-9, whilst Table 15-10 summarises the selected pit shell results for each pushback and a USD 1300/oz shell. The Reserve Case pit optimisation has been used to determine the final pit extents for the LoM plan.

The ultimate pit shell selection was driven by maximising the pit inventory while considering the economic viability of the open pit, which considers the NPV in conjunction with the overall stripping ratio.

- The discounted open-pit value for each nested pit shows that the 'Best' and 'Specified' scheduling options, start to plateau at USD 1100/oz.
- The 'Specified' scheduling options trend is slightly lower than the 'Best' case, but significantly better than the 'Worst' case. This result indicates that Lafigué will benefit from a pushback-phased mining approach, but this diminishes after a USD 950/oz pit shell, where the 'Specified' case decreases relative to the 'Best' case.
- There are two distinct step changes at USD 725/oz and USD 875/oz, with minor step changes between USD 900/oz and USD 1000/oz. The pit starts plateauing between USD 1150/oz and USD 1300/oz.
- Substantial step changes within tiny gold price margins indicate that concentrated higher-grade zones drive the incremental pits generated by Whittle. This is better represented in Figure 15-13.
- Beyond USD 1150/oz on the graph, the Specified scheduling options show a stable but steady decreasing discounted cash flow to USD 1600/oz.

The USD 1175/oz pit shell was chosen as the final optimal pit shell for scheduling, which should deliver 44.6 Mt of ore (diluted) at a Whittle stripping ratio (excluding ramps and bench geometry) of 9.1:1 (t:t), and a Whittle cut-off grade 0.53 g/t Au for fresh ore. The total metal within the final pit shell is 2.66 Moz. The USD 1175/oz pit shell has a slightly higher specified NPV (USD 7M) compared to USD 1300/oz, but has 100 koz less gold.

The USD 1300/oz pit shell required waste mining to be brought forward from pushback 4 to pushback 1. For practical mining reasons, this increases the initial waste strip requirements and increases the cost.

Pushback shells were selected at each of the major step changes. Sensitivities were run to test the specified case NPV with different pushback combinations, whilst applying a 50 m minimum mining width. Limited to no benefit was realised by using six pushback shells (Shells: 725, 850, 875, 900, 925, 1175 and 1300) USD/oz to four (Shells: 725, 875, 900 and 1300) USD/oz, as shown in Figure 15-6 to Figure 15-9 and Table 15-9 and Table 15-10. Based on these results and from the strategic schedule discussed later within this schedule, pushback shells (725, 875, 900 and 1175) USD/oz were selected for pit designs.

- Pit shells (725, 850, 875, 900, 925,1175 and 1300) USD/oz - Whittle NPV USD 806 M;
- Pit shells (725, 850, 875, 900, 925 and 1175) USD/oz - Whittle NPV USD 805 M;
- Pit shells (725, 850, 875, 900 and 1175) USD/oz - Whittle NPV USD 803 M;
- Pit shells (725, 850, 875, 900 and 1300) USD/oz - Whittle NPV USD 803 M.

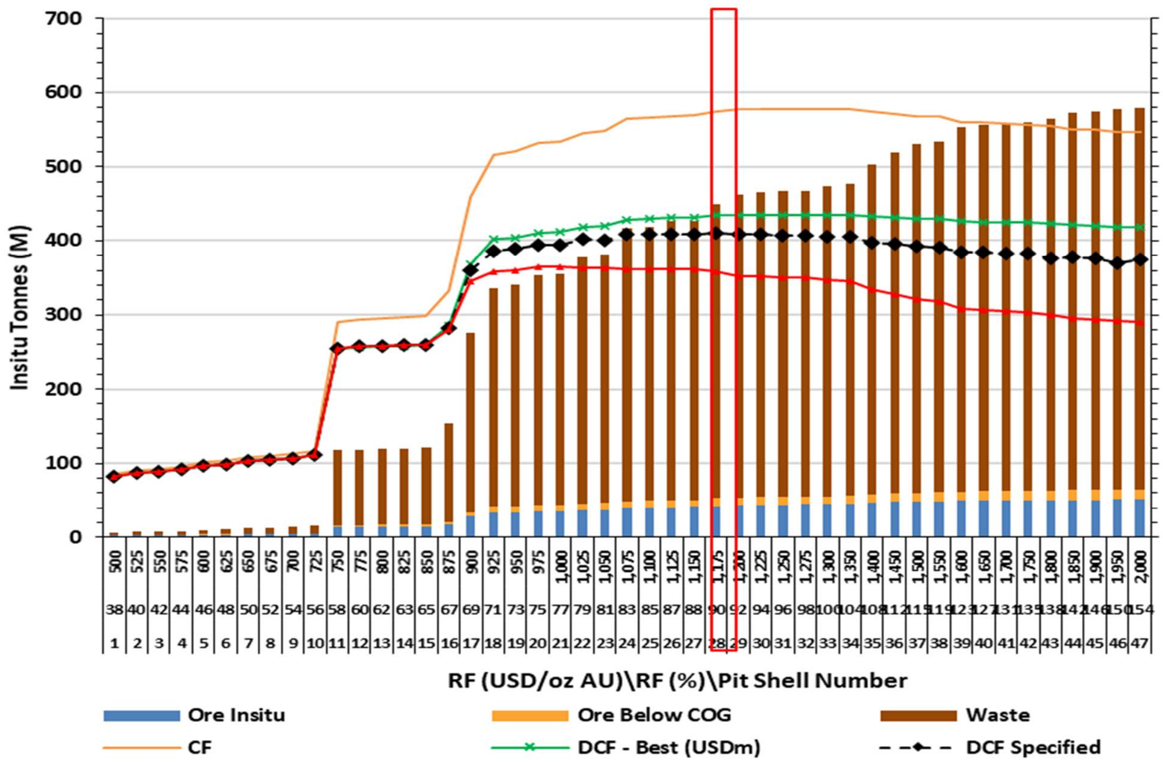


Figure 15-6: Reserve Case (MI) Metal Price Sensitivity

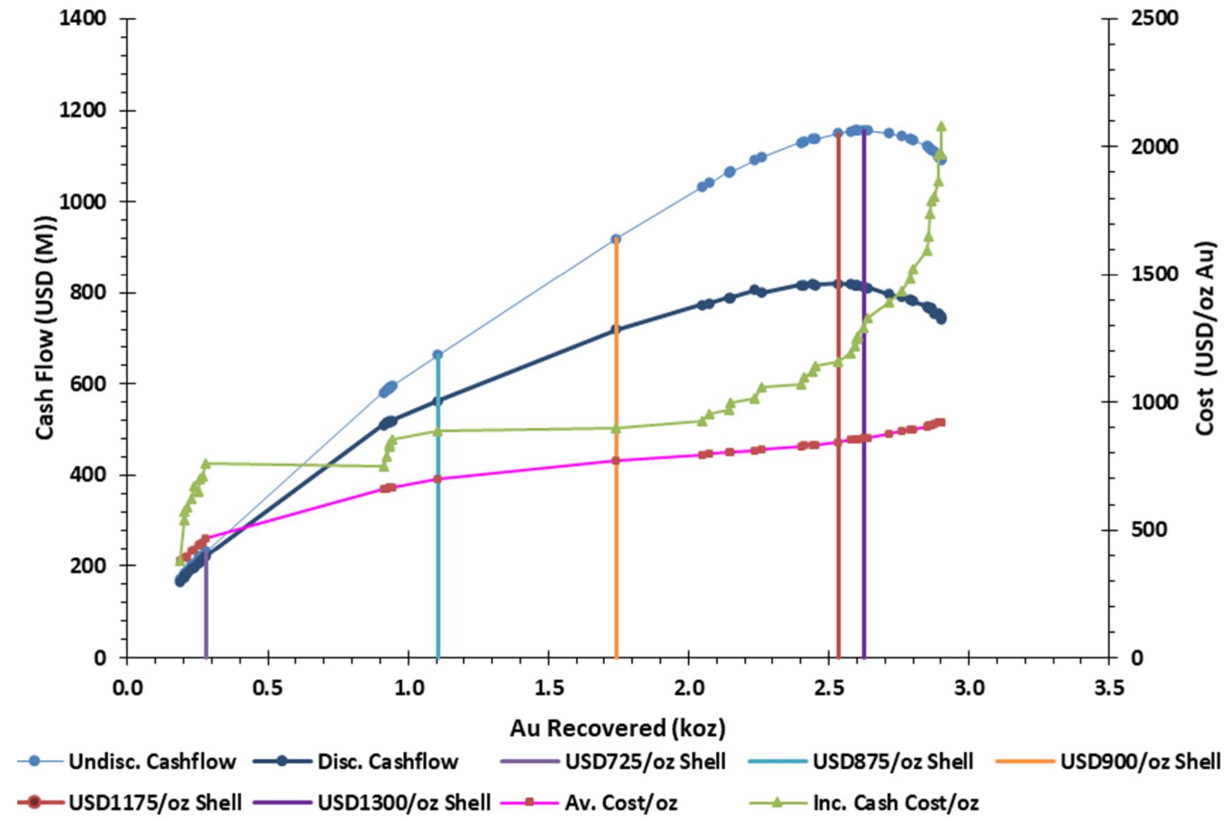


Figure 15-7: Reserve Case (MI) Recovered Oz in Relation to Cash Flow and Cost per oz

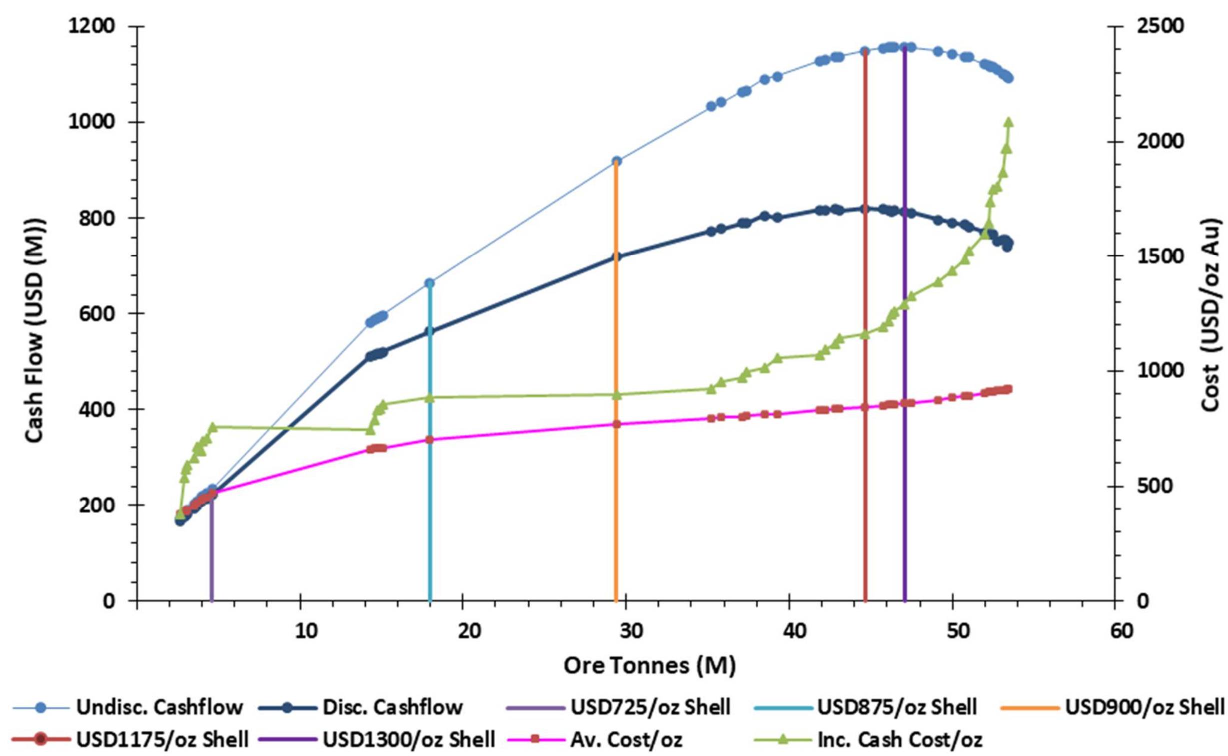


Figure 15-8: Reserve Case (MI) Ore Tonnes in Relation to Cash Flow and Cost per oz

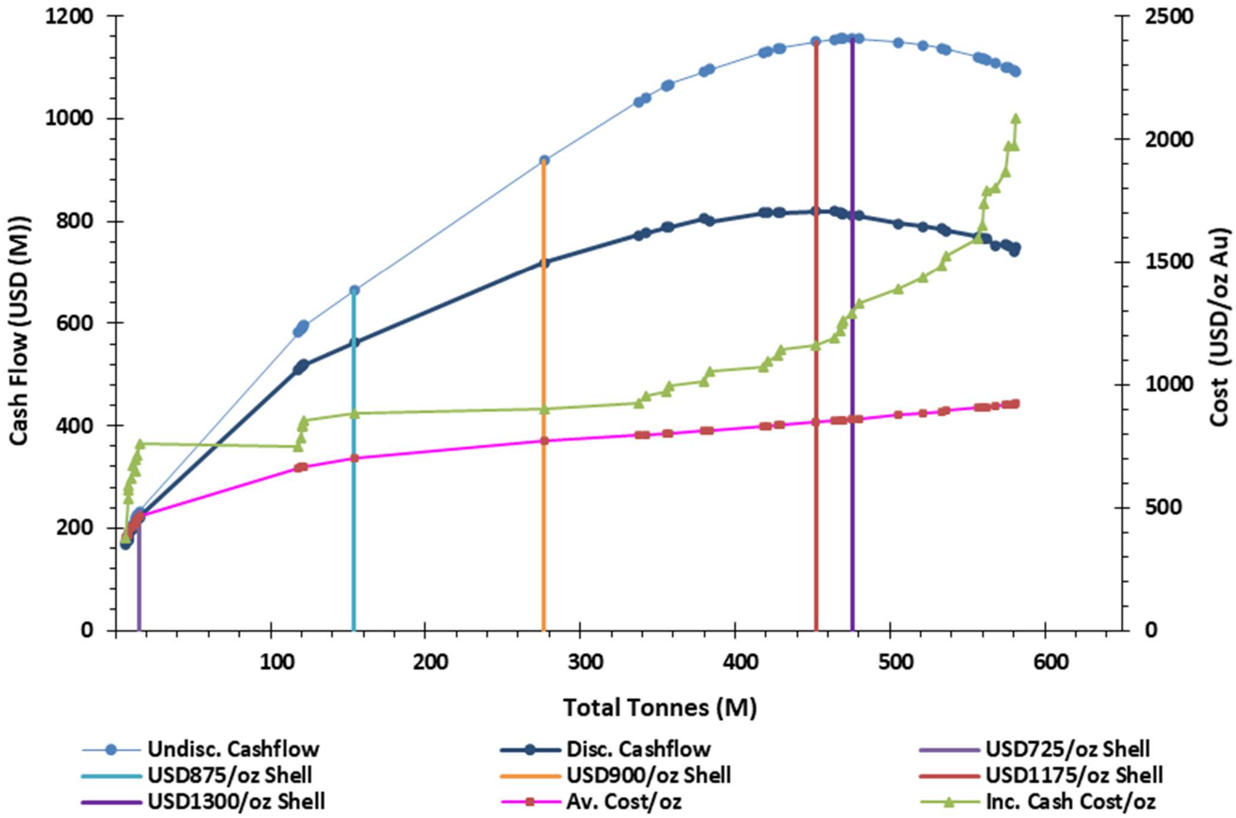


Figure 15-9: Reserve Case (MI) Total Tonnes in Relation to Cash Flow and Cost per oz

Table 15-9: Reserve Case (MI) Pit-by-Pit Results

Gold Price USD/oz	Total Rock (Mt)	Waste (Mt)	Ore (Mt)	Grade (g/t Au)	Metal (Moz Au)	SR (t:t)	CF (USD M)	DCF – Best (USD M)	DCF – Worst (USD M)	DCF – Specified (USDM)	Rec Metal ((oz Au)	Cash Cost (USD/oz)
600	10.13	6.65	3.48	2.15	0.24	1.91	202.44	194.32	194.32	194.32	0.23	416.71
625	10.54	6.86	3.67	2.10	0.25	1.87	206.77	198.04	198.04	198.04	0.24	424.22
650	12.56	8.64	3.92	2.10	0.26	2.21	217.09	207.36	207.36	207.36	0.25	438.52
675	13.03	8.95	4.07	2.07	0.27	2.20	220.67	210.39	210.39	210.39	0.26	444.49
700	13.93	9.65	4.28	2.04	0.28	2.26	225.90	215.08	215.07	215.07	0.27	453.41
725	15.25	10.62	4.63	1.98	0.29	2.30	233.07	221.86	221.79	221.79	0.28	467.97
750	118.05	103.73	14.32	2.08	0.96	7.25	582.11	508.29	510.72	510.72	0.91	662.34
775	119.39	104.85	14.54	2.07	0.97	7.21	587.33	511.92	514.43	514.43	0.92	663.71
800	120.17	105.40	14.77	2.06	0.98	7.13	591.10	514.50	516.79	516.79	0.93	665.13
825	120.59	105.72	14.88	2.05	0.98	7.10	592.82	515.92	517.84	517.84	0.93	665.81
850	121.65	106.59	15.06	2.04	0.99	7.08	596.05	518.57	519.77	519.77	0.94	667.25
875	154.19	136.21	17.98	2.01	1.16	7.58	663.85	572.32	560.93	562.76	1.11	699.88
900	276.64	247.24	29.40	1.93	1.83	8.41	918.44	735.20	690.23	718.73	1.74	772.91

Table 15-9: Reserve Case (MI) Pit-by-Pit Results

Gold Price USD/oz	Total Rock (Mt)	Waste (Mt)	Ore (Mt)	Grade (g/t Au)	Metal (Moz Au)	SR (t:t)	CF (USD M)	DCF – Best (USD M)	DCF – Worst (USD M)	DCF – Specified (USDM)	Rec Metal ((oz Au)	Cash Cost (USD/oz)
925	337.45	302.24	35.22	1.90	2.15	8.58	1032.63	803.63	718.00	773.37	2.05	795.77
950	342.24	306.39	35.86	1.89	2.18	8.54	1042.09	808.73	720.89	777.22	2.08	797.89
975	356.02	318.92	37.11	1.89	2.25	8.59	1064.36	821.86	728.93	789.06	2.14	803.32
1000	357.41	320.05	37.36	1.88	2.26	8.57	1066.96	823.40	729.67	788.62	2.15	804.09
1025	379.88	341.40	38.48	1.90	2.35	8.87	1091.01	837.57	727.87	805.50	2.24	811.94
1050	383.94	344.70	39.24	1.88	2.37	8.78	1096.92	840.82	728.42	800.41	2.26	814.58
1075	418.34	376.46	41.88	1.87	2.52	8.99	1129.10	857.80	724.40	817.02	2.40	829.63
1100	420.95	378.72	42.23	1.87	2.53	8.97	1131.75	859.13	724.30	817.28	2.41	831.09
1125	428.08	385.23	42.85	1.86	2.56	8.99	1137.08	861.82	723.34	818.08	2.44	834.53
1150	429.40	386.33	43.07	1.86	2.57	8.97	1138.17	862.28	723.11	816.37	2.45	835.39
1175	452.35	407.71	44.64	1.85	2.66	9.13	1150.00	868.34	716.19	819.50	2.53	846.17
1200	464.60	418.87	45.73	1.84	2.71	9.16	1154.96	870.72	705.61	819.05	2.58	852.21
1225	467.54	421.46	46.09	1.84	2.72	9.15	1155.96	871.17	703.48	817.54	2.59	853.96
1250	469.21	422.92	46.30	1.83	2.73	9.13	1156.34	871.34	702.22	814.63	2.60	855.03
1275	469.75	423.32	46.43	1.83	2.73	9.12	1156.46	871.35	701.56	815.18	2.60	855.53
1300	475.70	428.62	47.08	1.82	2.76	9.10	1156.64	871.29	694.92	812.45	2.62	859.26
1350	479.86	432.36	47.50	1.81	2.77	9.10	1156.19	870.92	691.41	811.03	2.64	861.91
1400	505.62	456.49	49.12	1.80	2.85	9.29	1149.56	867.05	667.45	796.05	2.71	876.47
1450	521.65	471.68	49.97	1.80	2.90	9.44	1143.55	863.82	653.76	790.33	2.76	885.35
1500	533.29	482.57	50.73	1.80	2.93	9.51	1137.60	860.67	641.78	786.01	2.79	892.27
1550	536.32	485.33	50.99	1.79	2.94	9.52	1135.61	859.60	637.29	781.80	2.80	894.26
1600	556.71	504.74	51.97	1.79	2.99	9.71	1121.18	852.05	616.27	770.38	2.85	906.29
1650	559.71	507.47	52.24	1.79	3.00	9.71	1118.30	850.58	611.76	767.82	2.86	908.43
1700	560.86	508.53	52.33	1.78	3.00	9.72	1117.04	849.95	610.34	766.46	2.86	909.27
1750	562.63	510.16	52.47	1.78	3.01	9.72	1114.96	848.91	607.70	765.44	2.86	910.57
1800	568.03	515.27	52.76	1.78	3.02	9.77	1109.22	846.03	600.88	752.94	2.87	914.11
1850	575.01	521.88	53.13	1.78	3.03	9.82	1101.45	842.17	589.88	755.66	2.89	918.63
1900	576.82	523.59	53.23	1.77	3.04	9.84	1099.03	841.01	587.34	753.52	2.89	919.94
1950	580.46	527.09	53.38	1.77	3.04	9.87	1094.57	838.82	582.74	741.26	2.90	922.35
2000	581.66	528.21	53.45	1.77	3.05	9.88	1092.80	837.93	580.80	749.44	2.90	923.26

Table 15-10: Reserve Case (MI) Select Pit Shells Optimisation Results

Lafigué FS	Units	USD 675/oz	USD 875/oz	USD 900/oz	USD 1175/oz	USD 1300/oz
In-situ Inventory	(Mt)	4.4	17.1	28.0	42.5	44.8
• Gold	(g/t)	2.07	2.11	2.03	1.95	1.91
• Gold	(koz)	294	1161	1829	2660	2756
Modifying Factors						
• Mining Dilution	(%)	From Regularisation 5x5x2.5 with additional 5% Dilution				
• Mining Recovery	(%)					
Diluted						
• Inventory	(Mt)	4.6	18.0	29.4	44.6	47.1
• Grade	(g/t Au)	1.98	2.01	1.93	1.85	1.82
• Contained Metal	(koz Au)	294	1161	1829	2660	2756
Quantities						
• Total Rock	(Mt)	15.3	154.2	276.6	452.4	475.7
• Mineral Inventory	(Mt)	4.6	18.0	29.4	44.6	47.1
• Waste + OM	(Mt)	10.6	136.2	247.2	407.7	428.6
• Stripping Ratio	(t:t)	2.3	7.6	8.4	9.1	9.1
Operating Expenditures						
• Mining	(USD/t mined)	2.90	2.71	2.75	2.78	2.78
• Rehabilitation Cost	(USD/t of ore)	0.06	0.06	0.06	0.06	0.06
• Processing CIL + G&A	(USD/t of ore)	14.80	15.45	15.63	15.71	15.71
• Au Selling Cost	(USD/oz)	62.50	62.50	62.50	62.50	62.50
• Total Cash Cost	(USD/oz)	2.90	2.71	2.75	846	2.78
Product						
• Au Metallurgical Recovery	(%)	95.25	95.32	95.28	95.26	95.25
• Recovered Metal	(koz Au)	280	1106	1743	2534	2625
Economic Summary						
• Metal Price	(USD/oz)	1300	1300	1300	1300	1300
• Revenue	(USD M)	364	1438	2265	3294	3412
• Mining Costs	(USD M)	44	418	762	1257	1323
• Processing Costs CIL	(USD M)	69	278	459	701	740
• Selling Costs	(USD M)	17.5	69.1	108.9	158.4	164.0
• Cashflow	(USD M)	233.1	663.9	918.4	1150.0	1156.6
• Discount Rate	(%)	5.0%	5.0%	5.0%	5.0%	5.0%
• Mill Rate	(Mt/a)	4.0	4.0	4.0	4.0	4.0
• DCF - Best Case	(USD M)	221.86	572.32	735.20	868.34	871.29
• DCF Specified	(USD M)	221.79	562.76	718.73	819.50	812.45
• DCF - Worst Case	(USD M)	221.79	560.93	690.23	716.19	694.92

Table 15-10: Reserve Case (MI) Select Pit Shells Optimisation Results

Lafigué FS	Units	USD 675/oz	USD 875/oz	USD 900/oz	USD 1175/oz	USD 1300/oz
• Project Life	(years)	1.4	5.4	8.9	13.5	14.3
Cut-Off Grade						
• OCOG - OPEX CIL	(USD/t_{ore})	24.34	38.71	41.54	43.88	43.81
• ECOG - OPEX CIL	(USD/t_{ore})	14.80	15.45	15.63	15.71	15.71
• OCOG CIL	(g/t Au)	0.6	1.0	1.1	1.2	1.2
• ECOG CIL	(g/t Au)	0.4	0.4	0.4	0.4	0.4
• ISOCOG CIL	(g/t Au)	0.7	1.2	1.2	1.3	1.3
• ISECOG CIL	(g/t Au)	0.4	0.5	0.5	0.5	0.5

Table 15-10 notes: OCOG = Operating Cut-Off Grade; ECOG = Economic Cut-Off Grade; ISOCOG = In situ Operating Cut-Off Grade; and ISECOG = In situ Economic Cut-Off Grade

With respect to Figure 15-10 to Figure 15-13, the following may be noted.

- Figure 15-10 shows the pit shells of the selected pushbacks in planview.
- Figure 15-11 and Figure 15-12 show the selected pit shells in section view at various locations, as illustrated in Figure 15-10.
- Figure 15-11 shows sections with the block model blocks coloured by gold grade and Figure 15-12, by ore classification.
- Figure 15-13 shows all the pit shells between USD 325/oz to USD 2000/oz in section view (Section A to E as illustrated in Figure 15-10). Again, pit shell growth is not uniform, and pit shells follow the high-grade nodes with limited incremental pits. Incremental pits were also analysed at USD 10/oz increments with no minimum mining width, and the results were similar.

Restricted incremental pit shells indicate that all the ore tonnes within a specific lens are required to sustain the cost of removing the waste tonnes above it. It is also clear that there are mainly four pushbacks clearly defined by Whittle in most areas of the deposit. Alternative incremental pushback phases might be obtained through software other than Whittle. Still, it should be noted that available ore tonnes and gold grade should be considered to balance gold production.

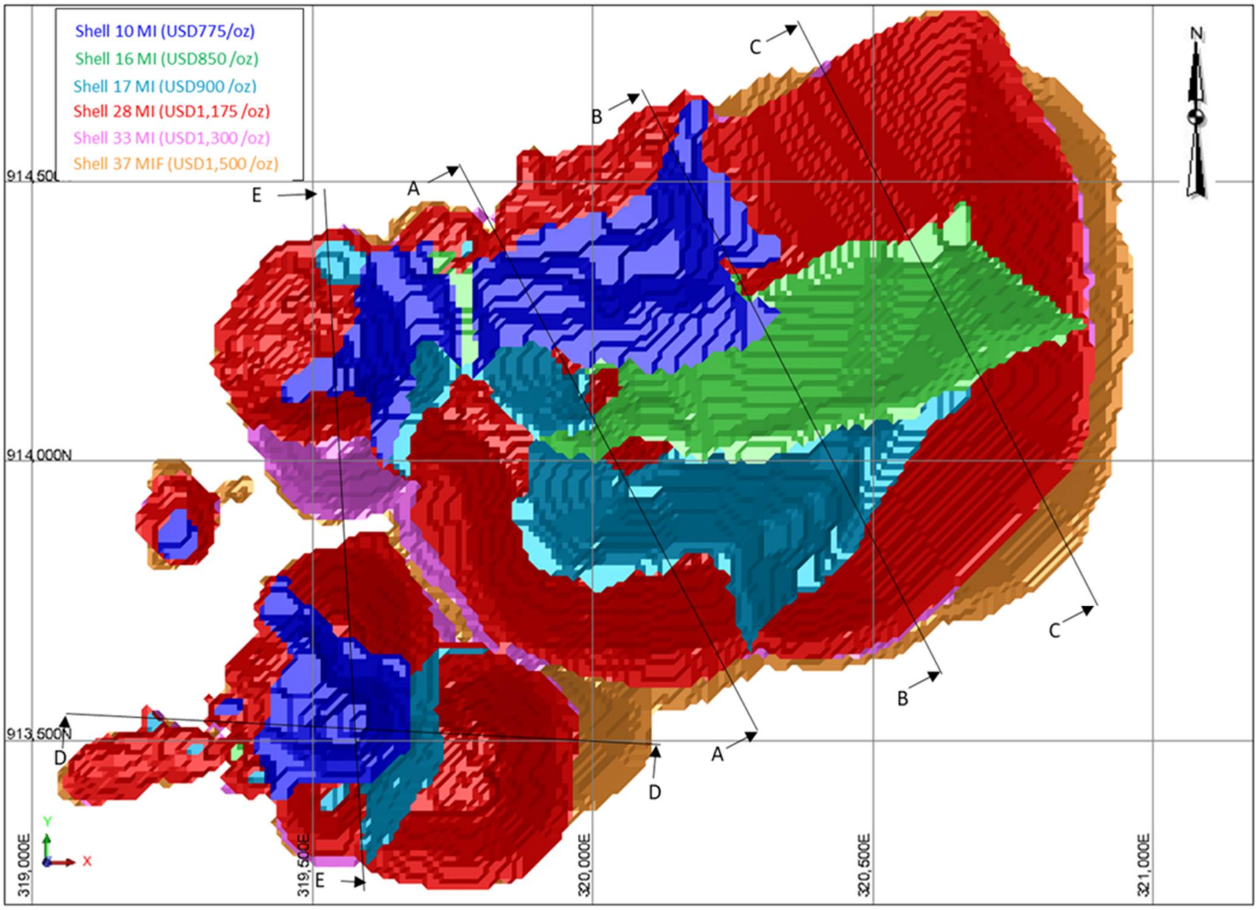


Figure 15-10: MI Pit Shells (Plan View) (SRK,2022)

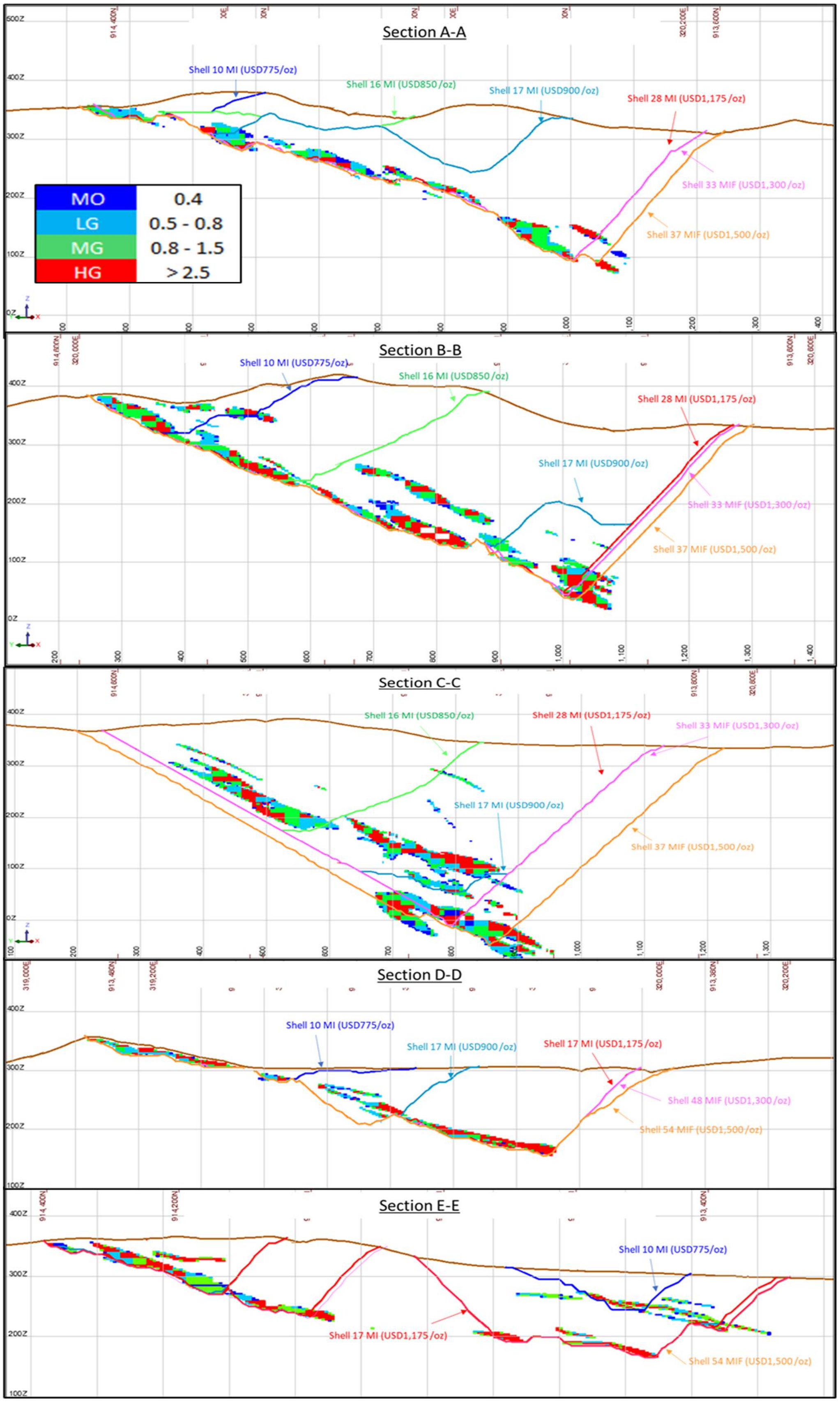


Figure 15-11: MI Pit Shells Sections with Block Model Coloured by Grade

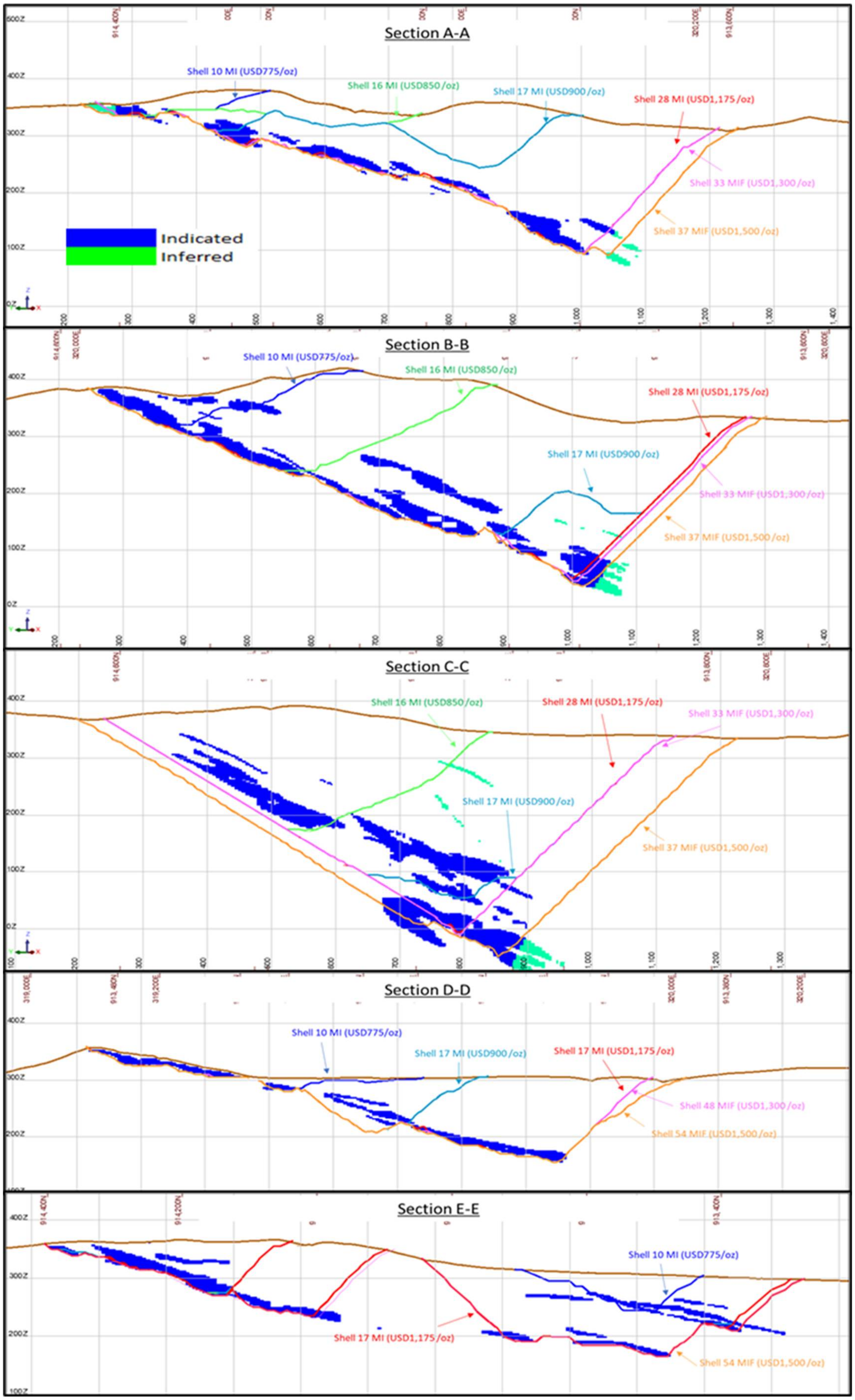


Figure 15-12: MI Pit Shells Sections Coloured by Classification

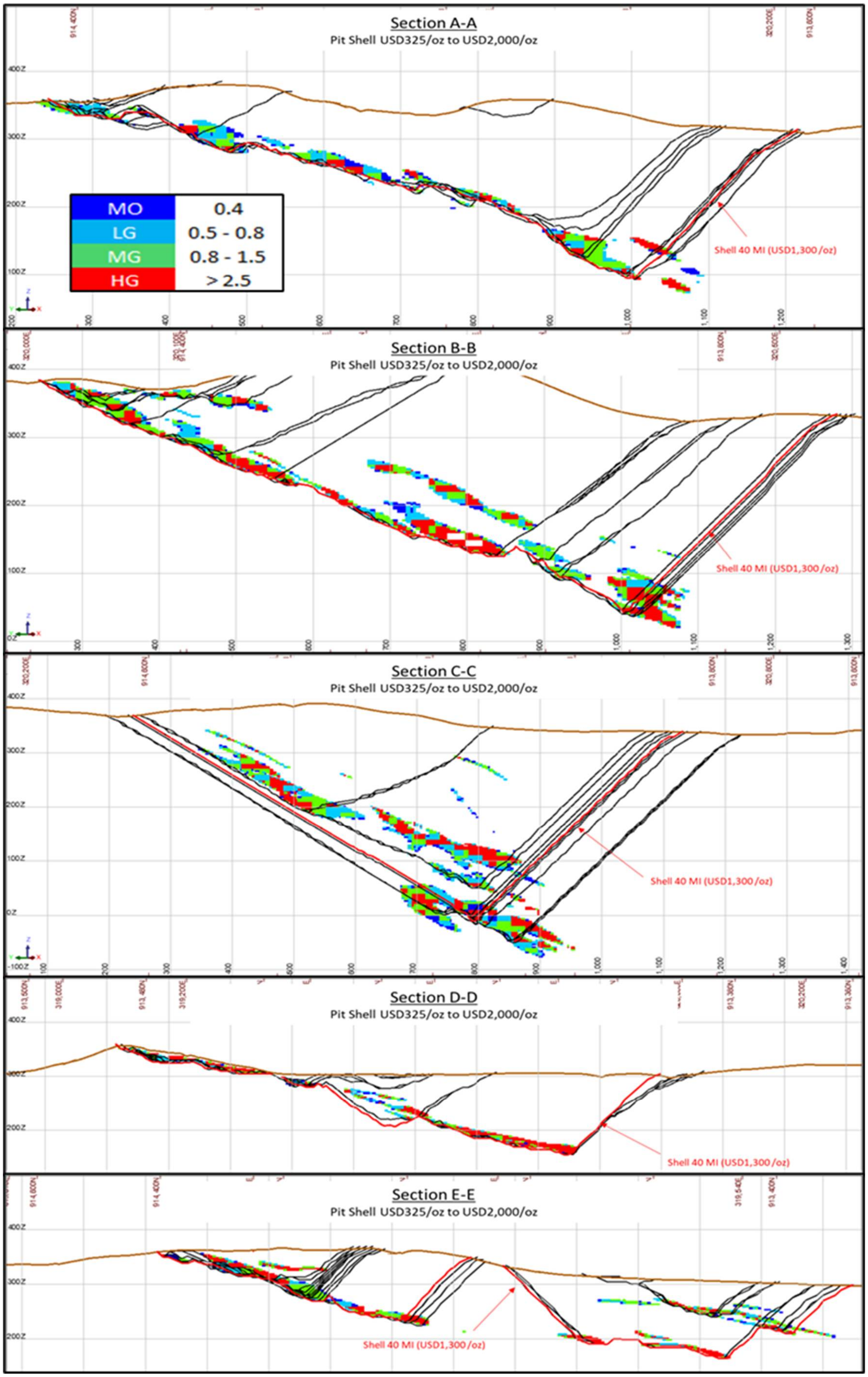


Figure 15-13: Pit Shell USD (325 to 2600)/oz in sections

15.4.2 Measured, Indicated and Inferred (MIF) Pit Optimisation Results

For the MIF case, inferred material was monetised with the Measured and Indicated to evaluate if there is any inferred material in the near term that might add value or be sterilised by the current MI pit. Figure 15-14 shows a plan view of the MI USD 1300/oz (red) pit shell to the MIF USD 1300/oz (blue) and USD 1500/oz (purple) pit shell, and Table 15-11 is a summary of inventories for each. The inferred inventories are mostly small thin lenses within the main pit at depth, expanding the pit highwall to the south.

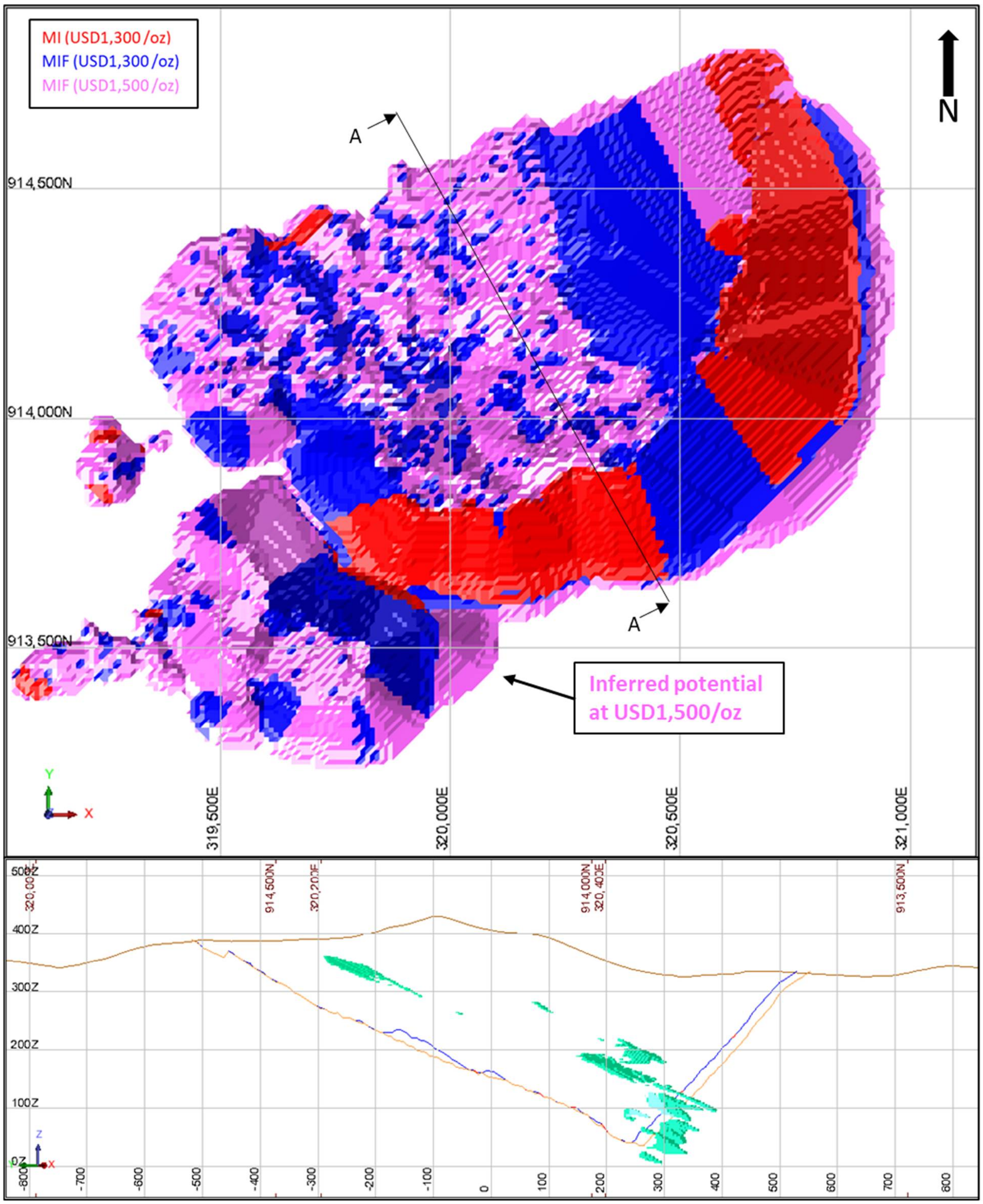


Figure 15-14: MI vs MIF Comparison

Table 15-11: MI vs MIF Comparison

Lafigué DFS Optimisation Results	**Units**	**USD 1300/oz MI**	**USD 1300 /oz MIF**	**USD 1500 /oz MIF**
Regularised 5x5x2.5				
• In-situ Inventory	(Mt)	44.8	46.7	54.0
• Gold grade	(g/t Au)	1.91	1.91	1.81
• Contained gold	(koz Au)	2756	2860	3136
Modifying Factors				
• Mining Dilution		From Regularisation 5x5x2.5 with additional 5% Dilution		
• Mining Recovery				
Diluted				
• Inventory	(Mt)	47.1	49.0	56.7
• Grade	(g/t Au)	1.82	1.81	1.72
• Contained Metal	(koz Au)	2756	2860	3136
Quantities				
• Total Rock	(Mt)	475.7	495.5	572.3
• Mineral Inventory	(Mt)	47.1	49.0	56.7
• Waste + OM	(Mt)	428.6	446.5	515.6
• Waste	(Mt)	418.0	434.4	505.0
• Inventory (Below Cut-off)	(Mt)	10.6	12.1	10.6
• Stripping Ratio	(t:t)	9.1	9.1	9.1
Operating Expenditures				
• Mining	(USD/t mined)	2.78	2.77	2.79
• Rehabilitation Cost	(USD/tore)	0.06	0.06	0.06
• Processing CIL + G&A	(USD/tore)	15.71	15.60	15.63
• Au Selling Cost	(USD/oz)	62.50	62.50	71.50
• Total Cash Cost	(USD/oz)	859	863	922
Production				
• Au Metallurgical Recovery	(%)	95.25	95.24	95.22
• Recovered Metal	(koz Au)	2625	2724	2986
LoM Economic Summary				
• Metal Price	(USD/oz)	1300	1300	1500
• Revenue	(USD M)	3412	3541	4479
• Mining Costs	(USD M)	1323	1368	1591
• Processing Costs CIL	(USD M)	740	765	886
• Selling Costs	(USD M)	164.0	170.3	213.5
• Cashflow	(USD M)	1156.6	0.0	0.0
• Discount Rate	(%)	5.0%	5.0%	5.0%

Table 15-11: MI vs MIF Comparison

Lafigué DFS Optimisation Results	Units	USD 1300/oz MI	USD 1300 /oz MIF	USD 1500 /oz MIF
• Mill Rate	(Mt/a db)	4.0	3.3	3.3
• DCF - Best Case	(USD M)	871.29	894.77	1,283.90
• DCF Specified	(USD M)	812.45	809.93	1,166.06
• DCF - Worst Case	(USD M)	694.92	707.53	1,003.98
• Project Life	(years)	11.8	14.9	17.2
Cut-Off Grade				
• OCOG - OPEX CIL	(USD/tore)	43.81	43.50	43.70
• ECOG - OPEX CIL	(USD/tore)	15.71	15.60	15.63
• OCOG CIL	(g/t Au)	1.2	1.1	1.0
• ECOG CIL	(g/t Au)	0.4	0.4	0.4
• ISOCOG CIL	(g/t Au)	1.3	1.3	1.1
• ISECOG CIL	(g/t Au)	0.5	0.4	0.4

15.4.3 Whittle Sensitivity analysis

Table 15-12 and Figure 15-15 to Figure 15-17, summarise the Whittle sensitivities for: Mining Cost, Processing Cost, Gold Price, and Slope Angle, run on ultimate shell of USD 1300/oz. The greener colours are the sensitivities with the most significant improvement on earnings before interest tax and amortisation excluding any capital cost (EBITA $_{\text{excl.CAPEX}}$) and the red colours with the highest decrease. The brighter the colour, the higher the sensitivity.

Table 15-12: Whittle Sensitivities

Sensitivity (%)	Total (Mt)	Waste (Mt)	Ore (Mt)	Grade (g/t)	Gold (Moz)	NPV (USD M)
Mining Cost						
• +15	452.3	407.8	44.5	2.0	2.66	652
• +10	452.3	407.8	44.5	2.0	2.66	703
• +5	452.3	407.7	44.6	1.9	2.66	753
• 0	452.3	407.7	44.6	1.9	2.66	804
• -5	452.3	407.7	44.7	1.9	2.66	855
• -10	452.3	407.6	44.7	1.9	2.66	905
• -15	452.3	407.6	44.7	1.9	2.66	956
Processing Cost						
• +15	452	409	43	1.99	2.64	766
• +10	452	409	44	1.98	2.64	779
• +5	452	408	44	1.96	2.65	793
• 0	452	408	45	1.95	2.66	804
• -5	452	407	45	1.93	2.67	816
• -10	452	407	45	1.92	2.67	829

Table 15-12: Whittle Sensitivities

Sensitivity (%)	Total (Mt)	Waste (Mt)	Ore (Mt)	Grade (g/t)	Gold (Moz)	NPV (USD M)
• -15	452	406	46	1.91	2.68	849
Gold Price						
• +15	452	405	47	1.87	2.69	1111
• +10	452	406	46	1.90	2.68	1029
• +5	452	407	45	1.92	2.67	917
• 0	452	408	45	1.95	2.66	804
• -5	452	409	44	1.98	2.64	691
• -10	452	410	43	2.02	2.63	577
• -15	452	411	41	2.06	2.60	470
Slope Angle						
• +15	430	383	47	1.96	2.81	939
• +10	431	385	46	1.95	28.84	615
• +5	443	397	46	1.94	2.74	865
• 0	452	408	45	1.95	2.66	804
• -5	454	410	44	1.95	2.64	762
• -10	444	401	43	1.94	2.53	765
• -15	452	410	42	1.94	2.49	729

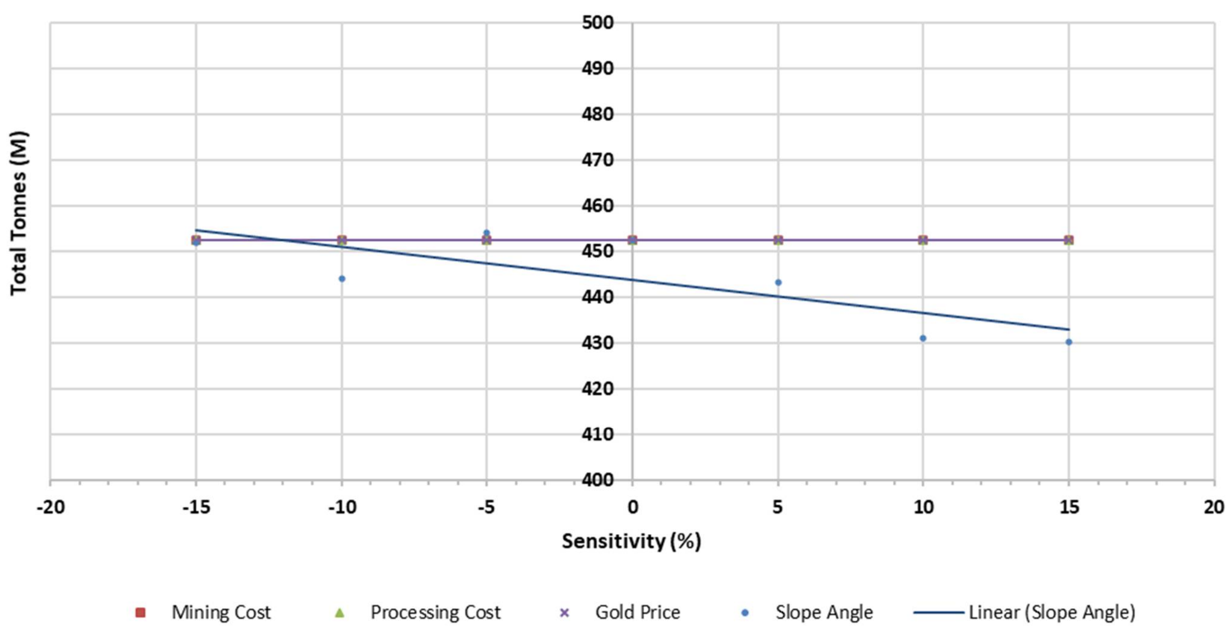


Figure 15-15: Sensitivity on Total Tonnes

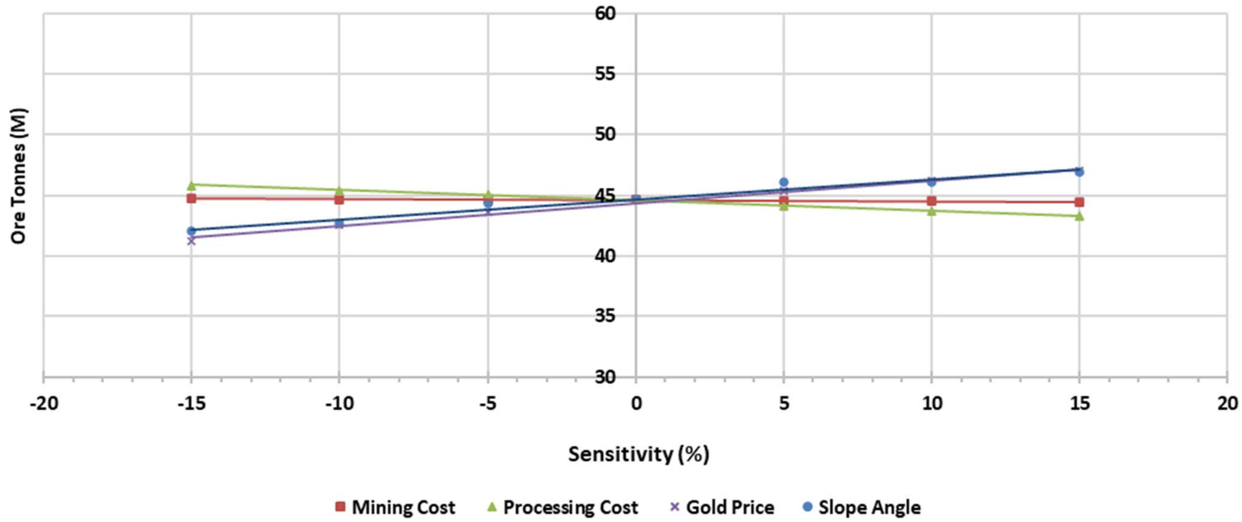


Figure 15-16: Sensitivity on Ore Tonnes

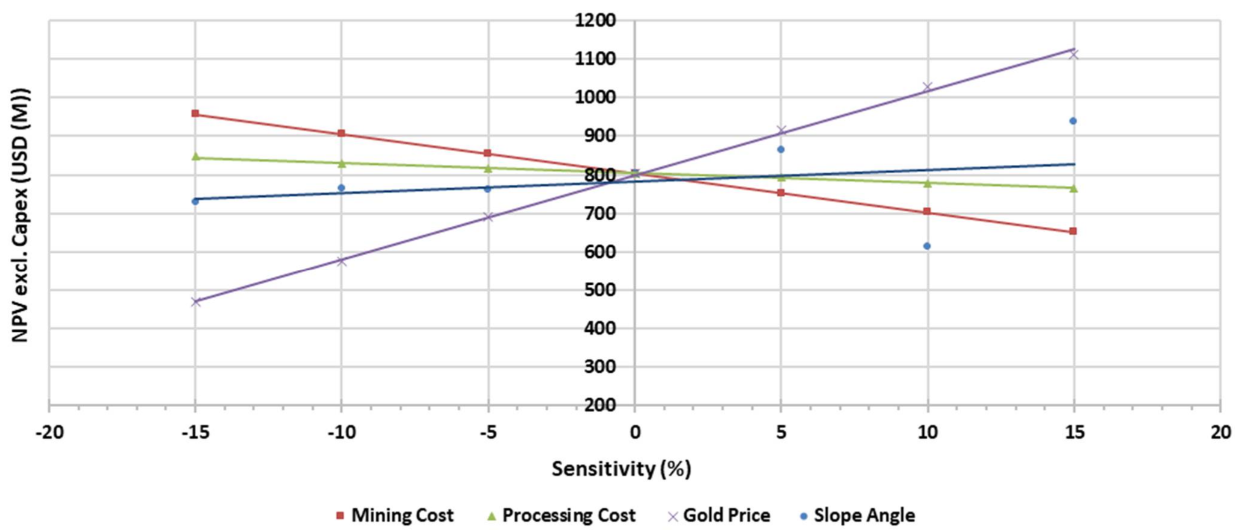


Figure 15-17: Sensitivity on NPV excl. Capex

15.4.4 Whittle Strategic Schedule

A strategic schedule was developed based on the selected pushback shells at (725, 875, 900) USD/oz and the final pit shell at USD 1175/oz. In addition, practical mining parameters were added to make the strategic schedule as achievable as possible. A minimum mining width of 50 m was applied with sink rates limited to 80 m per year.

Table 15-13 summarises the physicals from the strategic schedule, and Figure 15-18 the period face positions. Figure 15-18 shows that the Whittle strategic schedule reaches the final pit footprint in 2027 with main pit pushback 3 USD 900/oz and the last pit shell mining very close in sequence. Main pit pushback 3 stops mining at 140 mamsl and only starts going deeper when pushback 4 joins at 140 mamsl, as the pit progresses to the final depth.

Table 15-13: Whittle strategic Schedule

Periods	Total Tonnes	Total Waste	Process Throughput	Strip Ratio	Feed Grade	Gold Feed	Gold Produced	Cashflow	Disc. Cashflow
	(Mt)	(Mt)	(Mt)	(t:t)	(g/t)	(koz)	(koz)	(USD M)	(USD M)
2024	47.4	45.1	2.3	19.6	1.53	113.0	107.4	-18.1	-17.2
2025	55.0	51.0	4.0	12.8	1.59	204.2	194.1	32.5	29.5
2026	55.0	51.0	4.0	12.8	1.66	212.7	202.3	37.3	32.2
2027	55.0	51.0	4.0	12.8	1.73	221.3	210.6	43.9	36.1
2028	55.0	51.0	4.0	12.8	1.97	252.2	240.3	79.1	62.0
2029	55.0	51.0	4.0	12.8	2.31	297.0	283.4	128.8	96.1
2030	55.0	51.0	4.0	12.7	1.89	243.7	232.1	63.0	44.8
2031	29.9	25.9	4.0	6.5	2.26	290.5	277.1	183.0	123.8
2032	21.2	17.2	4.0	4.3	2.11	271.7	258.9	182.9	117.9
2033	13.6	9.6	4.0	2.4	2.12	272.3	259.6	204.3	125.4
2034	9.9	5.9	4.0	1.5	1.99	255.7	243.6	194.1	113.5
2035	0.4	0.1	0.3	0.5	2.83	25.0	23.9	22.9	13.3
Total	**452.3**	**409.8**	**42.5**	**9.6**	**1.95**	**2659.3**	**2533.2**	**1153.8**	**777.5**

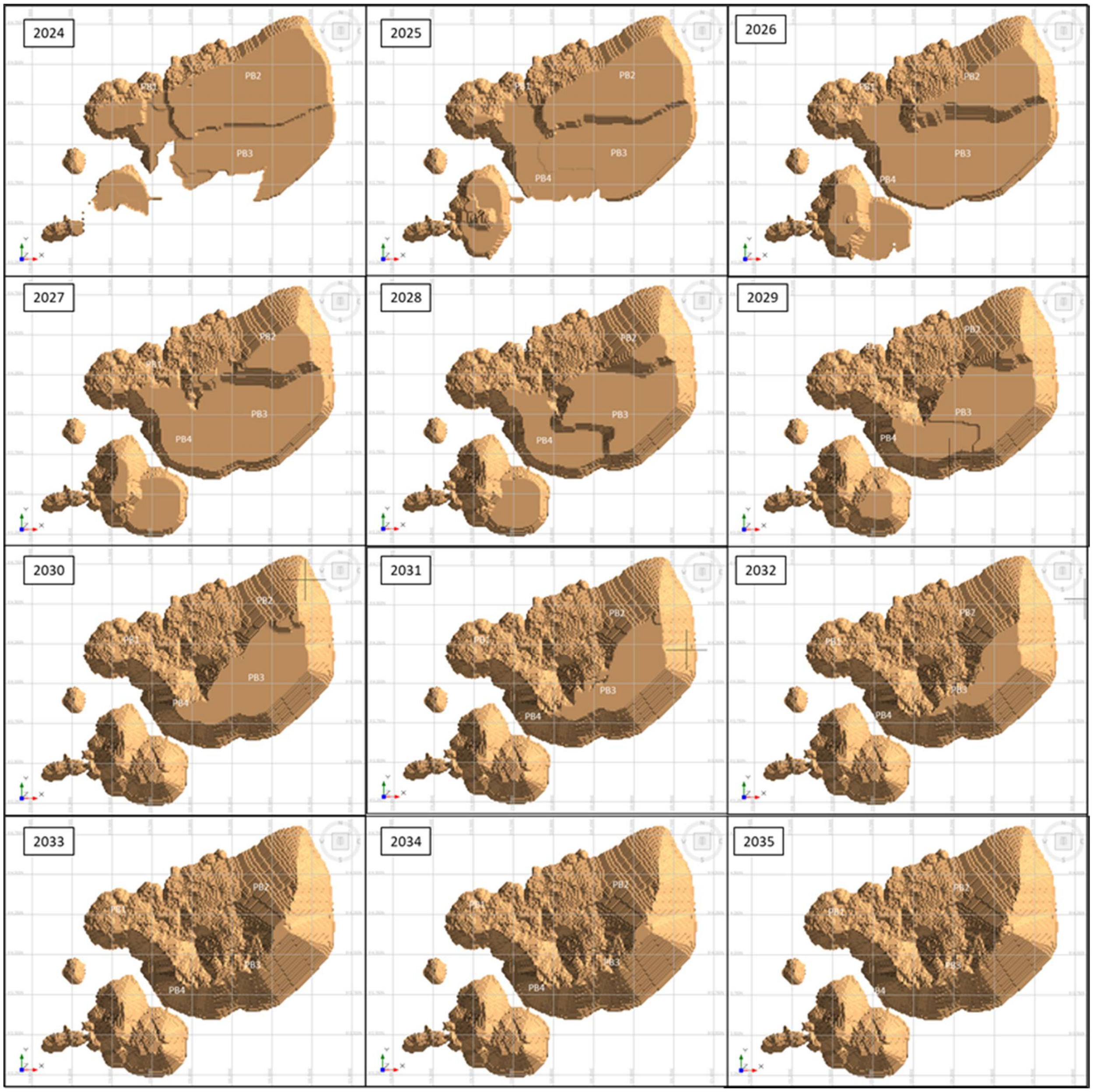


Figure 15-18: Whittle Strategic Schedule Period Face Positions

15.5 Pit Design

In accordance with CIM Definition Standards, all ultimate pit designs are based on current pit optimisation analysis, which effectively treats all Inferred Mineral Resources as waste. Therefore, for the subsequent production scheduling, Inferred Mineral Resources continue to be treated as waste, and are not assumed to be separately stockpiled for future processing.

The Engineered open pit designs are based on the strategic pushback sequence. The mine design process has been to develop practical open pit designs for use in detailed scheduling. Ramps were positioned towards the waste dump accesses and the RoM pad access to optimise haulage distances. There are four distinct step changes for stage selection, but during design, specific pushbacks within individual stages were adjusted to accommodate ramp infrastructure and ensure access to all benches.

The current operating strategy is based on contractor mining, whereby the contractor will scope the fleet according to their equipment availability. For example, the PFS study recommended Caterpillar 777G (Cat 777) or similar 100 tonne trucks, which might be more readily available in West Africa. However, the current inventories and strip ratio lends itself to larger equipment in the Caterpillar 789D (Cat 789) range or similar 200-tonne trucks, focused more on bulk waste mining.

For the DFS all interim and final pit design was done for both truck options for a more accurate cost trade-off. Final designs selected and discussed further within this section, are based on the Caterpillar 789D design criteria.

15.5.1 Geotechnical Pit Design Criteria

The geotechnical pit design criteria were developed by BG and is summarised in Section 15.1 Geotechnical domains consisted of nine domains in the Main pit and a further two domains for the satellite pits. In addition, decoupling berms of the appropriate height were included in the design should a ramp not pass the pit wall within the allowable Bench Stack Height.

15.5.2 Haul Road and Ramp Design Parameters

15.5.3 Haul Road Design Standards

The haul road design parameters were established considering the type and size of material hauling equipment used during mining.

The haul road dimensions were based on global standards of good practise. Guidelines specify that the vehicle operating width should be multiplied by a factor of 3.5 for double-lane traffic and a factor of two for single-lane traffic to determine the effective operating width of the haul road and to incorporate the road infrastructure required (i.e., the safety berm and drainage channel). The haul road gradient and width are discussed in Sections 15.5.4 and 15.5.5, following.

15.5.4 Haul Road Gradient

A reduction in haul road gradient significantly increases a vehicle's attainable uphill speed, while decreasing the haulage cycle times, fuel consumption, and stress on mechanical components thus by association the maintenance costs.

A haul road gradient of 1:10 (10% or 5.71°) was selected for the Project. The selection of the haul road gradient was based on best practise for the type of trucks proposed.

15.5.5 Haul Road Width

Depending on the preferred truck option, the in-pit roads and ramps were designed to have an overall width of 28 m for dual haul roads and 18 m for a single lane road. Table 15-14 following summarises the haul road design parameters. In addition, ramps include additional room for drainage ditches and safety berms, as illustrated in Figure 15-19 and Figure 15-20, for double and single-lane roads respectively.

Table 15-14: Haul Road and Ramp Design Parameters

Truck	Tyres		A	B	C	D	E	F
	Type	Dia (m)	Operating Width (m)	Bund Height (m)	Bund Width (m)	Drain Width (m)	Minimum Pavement (m)	Total Width (m)
Dual Haul Road								
• Cat 777G	27.00R49	2.7	6.3	1.3	4.7	1.0	18.8	25
• Cat 789D	37.00R57	3.5	7.0	1.7	5.7	1.0	21.0	28
Single Haul Road								
• Cat 777G	27.00R49	2.7	6.2	1.3	4.7	1.0	9.3	15
• Cat 789D	37.00R57	3.5	7.0	1.7	5.7	1.0	11.5	18

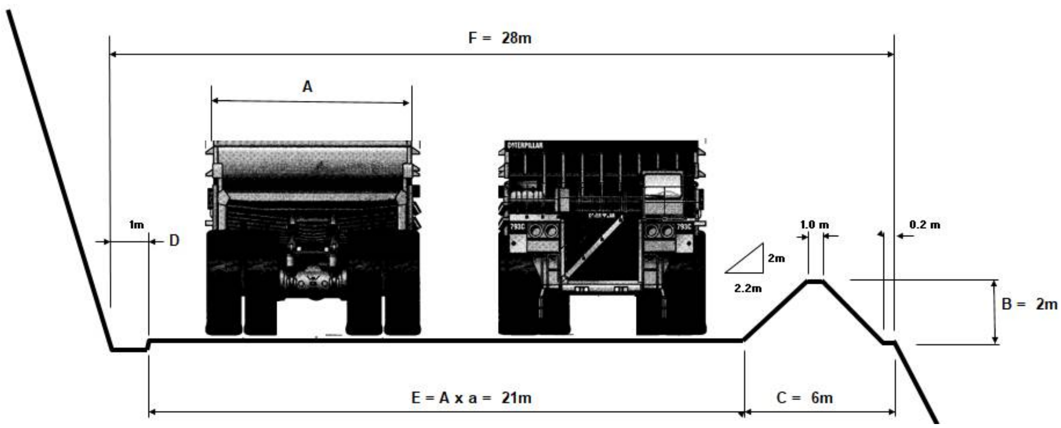


Figure 15-19: Dual Haul Road Cross Section

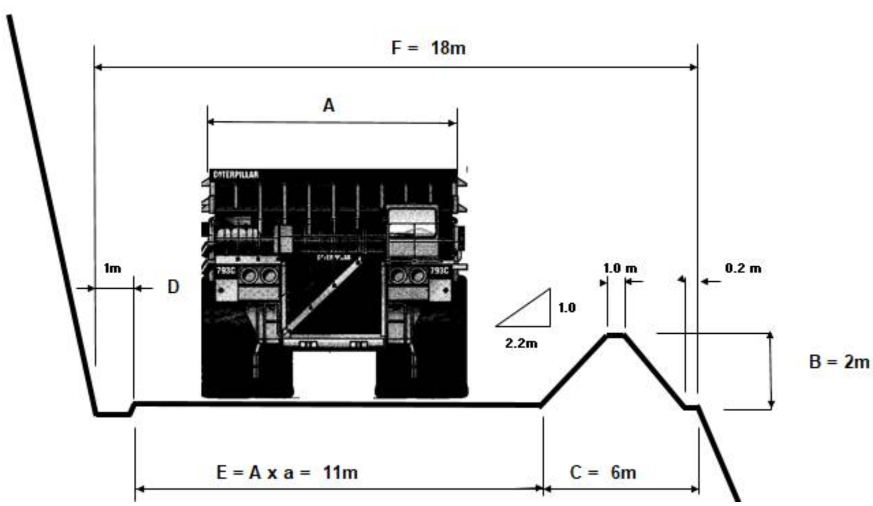


Figure 15-20: Single Haul Road Cross Section

The design, construction and maintenance of haul roads have a considerable impact on haulage cost, which makes up a significant percentage of the total mining cost. Therefore, appropriate detailed designs for haul road construction are developed for the Site.

There are considerable benefits to proper haul road design and construction. These include:

- reduced tyre wear and damage;
- reduction in cycle time due to improved haulage efficiency;
- decrease in fuel consumption; and,
- reduced truck component wear.

Therefore, generating a minimum site-wide construction standard for both new and existing haul roads is necessary.

15.5.6 Minimum Mining Width

The equipment choices limit the minimum operating width for the pit. For example, for a single-sided loading configuration, a minimum mining width of 25 m should be sufficient for a safe and effective operating environment.

15.5.7 Design Approach

The following methodology was followed during the design process:

- use the selected optimal pit shells derived from the pit optimization as the design limit;
- use the latest block model to show the ore distribution; and
- apply the pit design criteria and geotechnical parameters as discussed in the preceding sections. Pit walls were expanded with the addition of haul roads where required, and the haul road width was reduced at the lower pit levels to minimise waste stripping as far as practical.

The design work was performed in Deswik® CAD software. The pushbacks were designed based on the selected interim pit shells. The designs were used to evaluate the tonnage and grades of all the different material types, which can be used to perform production scheduling.

Ramp positioning is an integral design component and directly influences strip ratio and haulage distances.

The exit positions of the ramps were determined, considering the proposed location of the primary crusher and the waste dump.

15.5.8 Open Pit Design

Figure 15-21 to Figure 15-24, following, illustrate the Lafigué stage designs. The hanging wall is to the south of the pit, while the footwall is in the north. The number of ramps passing the hanging wall was kept as low as practicable, to avoid lowering the overall slope angle and increasing the waste tonnage. The pit design in the north followed the ore body as closely as possible to reduce dilution and losses in the footwall.

Interim Stages were selected with Stage 1 at USD 725/oz, which includes pushbacks 1 for Pit A, Pit B and Pit C. Stage 2 was designed between USD 850 and 875/oz, to accommodate the minimum mining width and expands Pit A to Pushback 2 boundary. Stage 3 included Pit A pushback 3 and Pit B pushback 2 and was designed between USD 900 and 1050/oz, to allow consistent ramp access between north and south, transitioning into Stage 4, designed at USD 1175/oz pit shell. Stage 4 mines the final Pit A pushback 4, forming the ultimate pit boundary.

15.5.9 Lafigué Stage 1

Stage 1 comprises the satellite pit, Pit C, Pushback 1 of the satellite pit, Pit B, and Pushback 1 of the Main pit, and Pit A. Pushback 1 of the Main pit reaches a maximum depth of 90 m at 290 mamsl. Therefore, Pit B is brought in during stage 1 to bring the oxide material forward in the schedule.

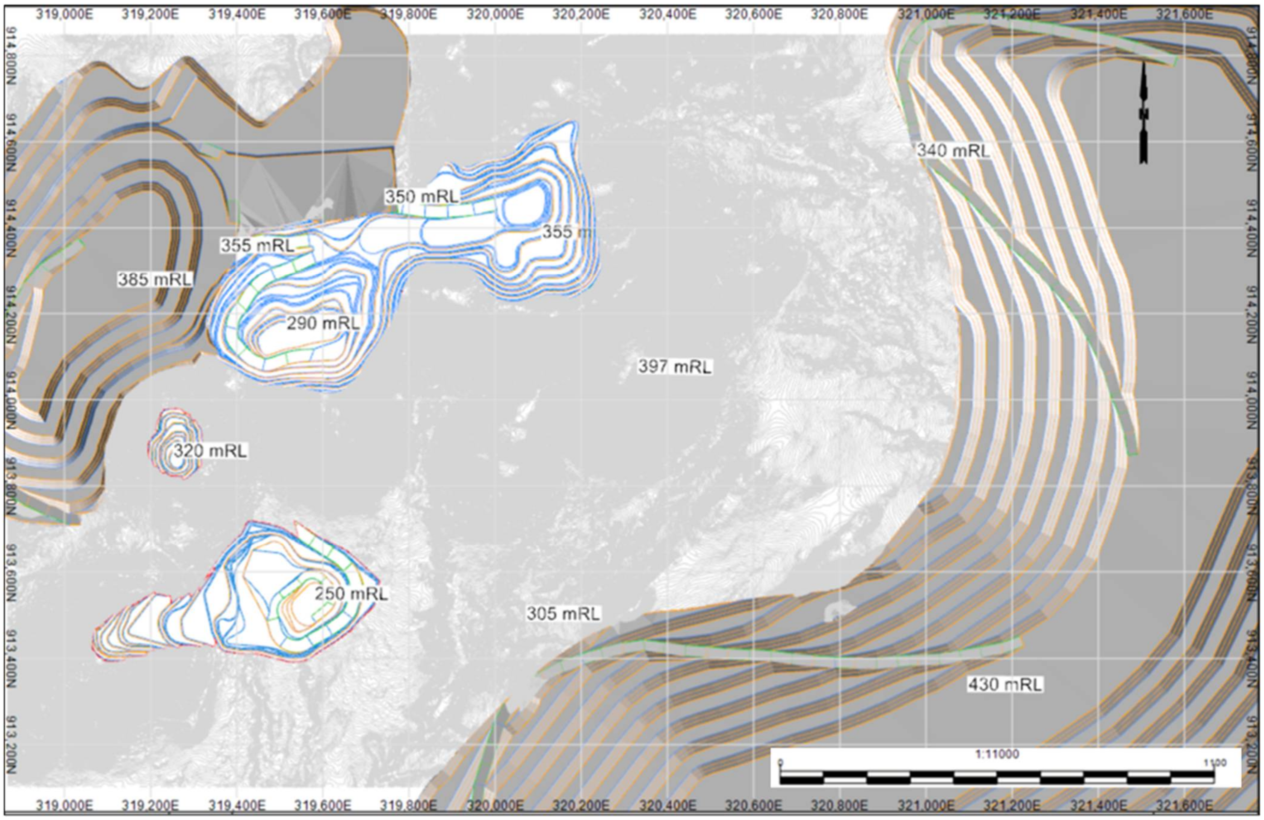


Figure 15-21: Lafigué Stage 1 Design (Plan View) (SRK,2022)

15.5.10 Lafigué Stage 2

Stage 2 extends the Main pit to the east and down to 135 mamsl, resulting in a maximum depth of 230 m. Pit B remains the same during Stage 2 to focus mining on the Main pit. The Main pit's north wall (footwall) forms part of the final pit wall.

Pit A pushback 2 design was adjusted to provide continuous ramp access to the RoM pad. Additionally, additional material was mined in the saddle between east and west to avoid requiring a separate or temporary ramp to mine this material going into pushback 3. These changes resulted in Stage 2 being between USD 850 and 875/oz.

The first footwall ramp switchback was added earlier to avoid this switchback in Geotech zone DS6, and for the ramp to join up at 235 mamsl, forming a common point for Pit A Pushback 3.

Temporary highwalls were designed two degrees steeper between pushback 2 and pushback 3. This highwall never exceeds 80 m in height.

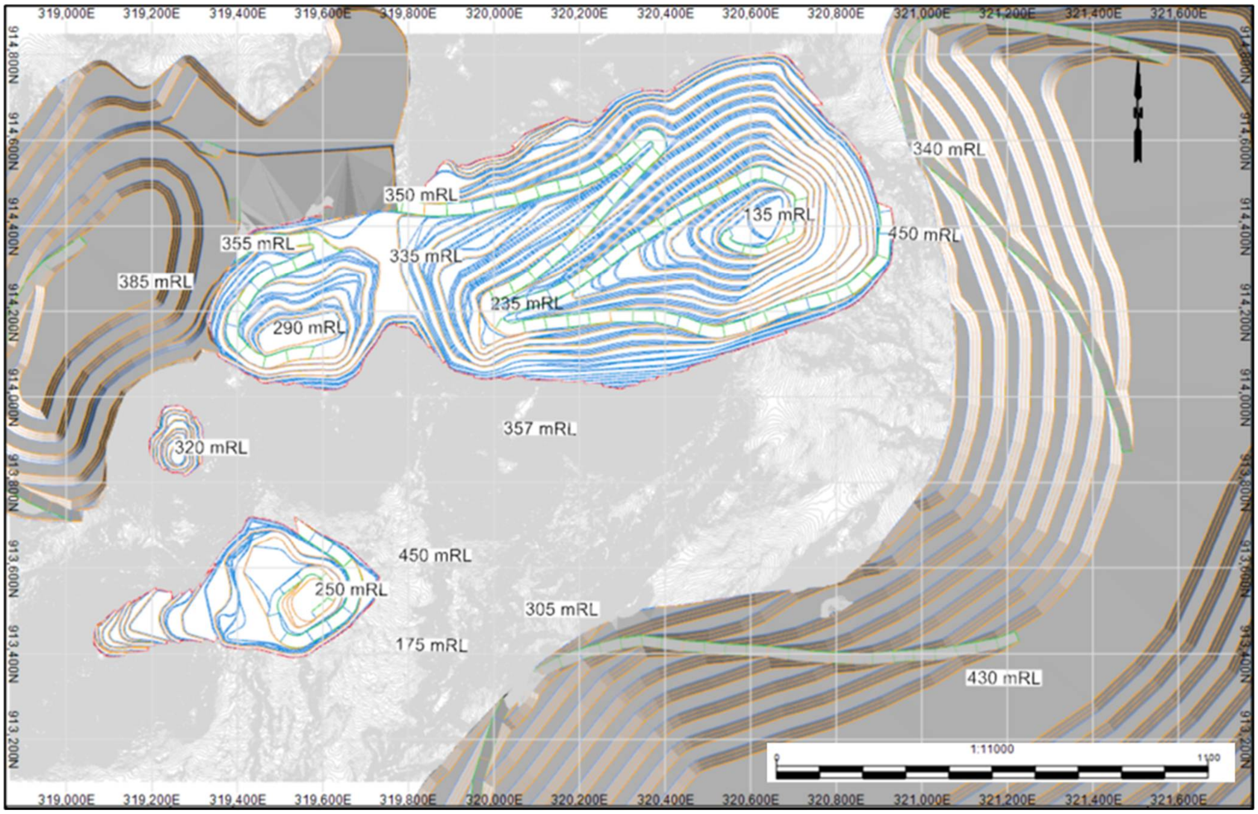


Figure 15-22: Lafigué Stage 2 Design (Plan View) (SRK,2022)

15.5.11 Lafigué Stage 3

Stage 3 reaches a maximum depth of 235 m at the 140 mamsl in the Main Pit A pushback 3, with Pit B and Pit C at final limits. Main pit pushback 3 was limited to 140 mamsl to join up with pushback 4, which extends the Main pit deeper and to the south. This south wall then forms part of the final highwall.

Pit A pushback 3 was adjusted slightly in width and limited to 140 mamsl, to allow continuous ramp access with pushback 4 between 265 and 215 mamsl and below 140 mamsl.

Temporary highwalls were designed two degrees steeper between pushback 3 and pushback 4. This highwall never exceeds 40 m in height.

Whittle targeted high-grade ore to a depth of 30 mamsl in the eastern corner of Pit A pushback 3 with pushback 4 stripping to gain access to ore in the west and then go deeper in the east to mine higher-grade ore between 30 and 0 mamsl.

There is an opportunity to extend Pit A pushback 3 deeper to mine all the ore tonnes Whittle selected for Pit A pushback 3; however, the strategic schedule indicated that mining deeper at this stage would mainly add material to the stockpile and is not explicitly required to sustain plant feed. In addition, it should be noted that if Pit A pushback 3 is extended to the entire boundary and depth at 140 mamsl, Pit A pushback 4 will require additional ramps with switchbacks to maintain bench access. The additional ramps and switchbacks will add significant waste to the design and increase truck cycle times.

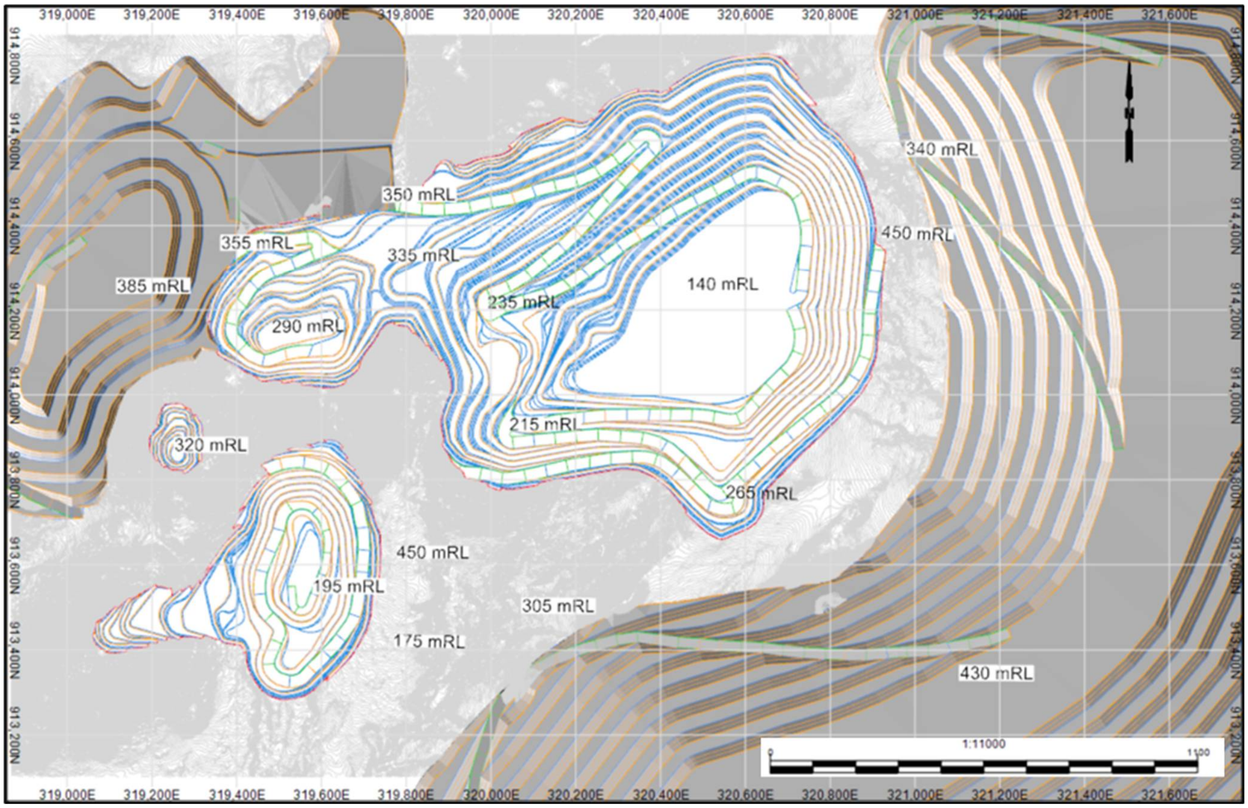


Figure 15-23: Lafigué Stage 3 Design (Plan View) (SRK,2022)

15.5.12 Lafigué Stage 4

Stage 4 extends the Main pit deeper to its maximum depth of 338 m at 0 mamsl. It also advances to the southwest to reach the final pit wall, forming the ultimate pit design.

Table 15-15 summarises the pit design inventories by stage. The inventories show that the designs do not precisely follow the selected Whittle shells. Designing to follow the pit shells to operate in isolation from each other required multiple switchbacks resulting in flatter overall slopes, more waste being mined and a significant increase in cycle times. In addition, the strategic schedule showed that Lafigué quickly extends to final limits, and pushbacks add limited value.

Figure 15-11 in the optimisation section (Section 15.4.1) showed that there are predominantly only two pushbacks in the southwest (Section A-A) and northeast (Section C-C) within the Main pit and four potential pushbacks in the area of Section B-B. The Whittle optimisations also show a limited benefit from pushbacks after USD 1000/oz with consistent strip ratios to USD 1175/oz. For this reason, the pits were engineered, focusing on mining access from multiple sides with continuous bench access, and optimised haulage distances while following the face positions from the strategic and interim tactical schedules as best possible. For the final tactical schedule, the various pushbacks were sub-divided into smaller locations based on elevation to focus on reducing haulage distances and multiple switchbacks within a single haul cycle.

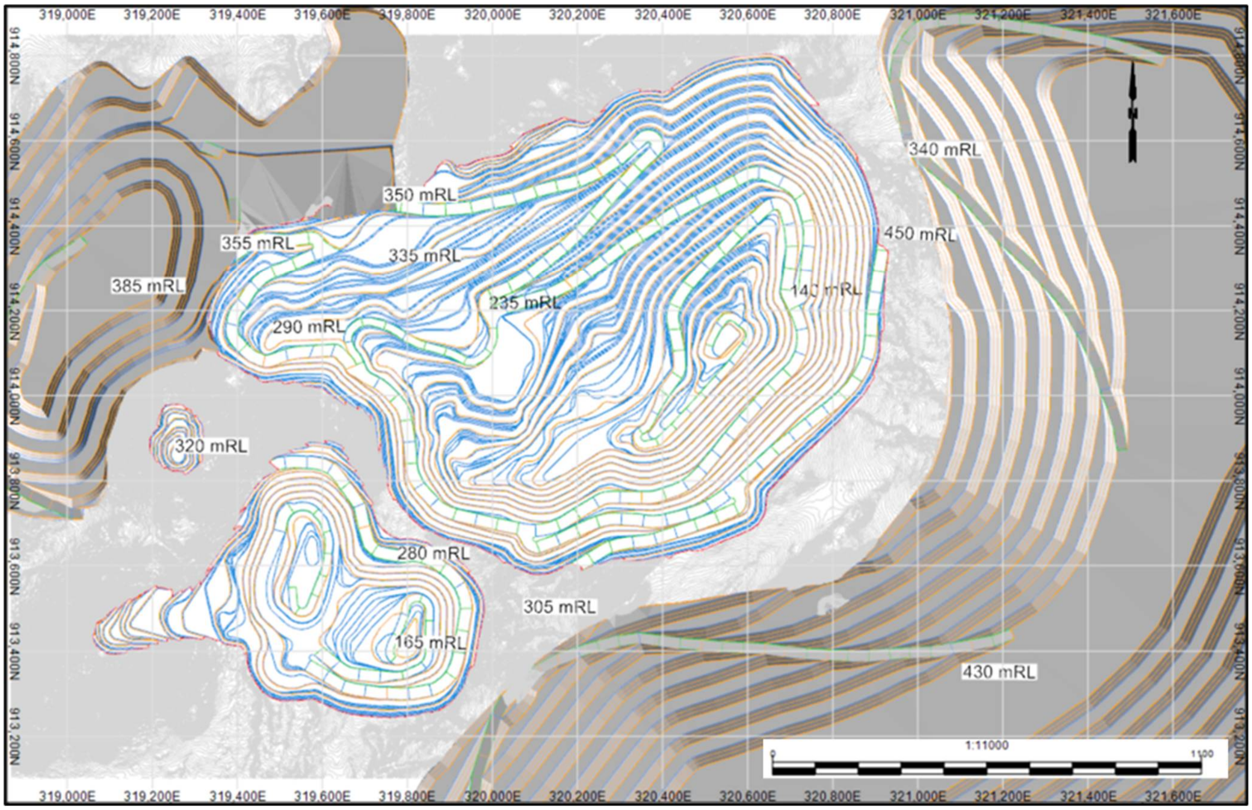


Figure 15-24: Ultimate Lafigué Pit Design (Plan View) (SRK,2022)

15.6 Mining Inventory Summary

Table 15-15, Table 15-16 and Table 13-1, following, summarise the total pit design inventory by weathering type. All inventories are depicted as dry metric tonnes (dmt) unless stated otherwise.

The higher strip ratio for Pit A pushback 3 is because the ore at depth below 140 mamsl was added to pushback 4. Pushback 3 can be mined deeper, decreasing the strip ratio substantially, but this will require temporary ramps and risk Pit A pushback 4 losing ramp access to benches between 140 mamsl and 30 mamsl. The strategic and tactical schedules showed that adding more ore to pushback 3 is not required as enough ore is available. Adding more ore to pushback 3 will reduce waste stripping requirements for pushback 4, but the schedule showed fluctuations in the ounce profile due to the balance between HG and LG ore not being achieved. These findings again align with the interpretation of the optimisation results and strategic schedule. All inventories are reported based on the Cat 789 design, which has a slightly larger pit and more waste material due to increased ramp width.

Table 15-15: Pit Design Summary

Deposit	Total (kt)	Waste (kt)	Ore (kt)	Grade (g/t)	Gold (koz)	Total (kbcm)	Waste (kbcm)	Ore (kbcm)
Pit A PB1	23 545	17 674	5871	1.60	301	9486	7169	2317
Pit A PB2	161 670	146 536	15 134	1.80	874	59 307	53 897	5410
Pit A PB3	138 871	131 774	7097	1.57	358	50 048	47 508	2540
Pit A PB4	118 139	99 591	18 548	1.63	973	42 488	35 885	6603
Pit B PB1	5 971	5090	881	1.71	48	2787	2403	384
Pit B PB2	18 621	17 562	1060	1.98	67	6997	6626	371
Pit B PB3	24 361	23 308	1053	2.50	85	9266	8898	368
Pit C	436	268	168	1.51	8	206	133	73
Total	**491 615**	**441 802**	**49 813**	**1.69**	**2714**	**180 584**	**162 517**	**18 067**

Table 15-16: Pit Design Summary by Weathering

Waste/Ore	Weathering	Unit	Total	Pit A PB1	Pit A PB2	Pit A PB3	Pit A PB4	Pit B PB1	Pit B PB2	Pit B PB3	Pit C
Waste	Laterite	kt	2015	474	900	106	22	185	54	254	19
	Oxide	kt	19 508	1945	6471	3302	2329	1855	1483	2076	46
	Transitional	kt	32169	5 227	10 222	6054	5413	1148	1315	2593	198
	Fresh	kt	388 111	10 028	128 943	122 311	91 827	1902	14 710	18 386	3
Total	**Total**	**kt**	**441 802**	**17 674**	**146 536**	**131 774**	**99 591**	**5090**	**17 562**	**23 308**	**268**
Ore	Laterite	kt	24	16	0	0	0	9	0	0	0
	Oxide	kt	843	622	0	0	0	206	1	0	13
	Transitional	kt	2079	1575	96	0	8	239	6	0	155
	Fresh	kt	46 866	3658	15 038	7097	18 540	426	1052	1053	0
Total	**Total**	**kt**	**49 813**	**5871**	**15 134**	**7097**	**18 548**	**881**	**1060**	**1053**	**168**
Total	Laterite	kt	2039	490	900	106	22	194	54	254	19
	Oxide	kt	20 351	2568	6 471	3302	2329	2061	1484	2076	59
	Transitional	kt	34 248	6802	10 317	6054	5420	1387	1321	2593	353
	Fresh	kt	434 977	13 687	143 981	129 409	110 367	2329	15 762	19 439	4
Total	**Total**	**kt**	**491 615**	**23 545**	**161 670**	**138 871**	**118 139**	**5971**	**18 621**	**24 361**	**436**
	SR (W:O)		8.9	3.0	9.7	18.6	5.4	5.8	16.6	22.1	1.6

Table 15-17: Pit Design inventory by Weathering

Weathering Type	Total (kt)	Waste (kt)	Ore (kt)	Grade (g/t)	Gold (koz)	Total (kbcm)	Waste (kbcm)	Ore (kbcm)
Oxide	22 390	21 523	867	1.23	34	13 773	13 238	535
Transitional	34 248	32 169	2079	1.39	93	13 879	13 036	843
Fresh	434 977	388 111	46 866	1.72	2 587	152 932	136 244	16 689
Total	491 615	441 802	49 813	1.69	2 714	180 584	162 517	18 067

At the marginal cut-off grade of 0.4 g/t Au, a total of 50.6 Mt at 1.68 g/t Au was included in the pit designs. Table 15-18 summarises the pit design ore inventory by resource category. Inferred material was treated as waste for scheduling and reporting purposes.

Table 15-19 compares the pit shell (MI) and the ultimate pit designs. The pit designs mine additional waste to maintain the batter angle and berm width requirements for stability while the pit shell follows the block model. Extra waste is also mined due to ramps in the hanging wall. Further design optimisation is ongoing.

Table 15-18: Pit Design Ore inventory by Resource Category

Item	Units	Measured	Indicated	Inferred	Total
Tonnes	Mt	-	49.81	0.95	50.76
Gold grade	g/t	-	1.69	1.24	1.68

Table 15-19: Pit Design Comparison

Item	Units	Design	Whittle	Variance
Total	Mt	491.61	452.35	0.08
Waste	Mt	445.69	407.71	0.09
Ore	Mt	45.92	44.64	0.03
Grade	g/t Au	1.80	1.85	-0.03
Ounces	Moz	2.66	2.66	0.00
Strip ratio	Waste:ore	9.71	9.13	0.06

15.7 Waste Rock Dumps

Waste rock dump design considerations, size and location are discussed in Section 18, whilst waste rock geochemistry is discussed in Section 20.

15.8 Historical Mineral Reserve Statements, December 2020

Snowden completed the 2020 reserve estimate. In addition, Snowden completed pit optimisation and mine design and developed the LoM schedules based on the Endeavour Lafigué deposit block model (fetekro_bm_july202.dm).

The Reserve statement, effective 31 December 2021, reported at USD 1500/oz gold price and a marginal cut-off grade of 0.4 g/t Au is seen in Table 15-20. The Probable Mineral Reserve reported for the Lafigué project was 32.0 Mt at 2.1 g/t Au containing 2160 Au koz with an estimated LoM of 11 years.

Table 15-20: Historical Mineral Reserve Statements December 2020

Item	Units	Proved	Probable	Total Mineral Reserve
Ore	Mt	0	32.0	32.0
Gold grade	g/t Au	0	2.1	2.1
Contained gold	Moz	0	2.1	2.1

Table 15-20 notes: Some numbers may not sum correctly due to rounding

15.9 Mineral Reserve Statement as of the 01 June 2022

The reserve estimate is in accordance with the Canadian National Instrument 43-101, and adhering to the CIM Definition Standards guidelines (CIM, 2014). Therefore, public Reports dealing with exploration results, mineral resources and mineral reserves must use only the terms Proved or Probable mineral reserves, Measured, Indicated, and Inferred mineral resources and exploration results as shown in Figure 15-25.

Figure 15-25 illustrates the framework for classifying tonnage and grade estimates to reflect different levels of geoscientific confidence and degrees of technical and economic evaluation. Mineral Resources can be estimated based on geoscientific information with some input from other relevant disciplines. Mineral Reserves, modified Indicated and Measured Mineral Resources (shown within the dashed outline in Figure 15-25), require consideration of the modifying factors affecting extraction. Measured Mineral Resources may convert to either Proved or Probable Mineral Reserves depending on the certainty associated with the modifying factors that are taken into account in the conversion from Mineral Resources to Mineral Reserves. The broken arrow in Figure 15-25 demonstrates this relationship. Although the trend of the broken arrow includes a vertical component, it does not in this instance, imply a reduction in the level of geoscientific knowledge or confidence. In such a situation, these modifying factors should be fully explained. The term 'modifying factors' includes mining, metallurgical, economic, marketing, legal, environmental, social, and governmental considerations.

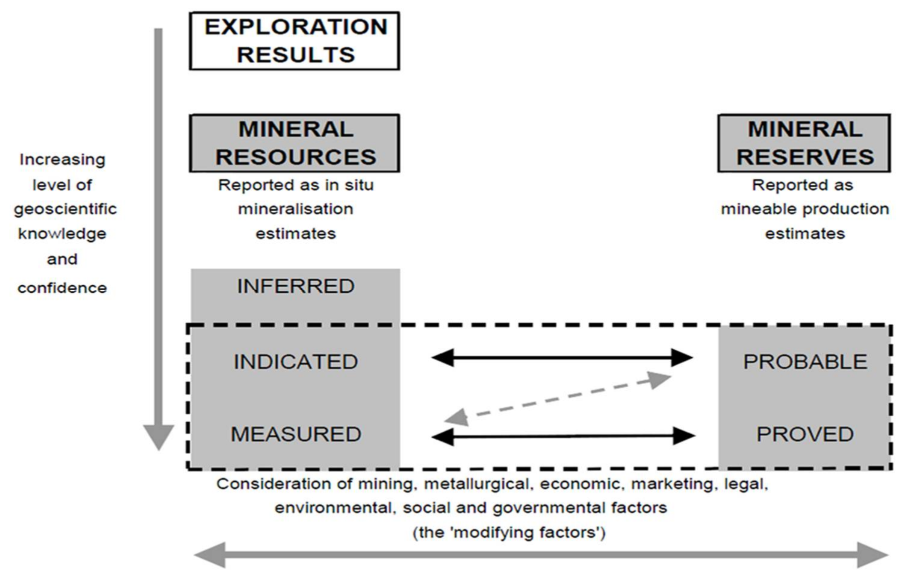


Figure 15-25: Relationship Between Exploration Results, Mineral Resources & Mineral Reserves

SRK confirms that the Mineral Reserve statement presented in Table 15-21 has been derived from the Mineral Resource with an 'Effective Date' of 15 May 2022 authored by SRK (31113_Lafigué_MRE_May_2022).

The Mineral Reserve reported by SRK is constrained within an engineered design pit based on the optimised pit shell generated solely on the Measured and Indicated classified portion of the Mineral Resource. Inferred material within the pit design was treated as waste for reporting purposes.

The Reserve statement, effective 01 June 2022, reported at USD 1300/oz gold price and a marginal cut-off grade of 0.40 g/t Au is presented in Table 15-21. The Probable Mineral Reserve reported for the Lafigué project is 49.81 Mt at 1.69 g/t Au, containing 2.71 Moz of Au with an estimated LoM of 13 years. The reported mineral reserve is associated with 441.8Mt of waste mining corresponding to 8.9 to 1.0 waste to ore strip ratio on mass.

Table 15-21: Lafigué Project Mineral Reserve Statement 01 June 2022

Classification Category	Material Type	Tonnes (Mt)	Grade, Au (g/t)	Metal Content, Au (Moz)	CoG (g/t)
Proved	Oxide				
	Transitional				
	Fresh				
Sub-Total Proved	**ALL**				
Probable	Oxide	0.87	1.23	0.03	0.40
	Transitional	2.08	1.39	0.09	0.40
	Fresh	46.87	1.72	2.59	0.40
Sub-Total Probable	**ALL**	**49.81**	**1.69**	**2.71**	**0.40**
Proved and Probable	Oxide	0.87	1.23	0.03	0.40
	Transitional	2.08	1.39	0.09	0.40
	Fresh	46.87	1.72	2.59	0.40
Total Ore Reserve	**ALL**	**49.81**	**1.69**	**2.714**	**0.40**

Table 15-21 notes:

- Some numbers may not sum correctly due to rounding.
- The statement was depleted based on the depletion surface of end August 2021 ("topo_artisanal_09092021"), including Artisanal mining up to this date.
- Above cut-off grade of 0.40 g/t Au using an Au price of USD 1300/oz.
- Modifying factors for dilution range from (0 to 14)% and mining recovery between (95 to 100)%.
- All figures are rounded to reflect the relative accuracy of the estimate.
- Ore Reserves have demonstrated economic viability.
- The pit inventories were constrained within a pit design. The Ore Reserve comprises a mine life of some thirteen years, with an additional one years of stockpile feed, totalling 13 years.
- The mineral resources and reserves have been estimated and reported in accordance with Canadian National Instrument 43-101, 'Standards of Disclosure for Mineral Projects and the Definition Standards adopted by CIM Council in May 2014. The Ore Reserve is given based on 100% ownership of the property.

15.10 Data Verification

The methodology applied to verifying/using the data presented in Section 15, and any limitations thereof, are discussed more fully in Section 12.

15.11 Comments on Section 15

15.11.1 Geotechnical

The geotechnical model is formed from a number of base models which BG consider range from Conceptual to Design and Construction Level. Whilst a number of the base models fall below Feasibility Level, BG consider that inherent uncertainties and variabilities in the geotechnical model as they stand, are such that there would be no foreseen adverse effect of these elements on the operational viability of the overall slope design. The QP agrees that the slope design criteria proposed by BG appears appropriate given the data presented in the BG Feasibility Study.

15.11.2 Hydrogeology

The QP considers that the pit hydrogeological characterisation and dewatering design has been undertaken to a 'pre-feasibility study level' of development. The current data is adequate to support the Ore Reserves, but further work is required to better define the geological structural model and the associated hydrogeological conditions. However, whilst there are hydrological risks, these are not considered significant relative to the geotechnical risks.

15.11.3 Mineral Reserves

The reserve estimate has been undertaken to a DFS level of accuracy in accordance with the Canadian National Instrument 43-101, and adhering to the CIM Definition Standards guidelines (CIM, 2014).

15.12 Interpretations and Conclusions

Interpretations, conclusions, and risks for Section 15 are presented in Section 25.

15.13 Recommendations

Recommendations/forward work programme activities for Section 15 are presented in Section 26.

15.14 References

References cited in the preparation of Section 15 are presented in Section 27.

16. MINING METHODS

16.1 Introduction

The following section focuses on the mining engineering aspects and associated assumptions as incorporated into the current Lafigué DFS, with specific comment on mining methods and mining equipment as well as associated assumptions risks and opportunities.

16.2 Mining Method

The Project will make use of conventional open pit truck and excavator operation with the production unit operations (drilling, blasting, loading, hauling, and dumping) carried out by contractor mining personnel and equipment. The mining contractor will be responsible for short term production planning, drilling (production and grade control), loading and hauling. Blasting will be carried out by a specialised blasting contractor that will also be responsible for the supply of explosives.

The saprolite is anticipated to be primarily free-dig, potentially requiring ripping with 14% of oxide material planned for blasting. A low powder factor was estimated for blasting the oxide material, with blasting of oxides mainly consisting of fracturing the harder laterite cap with a low powder factor of 0.32 kg/m³. As the rock strengths increase, blasting will be utilised more regularly within the transitional zone with powder factors estimated at 0.59 kg/m³. All the fresh rock will be blasted with powder factors estimated at 0.76 kg/m³. Production drilling of transitional and fresh material will be undertaken by top hammer drills drilling 152 mm diameter holes.

Mining is envisioned to occur in 10 m benches, with double batters to achieve the final 20 m bench. Mining will occur in 3 to 4 flitches depending on required vertical selectivity. This practise decreases dilution by using selectivity practises utilising smaller loading units for ore loading. Ore and waste will be loaded with hydraulic diesel shovels and all material will be hauled out of the pits by diesel powered trucks. The material will be hauled to various destinations as part of the overall mining strategy:

- directly to primary crusher;
- RoM pad stockpile;
- topsoil stockpiles;
- aggregate stockpile; and
- waste dump.

16.3 Hydrogeology and Open Pit Water Inflow Review

SRK has conducted a review and assessment of all hydrological data and reports that have a bearing on the mine design and are likely to influence the technical information included in the Life of Mine Plan ('LoMp') and Ore Reserves. The review of the Hydrological data excludes independent verification by means of re-calculation. Whilst SRK has exercised all due care in reviewing the supplied information, SRK does not accept responsibility for finding any errors or omissions contained therein and disclaims liability for any consequences of such errors or omissions. The following section summarises the data reviewed and should not be taken as superseding the main authored report.

A hydrogeological assessment was completed by Endeavour in May 2021 (EMS, 2021) and is supported by hydrogeological field programmes undertaken in November/December 2020 and April 2021. Hydrological program was guided by the pre-feasibility study (PFS) pit limits which is slightly smaller than the DFS. This does not influence the review or recommendations.

16.3.1 Climate and Surface Water Hydrology Characterisation

The design climatology which forms the hydrological basis of the Feasibility Study, was developed by Knight Piesold (KP, 2021b). The design average annual rainfall (i.e., mean annual precipitation; 'MAP') is 1199 mm, with the 100-year annual recurrence interval ('ARI') wet rainfall totalling 1772 mm and the 100-year ARI dry rainfall totalling 583 mm. The rainy season extends from April to October with 70% of rainfall occurring in the wet season.

The 24-hour 100-year ARI storm rainfall is 189 mm. This figure was adopted for open pit flood risk assessment.

Groundwater recharge has been estimated to vary between 1 and 3% of MAP, equivalent to between 11 and 34 mm/a.

The footprint of the open pits is located on, or close to, the surface water catchment divides of four catchments. The topography therefore typically slopes away from the pit perimeters.

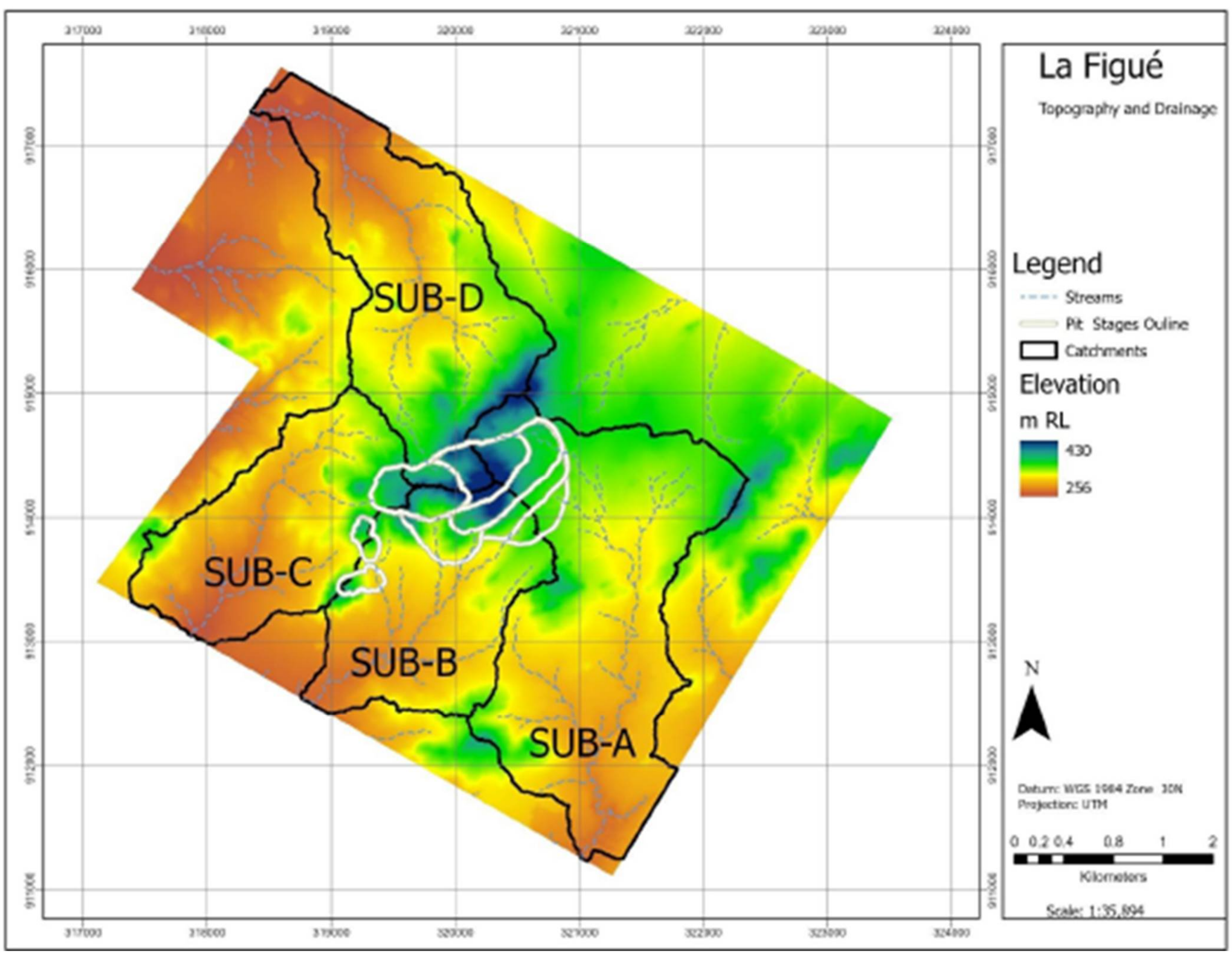


Figure 16-1: Topography and Drainage setting of Lafigué Open Pits (EMS, 2021)

16.3.2 Groundwater Characterisation

A hydrogeological drilling and testing program was conducted by Foraco International SA in November and December 2021, with a second programme taking place in April 2021 involving Geodrill, who were appointed to drill and airlift test deep RC holes under the supervision of a hydrogeologist. Nine hydrogeological wells were drilled in total; hole details are summarised in Table 16-1 and hole locations are shown in Figure 16-2.

Table 16-1: Hydrogeological Boreholes for Groundwater Characterisation

Site ID	X (m)	Y (m)	Z (m)	Depth (mRL)	Water Strike (mbgl)	Water Strike (mamsl)	Final Yield (m^3/a)	Casing Radius (mm)	Well Radius (mm)	SWL (mbgl)
FTBH01	320914	914502	337	145.5	21.5, 77	315, 260	0.7	140	165	23.1
FTBH02	320637	913806	340	180	86, 87.5, 93		5	140	165	32
FTBH03	320015	913616	312	150	45.8, 104.5		14.4	200	254	14.1
FTBH04	319568	913980	371	108.8	92		0.5	140	165	68.48
FTBH05	320186	914623	381	150	33		0.72	140	165	31.35
FTBH06	319531	914404	360	102	84		Minor seepage	-	140	-
FTBH07	319811	913845	320	150	41, 100, 139		3	-	150	21
FTBH08	320543	914226	363	348	198, 289	198, 298	1.3	-	140	>150
FTBH09	319988	913838	346	210	71, 170, 192		4	-	140	44

Table 16-1 notes: mRL is equivalent to mamsl

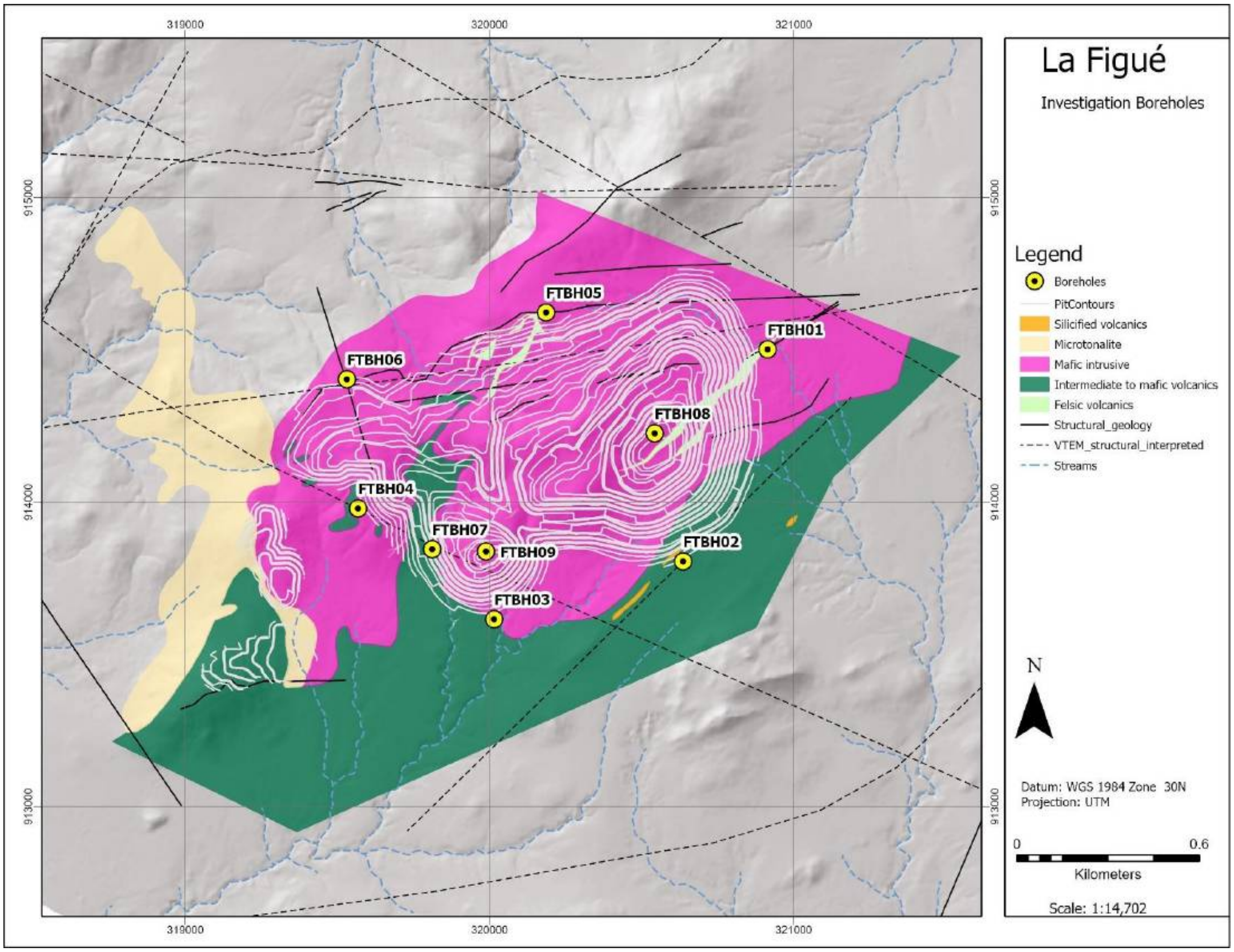


Figure 16-2: Location of Hydrogeological Boreholes Relative to Planned Lafigué Open Pits (EMS, 2021)

The objective of each hydrogeological borehole was as follows:

- FTBH01: targeting a mapped north-east structure between felsic and mafic intrusive on the east wall.
- FTBH02: to investigate an interpreted VTEM structure on the southeast wall.
- FTBH03: to intersect the contact between mafic volcanic and mafic intrusive at the south wall to a final depth of 150 mbgl.
- FTBH04: to investigate an interpreted VTEM structure on the west wall to a final depth of 108.8 mbgl.
- FTBH05: targeting a mapped EW structure between felsic and mafic intrusive on the north wall.
- FTBH06: targeting a structural intersection on the northwest wall to a final depth of 102 m.
- FTBH07: to investigate a contact between felsic volcanic, mafic volcanic, and mafic intrusive within the pit shell.
- FTBH08: to investigate the deepest section of the pit shell (the hole was drilled to a final depth of 348 mbgl).
- FTBH09: to investigate the deepest section of Stage 6 of the pre-feasibility study (PFS).

The laterite/saprolite thickness ranged from 3 to 18 m across the footprint of the pits, with fractured and underlying weathered saprock ranging between 2 and 24 m. Total thickness of the weathered Rock Mass Unit ('RMU') therefore ranges between 8 and 27 m, consistent with the characterisation in the geotechnical design specification study of <50 m (Bastion, 2021).

The following groundwater characteristics were determined:

- Groundwater occurrence is associated with open fractures in intrusive mafic and mafic volcanic rock units. Where only mafic intrusives are present, the yield is typically less than 5 m^3/h. Open fractures at rock contacts between mafic intrusive and mafic volcanic units yield >10 m^3/h, especially in the southwest area of the pit shell.
- Compared with collar elevations of hydrogeological boreholes (312 to 380 mamsl), groundwater is mainly intercepted between 250 to 300 mamsl where yields are <3 m^3/h. Relatively higher yields (up to 15 m^3/h) are noted between 200 and 250 mamsl, with isolated water-bearing fractures with lower yields in the lower bedrock down to 150 mamsl below which fractures are rare and not water-bearing.
- The saprock transition zone is relatively devoid of groundwater with the exception of the east and north pit sectors, where low yields associated with an elevated water table were observed.
- The deepest section of the pit floor (investigated with FTBH08) is relatively dry, with a yield in the region of 1 m^3/h.
- No water (or seepage) was intercepted in the saprolites. This indicates that the saprolites do not store water and maybe sufficiently permeable to transmit any recharge to the underlying bedrock. This suggests that any ponding water on the pit crest (in the rainy season in particular) may be readily transferred to the pit face.

Groundwater levels at 87 locations (a combination of exploration and hydro boreholes) were monitored. The average static groundwater level is 44 mbgl. At most pit locations covering almost two-thirds of the pit footprint (from the northern catchment divide to the south), the groundwater level is between 300 and 320 mamsl. In the upper north area, the groundwater level is higher than 320 mamsl and is influenced by the NE trending regional structure in that locality. General flow direction is southwest in the southern area and northward within the northern pit area.

Pumping tests were undertaken in two of the hydrogeological boreholes, FTBH02 and FTBH03:

- A constant flow rate of 5 m^3/h was achieved at FTBH02 resulting in a drawdown of over 40 m after 3 days of pumping. The test confirmed inflow zones at 46, 52 and 64 mbgl. A variable response was noted in two nearby observation boreholes during the test, one located 90 m and the other 100 m from the pumping well. Analysis of the test results indicated a hydraulic conductivity of 0.065 m/d (7.5×10^{-7} m/s);
- A constant flow rate of 14.6 m^3/h was achieved at FTBH03 resulting in a drawdown of over 80 m after 3 days of pumping. Inflow horizons at 36 and 46 mbgl were confirmed from the test. A variable response was noted in multiple nearby observation boreholes during the test indicating a fracture network extending in a southerly direction. Analysis of the test results indicated a hydraulic conductivity of 0.028 m/d (3.24x10-7 m/s) and a storage coefficient estimated at 2.08 x10-5.

Airlift tests were also undertaken in two hydrogeological boreholes, FTBH07 and FTBH09. The hydraulic conductivity estimates determined from these tests were 0.064 m/d (FTBH07) and 0.0264 m/d (FTBH09).

The geometric mean hydraulic conductivity for the water-bearing fractures was estimated based on all test results to be 0.039 m/d.

16.3.3 Numerical Groundwater Modelling

A 3D numerical groundwater model was developed in MINEDW to predict pore pressure distribution through different stages of the mine life (EMS, 2021). This model informed the slope stability analysis and design of pit slopes (Section 15). In summary, acceptable calibration of steady-state conditions was achieved based on the following:

- hydraulic conductivity of the mafic units was less than the geometric mean from the testwork, but within the range of values determined from the pumping tests;
- the felsic intrusives were simulated as low permeability dykes;
- the simulated hydraulic conductivity of the footwall was very low to achieve the higher groundwater heads in this area;
- representative specific storage of 2x10-7 was achieved for the water-bearing units; and
- calibrated recharge value varies from 16.4 mm/a for lower-lying areas, to 65 mm/a for areas with an elevation higher than 380 mamsl.

Table 16-2: Hydraulic Conductivity and Storativity Values used in the Calibrated Numerical Groundwater Model

Model Unit	Kxy (m/d)	Kz	Ss	Sy
Saprolite/Overburden	0.3	0.3	1x10-4	0.05
Mafic Intrusive	0.015	0.015	2x10-7	0.005
Mafic Volcanic	0.025	0.025	2x10-7	0.005
Felsic Intrusive	0.001	0.001	2x10-7	0.005
Footwall (elevated heads)	0.0002	0.0002	1x10-7	0.005

Table 16-2 notes:

- Kxy = horizontal hydraulic conductivity
- Kz = vertical hydraulic conductivity
- Ss = specific storage
- Sy = specific yield

In-situ, pre-mining pore pressures were determined and indicated that the Stage 1 open pit development would be above the phreatic (water table) surface. During Stage 2, the maximum predicted in-situ pore pressure would be such that depressurisation will not be required. However, during Stage 3 onwards, depressurisation will be required.

Forward prediction models were run by incorporating progressive development of the Lafigué pit shells into the calibrated model allowing pore pressures to respond dynamically to the excavation.

16.3.3.1 Passive Drainage Modelling Scenario

Predicted groundwater inflow rates with time are depicted in Figure 16-3. This scenario assumes that all groundwater is collected in in-pit sumps in the absence of any slope drainage infrastructure. Annual groundwater inflows range from 0 m^3/h in the first year of mining to 66 m^3/h in the last year of mining (End of FY11). The predicted yearly inflow rates indicate the annual minimum requirement for groundwater dewatering. Stage 1 and 2 will be mined above the water table. During Stage 3 onwards, over 70% of the groundwater inflow is predicted to be derived from the mafic volcanic unit.

Natural seepage from the saprolites and overburden is not predicted to occur. However, transient seepage from the overburden and saprolites should be expected during the rainy season as the groundwater system recharges from incidental rainfall and surface runoff.

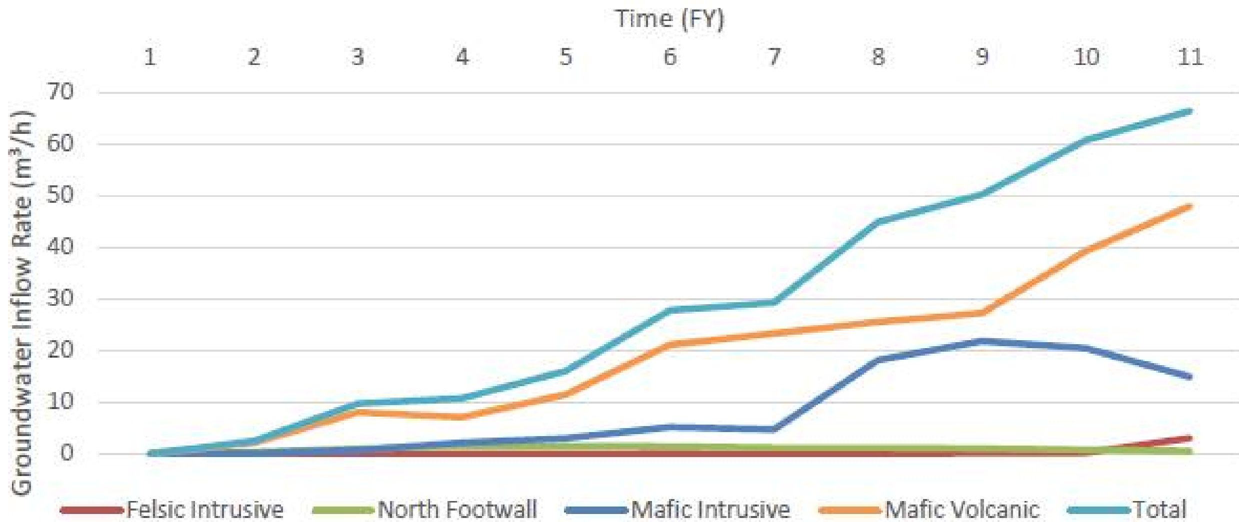


Figure 16-3: Predicted Groundwater Inflow Rates at Lafigué Open Pits (EMS, 2021)

16.3.3.2 Active Drainage Modelling Scenario

This scenario assumes groundwater flow is intercepted by a system of peripheral (ex-pit) and in-pit dewatering wells and horizontal drains. Nine dewatering wells were modelled (three in-pit and six ex-pit; all simulated to a maximum depth of 200 m), initially pumping a combined rate of 112 m^3/h; refer to Figure 16-4 for simulated borehole locations.

These relatively high initial pumping rates are required to deplete storage in the fractured bedrock and lower pore pressures in advance of mining. Pumping rates will then decline as the confined fractured aquifer is drained. The long-term pumping rate intercepted by the system would eventually reduce to approximately 26 m^3/h. Horizontal drains into pit slopes have also been incorporated into the model. Simulated discharge from these drains increases to a maximum of 42 m^3/h in the final year of mining.

16.3.3.3 Limitations to Groundwater Modelling

There are several factors which constrain the pit slope depressurisation modelling:

- Limited groundwater data: particularly data on pore pressures with depth. Vibrating wire piezometers ('VWPs') will be installed within the footprint of the final pit to understand pre pressure responses, validate the current conceptual hydrogeological model and inform updates to future pore pressure modelling.
- Resolution of pit development: the current models are based on yearly time steps between pit shell stages. Increasing the resolution of the modelling by reducing the duration of each time step would increase confidence in the results and better simulate dynamic changes to the pore water pressures in response to mining.

- Geological uncertainty: the geological interpretation of the sub-surface will likely evolve based on further exploration and geotechnical drilling, and excavation of the pit itself. Of particular importance will be the review of the structural geology model to identify higher risk zones of the pit slopes, especially the footwall, and to provide a focus on the pore pressure monitoring.

Slope depressurisation aspects of groundwater are further covered in the geotechnical characterisation and design study undertaken by Bastion (2021), as summarised in Section 16.4.6. This study was completed after the hydrogeological study. Bastion considered the Large-Scale Structural Model to be at a conceptual level, with no 3D structural model. They defined two major shears as part of their study; structures which were not evaluated in the hydrogeological study and therefore not incorporated into the groundwater model.

16.3.4 Pit Water Management

The Lafigué pit water management system is designed to:

- Lower the groundwater table ahead of mining and reduce pore pressures behind pit walls;
- Remove ongoing groundwater seepage from the pit to the extent possible to minimize blasting costs and reduce operational costs associated with tyre wear etc.;
- Minimize the impact on mining associated with surface water accumulation in the pit in response to rainfall events; and
- Capture and divert stormwater runoff around the pit to prevent flooding.

Figure 16-4 shows the pit water management plan, including dewatering boreholes, horizontal drains, sump pumps and sump discharge pipe.

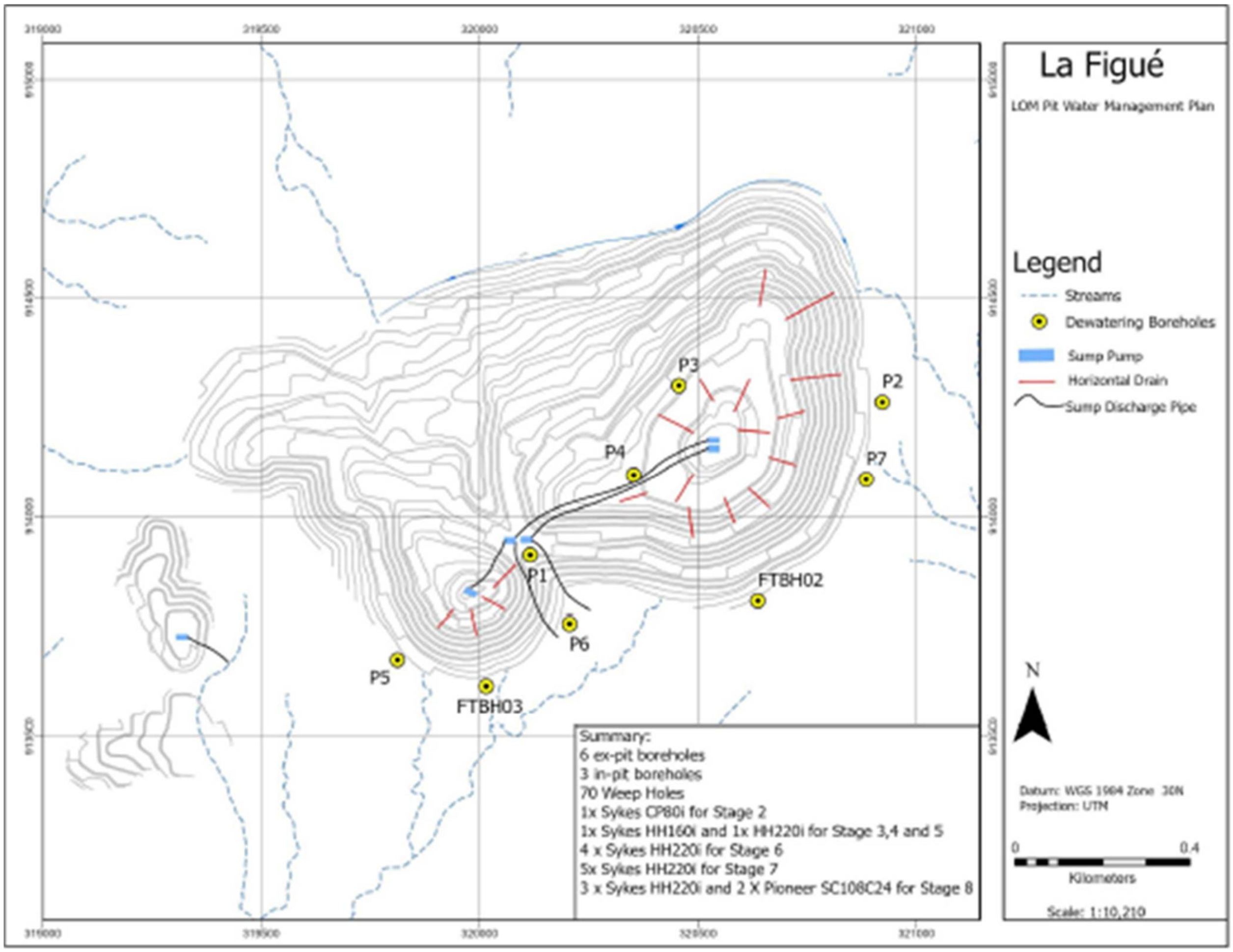


Figure 16-4: Lafigué Pit Water Management Plan

There are two broad contributions in need of consideration for pit dewatering: groundwater and surface water. Groundwater inflows comprise a relatively steady influx, although with some variation, whereas surface water inflows are controlled by climate conditions and the runoff characteristics of the catchment generating a highly variable and seasonal flux. The contribution of each is discussed more fully in Sections 16.3.4.1 to 16.3.4.3, following.

16.3.4.1 Groundwater Contribution

Groundwater flows will be captured as follows:

- Dewatering wells: Nine dewatering wells initially pumping 112 m^3/h to lower water levels pre-mining, pumping rates reducing to 26 m^3/h as water-bearing fractures are dewatered.
- Sub-horizontal drains: Across five mining sectors to drain pore pressures. The combined potential from such a system would be in the order of 2 to 42 m^3/h.
- Open ditches: To channel in-pit seepage that bypasses the interception system. This amounts to between 5 and 10 m^3/h, with an additional 2 to 42 m^3/h from the sub-horizontal drains.

The need for horizontal drains to reduce pore pressures across structures or dykes will need to be established based on operational experience and VWP monitoring. The number and spacing of any horizontal drains would be defined based on their observed effectiveness and the identification of structures to be targeted as part of the ongoing development.

16.3.4.2 Surface Water Contribution

Surface water entering the pit will be dewatered through in-pit sumps. Surface water inflows through the various stages of pit development have been estimated by applying appropriate runoff coefficients to the relevant catchments. It is assumed that any rainfall runoff generated outside of the pit footprint will be diverted away from the pit and is not managed as part of the dewatering system.

The surface water dewatering strategy is designed around managing the 1:100 year 24-hour rainfall event. It is assumed that all rainwater collected during the peak storm event would be evacuated from the pit floor within 5 days. The design would manage smaller rainfall events, monthly inflows during wet years and associated groundwater seepage bypass.

Based on the pumping requirements (Table 16-3), a combination of Sykes pumps (or similar specification pumps from other manufacturers) would be used for the in-pit sump dewatering system. The number of different pump models used was minimized where possible, to reduce the number of standby pumps required on-site and replacement pumps in the event of failure or maintenance. The proposed dewatering system for Stages 2 to 5 could dewater the pit in a single lift. Intermediate transfer pumping stations will be required for Stages 6, 7 and 8.

Table 16-3: Lafigué Open Pit Pumping Requirements

Pit Catchment	Units	STG2	STG3	STG4	STG5A	STG5B (STG3)	STG6	STG6B (STG5A)	STG6C (Transfer)	STG7	STG7B (STG6)	STG7 (Transfer)	STG8	STG8B (STG6)	STG8 Transfer
100-year 24 h event	mm	189	189	189	189	189	189	189	189	189	189	189	189	189	189
Time to dewater peak event	days	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Catchment area	m²	50300	271600	519000	539000	228000	397000	518000		652000	388000		752000	388000	
Groundwater inflow	m³	0	1920	1920	1344	576	3240	2160		4200	2400		5544	1800	
Precipitation to pump	m³	7605	41066	78473	81497	34474	60026	78322		98582	58666		113702	58666	
Total water inflow	m³	7605	42986	80393	82841	35050	63266	80482	143748	102782	61066	163848	119246	60466	179712
Total water inflow	m³/h	63	358	670	690	292	527	671	1198	857	509	1365	994	504	1498
Static Head (H_1)															
Top: Pipe max elevation	m	315	325	325	280	325	240	240	330	225	225	326	225	225	326
Bottom: Water elevation	m	300	260	260	200	260	135	200	235	140	135	220	20	135	220
Static Head (H_1)	m	15	65	65	80	65	105	40	95	85	90	106	205	90	106
Friction Head (H_2)															
Pipe length	m	60	100	100	180	100	200	80	220	130	160	200	500	160	200
Pipe nominal diameter	mm	140	225	315	315	225	315	315	315	315	315	315	315	315	355
Nominal pressure	bar	6	8	8	10	8	16	8	16	10	10	16	25	10	16
Dimension ratio		26	21	21	17	21	11	21	11	17	17	11	7	17	11
Pipe inside diameter	mm	129	204	285	278	204	258	285	258	278	278	258	230	278	290
Design flow rate per pump	m³/h	72	360	720	684	306	540	684	612	468	540	684	504	540	756
Fluid velocity (>2 and <3.5)	m/sec	1.5	3.1	3.1	3.1	2.6	2.9	3.0	3.3	2.1	2.5	3.6	3.4	2.5	3.2

Table 16-3: Lafigué Open Pit Pumping Requirements

Pit Catchment	Units	STG2	STG3	STG4	STG5A	STG5B (STG3)	STG6	STG6B (STG5A)	STG6C (Transfer)	STG7	STG7B (STG6)	STG7 (Transfer)	STG8	STG8B (STG6)	STG8 Transfer
Friction head loss/m		0.015	0.033	0.023	0.024	0.024	0.022	0.021	0.028	0.012	0.015	0.034	0.034	0.015	0.023
Friction head loss	m	0.9	3.3	2.3	4.2	2.4	4.4	1.7	6.1	1.5	2.4	6.8	16.9	2.4	4.6
Total Head	m	16	68	67	84	67	109	42	101	87	92	113	222	92	111
Pressure	bar	1.6	6.7	6.6	8.3	6.6	10.7	4.1	9.9	8.5	9.1	11.1	21.8	9.1	10.8
Power															
Power	kW	4	93	183	218	78	224	108	234	153	189	292	423	189	316
Power	HP	6	125	246	292	105	300	145	314	205	253	392	567	253	424
Pump efficiency	%	72	72	72	72	72	72	72	72	72	72	72	72	72	72
Pump Type		1x Sykes CP80i	1x Sykes HH160i	1x Sykes HH220i	1x Sykes HH220i	1x Sykes HH160i	1x Sykes HH220i	1x Sykes HH220i	2x Sykes HH220i	2x Sykes HH220i	1x Sykes HH220i	2x Sykes HH220i	2x Pioneer SC108C24	1x Sykes HH220i	2x Sykes HH220i

A total of seven Sykes and two Pioneer pumps will be required to dewater the pit sumps (note, this assumes optimization is carried out through pit development to re-use existing pumps in new stages); refer to Table 16-4.

Pit dewatering is included in the mining contractor rates, including in-pit dewatering capital and operating costs. Water quality and management of pit discharge water is dealt with in Section 8, Tailings and Water Management.

Table 16-4: Summary of Lafigué Pit Pumping Requirements

Pit Catchments	Total Head Requirement	Required Pump Capacity (m^3/h)	Selected Pump Arrangement
STG2	16	63	1x Sykes CP80i
STG3, 4 and 5	151	982	1x Sykes HH160i, 1x Sykes HH220i
STG6	252	1198	4x Sykes HH220i
STG7	292	1365	5x Sykes HH220i
STG8	425	1498	3x Sykes HH220i and 2x Pioneer SC108C24

16.3.4.3 Pit Perimeter Stormwater Management

The Lafigué open pit complex is located on the catchment divide and hence the pit development does not intercept upstream runoff. No specific designs are required to shed runoff from the pit perimeter accordingly, although crest berms should be installed to promote drainage downstream.

16.4 Open Pit Geotechnical Engineering Review

SRK has conducted a review and assessment of all Geotechnical material likely to influence the technical information included in the LoMp and Ore Reserves. The review of the Geotechnical data excludes independent verification by means of re-calculation. Whilst SRK has exercised all due care in reviewing the supplied information, SRK does not accept responsibility for finding any errors or omissions contained therein and disclaims liability for any consequences of such errors or omissions. The following section summarises the data reviewed and should not be taken as superseding the main authored report.

A geotechnical characterisation and design study for the Project, was undertaken in 2021 by Bastion Geotechnical ('BG') (Bastion, 2021) for the purpose of developing a feasibility study level geotechnical design criteria. The scope of the study included:

- Desktop study of all available geological and geotechnical data and reports.
- Review and re-log (where applicable) of recent geotechnical logging.
- Geotechnical logging and photo-logging of available legacy core.
- Laboratory testing liaison and data processing.
- Development of four basis models (Geological, Structural, Hydrogeological and Geotechnical models).
- Effect of shear strength models.
- Assessment of geotechnical domains and anticipated failure mechanisms.
- Design Acceptance Criteria (DAC) gateway.
- Limit Equilibrium (LE) slope stability analyses.
- Finite Element (FE) slope stability analyses.
- Recommendation of geotechnical design parameters.

Previously, Golder Associate (Pty) Ltd ('Golder') in 2020, developed the geotechnical design criteria (Golder Associates Africa (Pty), 7 September 2020). The 2021 feasibility study carried out by BG aimed to review the existing geotechnical data analysis, methodology, reporting and slope design criteria, assess the gaps and adjust inputs and design criteria as necessary.

16.4.1 Geological Model

BG consider the Geological Model to be at a design level of technical development, with five primary lithologies characterised within the project area:

- Weathered Rock Mass Unit ('RMU') (laterite, saprolite and saprock);
- Intrusive Mafic Rock: IMAF;
- Volcanic Mafic Rock: VMAF;
- Intrusive Felsic Rock: IFEL; and
- Volcanic Felsic Rock: VFEL.

Early pit stages will be formed primarily within the Weathered RMU, while in the final pit stage, the footwall is predominantly formed in IMAF and the hangingwall, IMAF and VMAF. IMAF is considered to be foliated and VMAF less strongly foliated. IFEL and VFEL are non-foliated.

16.4.2 Structural Model: Large-Scale Structures

BG considers the Large-Scale Structural Model to be at a conceptual level of technical development, with no 3D structural model provided to BG. BG defined two major shears (Shear 1 and Shear 2) and developed 3D surfaces as part of their study. Shear 1 is expected to impact slope stability on bench, inter-ramp, and overall slopes during the mining stages, while Shear 2 is likely to impact the slopes in the later mining stages. Both shears will mainly affect the footwall slopes. BG notes that the shears will be mostly mined out, but will remain in the footwall slope. The inter-ramp angle will consider such structures to ensure minimal undercutting.

16.4.3 Structural Model: Fabric

BG considers that the Structural Fabric Model is at Pre-Feasibility Study (PFS) level of technical development. Golder has previously discounted foliation, but work undertaken by BG has identified the presence of foliation, which has the potential to impact a design sector within the footwall. BG identifies a strong bias due to the majority of the boreholes with collars of 60°/000-340°, which affects data for the hangingwall slopes (but not the footwall). Foliation has been defined with an orientation of 35°/145°. No secondary joint sets have been described in the data presented by BG. As such, it would not be possible to undertake detailed bench and inter-ramp scale kinematic analyses.

16.4.4 Soil and Rock Mass Model

BG considers the Soil and Rock Mass Model to be at a Pre-Feasibility study Level of technical development. BG notes that the Weathered RMU is of limited thickness and will not materially impact slope stability. Within the fresh rock, structural conditions will control slope stability rather than circular failure through the rock mass. No internationally accepted soil logging system has been used to characterise the Weathered RMU and, BG notes that rock logging has captured appropriate data for use within Rock Mass Classification schemes; however, BG only undertook rock mass logging to a level of detail that would allow the development of Rock Mass Classification values in four boreholes (boreholes LFGT-08 to LFGT-11). At the same time, BG did not re-log boreholes previously logged by Golder (LFGT-01 to LFGT-07). Geological Strength Index (GSI) values have been defined as 80 for all rock units, indicating a very competent rock mass. Updated shear strength testing was undertaken as part of the BG study with basic friction angles developed for the IMAF and VMAF (32° and 34°, respectively). The Barton-Bandis Criterion (Nick & Stavros , 1980) was used to derive a Mohr-Coulomb equivalent shear strength of 60 kPa cohesion and 36° friction angle for the IMAF. At the batter scale, zero cohesion was applied. Reworking of Golder's Uniaxial Compressive Strength (UCS) testing that defined the Intact Rock Strengths (IRS) of (137 to 225) MPa.

BG applied isotropic slope stability analysis in the hangingwall and anisotropic analysis within the footwall.

16.4.5 Geotechnical Model

The upper benches will be mined in Weathered RMU (soil strength), and the fresh rock slopes primarily within IMAF, with some VMAF and fewer extents of IFEL and VFEL. The two identified footwall shears have the potential to interact with other, as yet unknown significant structures, which could lead to large-scale instability.

16.4.6 Slope Stability Analyses

The upper soil strength slopes will be sensitive to pore water pressure with circular analysis as the primary failure mechanism on a bench and inter-ramp scale. Kinematic analysis for planar instability identified footwall slope sectors Stage 3 DS2, Stage 4 DS2 and Stage 5 DS7 as sensitive to the presence of foliation planes on a bench scale, with the hangingwall unaffected by foliation. Inter-ramp slope stability analysis for the hangingwall returns a Factor of Safety in excess of 3 for a 52° Inter-Ramp Angle (IRA) and an IRA between 35° (Limit Equilibrium) and 38° (Finite Element) for a section of footwall affected by the foliation with slope depressurisation required. BG notes that the Limit Equilibrium anisotropic analysis can be considered conservative, and a 38° IRA is suitable for ongoing geotechnical validation and reconciliation during the design's implementation.

16.4.7 Seismicity

Seismic analysis was not considered in the geotechnical analysis undertaken by BG. This can be considered appropriate given the low levels of seismicity in Côte d'Ivoire and the fact that there have been few, if any, recorded instances in which earthquakes have been shown to produce significant slope instabilities in hard rock mines (Read, November 2009).

16.5 Mining Equipment

The recommended excavators for the Project are in a weight class of 100 to 200t that fall in the range of the Komatsu PC1250 and PC2000, supported by 100-t dump trucks Caterpillar 777 or Komatsu 785; however, with the changes in the Resource Model and the subsequent change in process requirements from 3 to 4 Mt/a (db), the equipment selection was revised to a larger fleet more suitable for bulk mining. An owner mining equipment and cost model was developed based on the equipment proposed in the preliminary contractor submission. The contractor proposed 150-t capacity Komatsu HD-1500 dump trucks with Komatsu PC3000 shovel for waste loading and 100-t capacity Komatsu 785-7 trucks with Komatsu PC2000 or equivalent for ore loading.

Other mining equipment configurations are being reviewed with detailed studies; however, the final mining fleet configuration and quantities depend on the agreed mining contractor. Still, the final equipment configuration and quantities should reasonably align with the results obtained from the owner mining equipment model.

It should be noted that the preferred excavator size for the mine, based on the orebody and selectivity requirements, is in the range of the Komatsu PC2000. Therefore, a larger ore excavator will result in an increased SMU; however, the impact would not significantly change the results of the DFS, due to the additional modifying factors already applied.

16.6 Haulage Analysis

The entire load and haul operation has been simulated using established pit face positions within MineSched®, associated haulage profiles and the Talpac® haulage simulation software from RPM Global. Talpac is an independent haulage fleet evaluation system. It has been designed to determine the productivity and economics of truck and loader hauling systems and is accepted as the global mining industry standard. The Talpac software package has a detailed database with most mobile equipment per manufacturer, type, and size. It is updated with the latest equipment models and associated technical data and specifications annually.

All the haul routes were plotted in Surpac as string files. These string files represent the haul road from the bottom of the pit to the primary crusher, the stockpiles, or the waste dump for each stage.

The productivity of the haul truck fleet will vary based on distances, gradients, material types, dump construction sequence, truck payload and cycle times.

The rolling resistance is an input for Talpac that influences the maximum speed and acceleration a haul truck can achieve for a specified haul segment. Table 16-5 and Table 16-6, following, summarises the rolling resistance inputs for various haul segments, and speed limitations per haul segment respectively applied in the haul cycle time study.

Figure 16-5 illustrates the haulage network that simulates the distance hauled on block-by-block bases, and Figure 16-6 the haulage cycle times per period.

Table 16-5: Rolling Resistance for Various Haul Segments

Segment type	Value	Description
Loading area	4.0%	50 m radius from the loader
In pit	3.0%	The flat segment from the loader area to the bottom of the first ramp
Ramps	2.5%	All the ramps at 10% gradient
Flat	2.0%	Rolling resistance of the haul routes on surface
Crusher area	2.0%	50 m radius from the crusher
On Dump	4.0%	The flat segment from the top of the dump ramp to the dumping area
Dumping area	4.0%	50 m radius surrounding the area where the load will be dumped

Table 16-6: Haul Truck Segment Speeds

Haul segment			Maximum speed (km/h)
Ramp	Up	Full	11
	Up	Empty	28
	Down	Full	30
	Down	Empty	37
Flat	Ex pit	Full	35
	Ex pit	Empty	50
	In pit/dump/load area/dump area	Full	25
	In pit/dump/load area/dump area	Empty	25
Switchback		Full	17
		Empty	19
Load/dump	Loader area	Full	25
	Dump area	Empty	25
Global speed limit			50

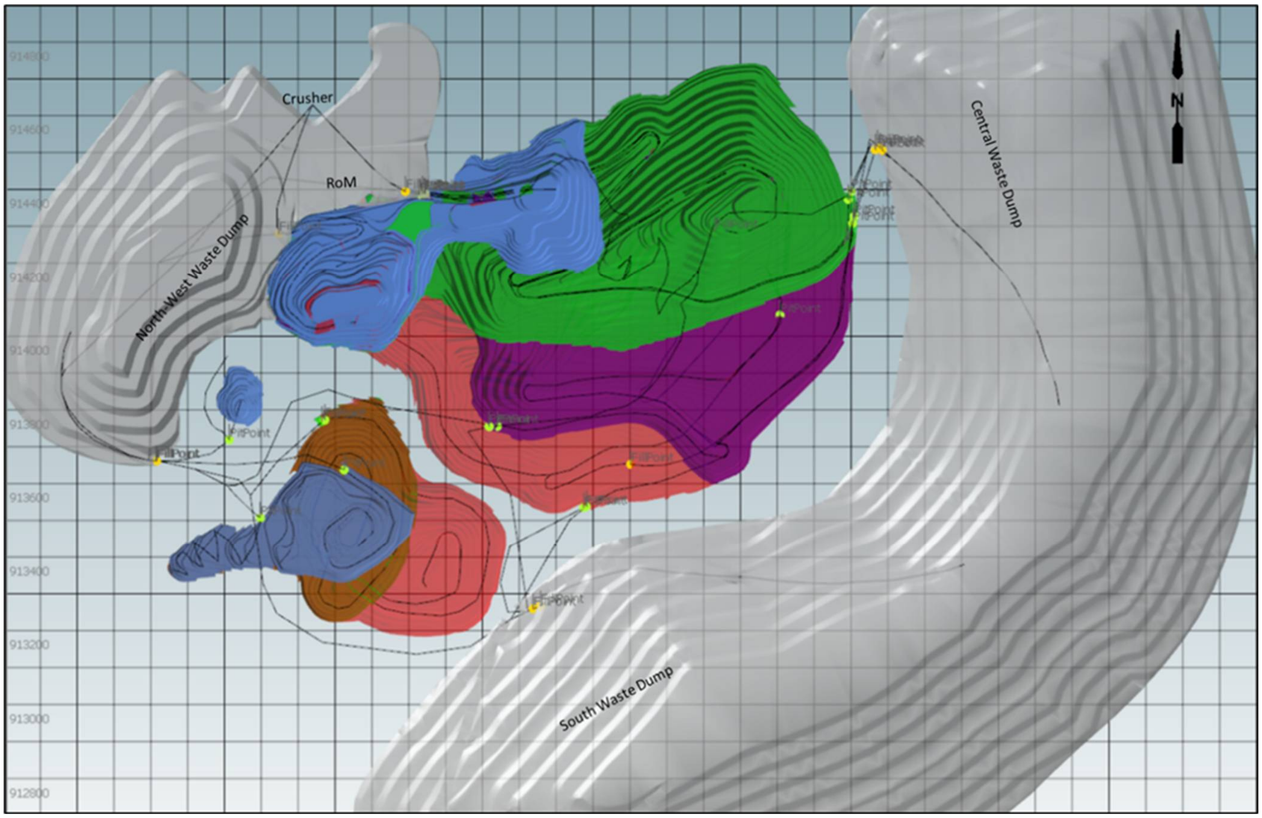


Figure 16-5: Haulage Network (image not to scale)

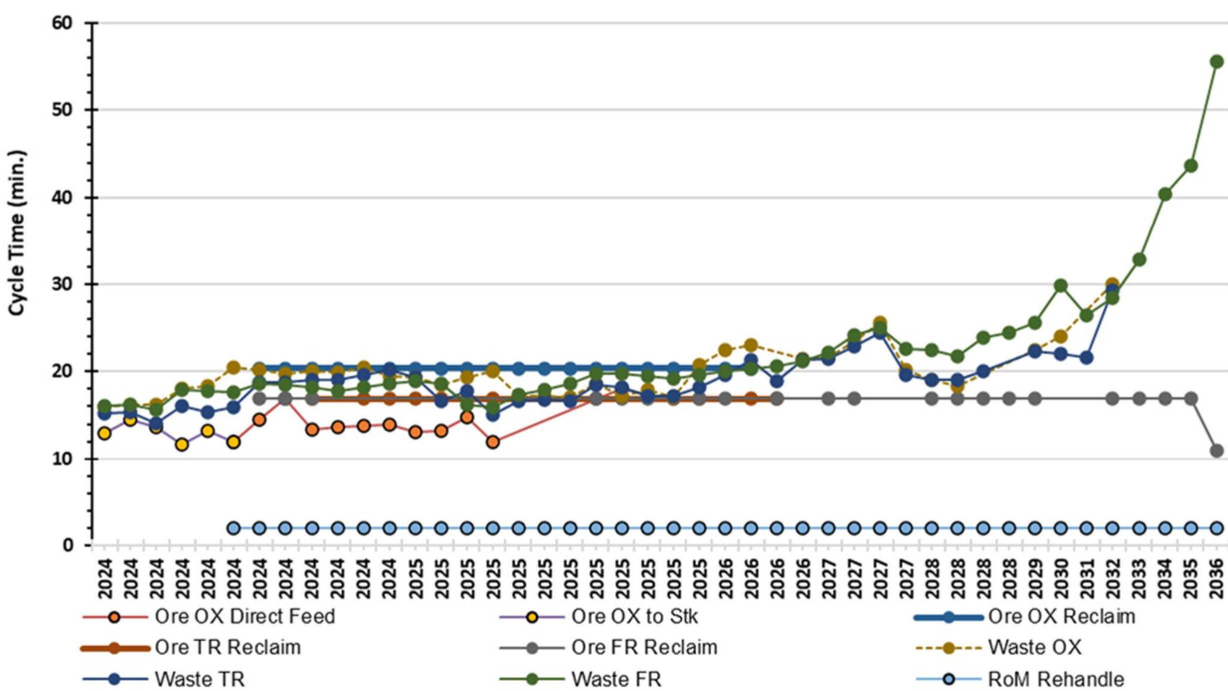


Figure 16-6: Haulage Cycle Time Results by Year

16.7 Equipment Operating Hours

It was necessary to make assumptions regarding how the shifts are operated to calculate the equipment's direct operating time per annum. Table 16-7 summarises the assumptions used for operating standby and use of availability, whilst Table 16-8 summarises the operational delays and utilisation assumptions. The time usage model for equipment availability and effective utilisation per equipment type is summarised in Table 16-9.

Table 16-7: Operating Standby & Use of Availability

Equipment	Shift Change	Meal Break	Fatigue Break	Safety Meetings	No Operator	Weather	Total Standby Hours	Operating Hours	Use of Aval.
	(h/shift)	(h/shift)	(h/shift)	(h/month)	(h/shift)	(day/a)	(h/a)	(h/a)	(%)
Waste Shovel	0.5	0.5		1.0		5.0	733	6713	90
Ore Shovel	0.5	0.5		1.0		5.0	733	6713	90
RoM Loader	0.5	0.5		1.0		5.0	733	6713	90
Stock Loader	0.5	0.5		1.0		5.0	733	6713	90
Waste Truck	0.5	0.5		1.0		5.0	733	6713	90
Ore Truck	0.5	0.5		1.0		5.0	733	6713	90
Large Drill	0.5	0.5		1.0		5.0	647	5924	90
Medium Drill	0.5	0.5		1.0		5.0	647	5924	90
Small Drill	0.5	0.5		1.0		5.0	647	5924	90
Track Dozer	0.5	0.5		1.0	1.0	5.0	1353	6093	82
Backhoe	0.5	0.5		1.0	1.0	5.0	1353	6093	82
Compactor	0.5	0.5		1.0	1.0	5.0	1353	6093	82
Motor Grader	0.5	0.5		1.0	2.0	5.0	1974	5472	73
Water Truck	0.5	0.5		1.0	2.0	5.0	1974	5472	73
Tire Handler	0.5	0.5		1.0	2.0	5.0	1974	5472	73
Fuel/Lube Truck	0.5	0.5		1.0	4.0	5.0	3215	4231	57
Service Truck	0.5	1.0	0.5	1.0	4.0	5.0	3835	3611	48
Lighting Plant					6.0	5.0	3825	3621	49
Light Vehicle					4.0	5.0	2584	4862	65

Table 16-8: Operational Delays & Utilisation

Equipment	Pre-Start (h/shift)	Blasting (h/d)	Travelling, Walking (h/shift)	Wait for Truck, Shovel, Crusher (h/shift)	Fuel, Water (h/wk)	Final Wall Scaling (h/d)	Other (h/shift)	Total Delay Hours (h/a)	Direct Operating Time (h/a)
Waste Shovel	0.1	0.5	0.3	0.3		0.6		949	5856
Ore Shovel	0.1	0.5	0.3	0.3		0.6		949	5856
RoM Loader	0.3	0.5	0.5	0.3				949	5764
Stock Loader	0.3	0.5	0.5	0.3				949	5764
Waste Truck	0.1	0.5		0.3	3.5			949	6075
Ore Truck	0.3	0.5		0.3	3.5		0.63	1186	5528
Large Drill	0.3	0.5	0.6		3.5			949	4975
Medium Drill	0.3	0.5	0.6		3.5			949	4975
Small Drill	0.3	0.5	0.6		3.5			949	4975
Track Dozer	0.3	0.5	1.0		3.5			1277	4457
Backhoe	0.3	0.5	0.5		3.5			912	4816
Compactor	0.3	0.5	0.3		3.5			730	5181
Motor Grader	0.3	0.5	0.3		3.5			730	5363
Water Truck	0.3	0.5	0.3		28.0			2004	4743
Tire Handler	0.3		0.3		3.5			547	4925
Fuel/Lube Truck	0.3	0.5	0.3		3.5			730	3502
Service Truck	0.3	0.5	0.3		3.5			730	2881
Lighting Plant	0.3				3.5			365	3257
Light Vehicle	0.3				3.5			365	4498

16.8 Equipment Productivities

Loader productivities depend on the truck they are paired with; therefore, a loader productivity estimate has been generated. The productivity estimate for the loading fleet for ore and waste is shown in Table 16-10 (pg. 16-466).

Table 16-9: Equipment Time Usage Model

Equipment List	Calendar Time (h/a)	Available Time (h/a)	Avail. (%)	Standby Time (h/a)	Operating Time (h/a)	Use of Avail. (%)	Delay Time (h/a)	Direct Operating Time (h/a)	Operating Efficiency (%)	Effective Utilisation (%)
Waste Shovel	8760	7446	85	733	6713	90	858	5856	87	67
Ore Shovel	8760	7446	85	733	6713	90	858	5856	87	67
ROM Loader	8760	7446	85	733	6713	90	949	5764	86	66
Stock Loader	8760	7446	85	733	6713	90	949	5764	86	66
Waste Truck	8760	7446	85	733	6713	90	638	6075	90	69
Ore Truck	8760	7446	85	733	6713	90	1,186	5528	82	63
Large Drill	8760	6570	75	647	5924	90	949	4975	84	57
Medium Drill	8760	6570	75	647	5924	90	949	4975	84	57
Small Drill	8760	6570	75	647	5924	90	949	4975	84	57
Track Dozer	8760	7446	85	1,353	6093	82	1,277	4816	79	55
Backhoe	8760	7446	85	1,353	6093	82	912	5181	85	59
Compactor	8760	7446	85	1,353	6093	82	730	5363	88	61
Motor Grader	8760	7446	85	1,974	5472	73	730	4743	87	54
Water Truck	8760	7446	85	1,974	5472	73	2,004	3469	63	40
Tire Handler	8760	7446	85	1,974	5472	73	547	4925	90	56
Fuel/Lube Truck	8760	7446	85	3,215	4231	57	730	3502	83	40
Service Truck	8760	7446	85	3,835	3611	48	730	2881	80	33
Lighting Plant	8760	7446	85	3,825	3621	49	365	3257	90	37
Light Vehicle	8760	7446	85	2,584	4862	65.3	365	4498	93	51

Table 16-10: Equipment Productivity per Material Type

Material		Oxide Ore	Transitional Ore	Fresh Ore	Oxide Waste	Transitional Waste	Fresh Waste	Oxide Waste	Transitional Waste	Fresh Waste
Loading Unit		PC2000	PC2000	PC2000	PC3000	PC3000	PC3000	PC3000	PC3000	PC3000
Bucket Size	(m^3)	0.0	0.0	0.0	0.0	0.0	0.0	12.0	12.0	12.0
Loading Spot Time	(min)	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75
Loading Cycle Time	(min)	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45	0.45
First Bucket Dump	(min)	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05
Haulage Unit		HD785	HD785	HD785	HD785-	HD785	HD785	HD1500	HD1500	HD1500
Capacity	(t)	100	100	100	100	100	100	191	191	191
Truck Fill Factor	(%)	100	100	100	100	100	100	100	100	100
Payload	(t)	92	92	92	92	92	92	142	142	142
Dump & Spot Time	(min)	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5
Loading Parameters										
Bucket Fill Factor	(%)	90	90	90	90	90	90	90	90	90
In-Situ Density	(t/bcm)	1.68	2.49	2.81	1.64	2.48	2.85	1.64	2.48	2.85
Swell Factor	(lcm/bcm)	1.10	1.30	1.45	1.10	1.30	1.45	1.10	1.30	1.45
Loose Dry Density	(dt/lcm)	1.53	1.92	1.94	1.49	1.91	1.97	1.49	1.91	1.97
Moisture Factor	(%)	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00
Passes	(#)	7	5	5	10	8	8	8	7	6
Loader Productivity	(Mt/a)	7.01	9.44	9.44	7.17	9.62	9.62	8.64	10.86	10.86
Loading rate	(t/h)	1044.68	1406.31	1406.31	1068.31	1433.42	1433.42	1286.82	1617.72	1617.72

Table 16-10 notes: all tonnes data are depicted as dry metric tonnes (dmt or t (db)) unless stated otherwise.

16.9 Blasting Operation

The primary role of blasting is to fracture the rock into fragments that can be efficiently excavated and handled by the downstream mining process. The nature of the rock mass largely influences the fragment size distribution resulting from blasting. Where the rock contains closely spaced joints and bedding planes, satisfactory fragmentation, displacement, and rubble pile looseness are usually achieved with a relatively low energy factor. Conversely, considerably higher energy factors and more shock energy are required where the rock is more competent and less fractured. As a result, a more significant number of new fracture surfaces have to be created to achieve the required degree of breakage.

In addition to these factors, other major blast design features that influence the outcome of a blast are the distribution of the explosive throughout the rock mass and the initiation and timing sequence of the explosive charges.

The design of any blasting plan depends on two variables: uncontrollable variables or factors such as geology, rock characteristics, regulations or specifications, the distance to the nearest structures, and controllable variables or factors. In addition, the blast design must provide adequate fragmentation to ensure that loading, haulage, and subsequent disposal or processing are accomplished at the lowest cost.

Further to the cost, the design of any blast must encompass the fundamental concepts of an ideal blast design and have the flexibility to be modified where necessary, to account for local geological conditions. The controllable and uncontrollable factors are discussed in this section and are used in the blasting and costing models wherever necessary.

An oxide portion (4.5%) within the production schedule was classed as free dig and will thus theoretically not require any blasting; however, drilling and blasting costs were applied to 12% of the free-dig material, which comprises all the laterite material.

16.10 Blast Design

The drilling and blasting productivity are based on the Epiroc 271 rotary drill rigs, predominantly working on waste and Epiroc D65 working on ore and waste, with an Epiroc T45 being used for pre-split. Table 16-11 presents the drilling and blasting parameters assumed for the study.

Table 16-11: Drill and Blast Productivity

Project Assumptions	Units	Large Drill Epiroc 271	Large Drill Epiroc 271	Large Drill Epiroc 271	Medium Drill Epiroc D65	Medium Drill Epiroc D65	Medium Drill Epiroc D65	Small Drill Epiroc T45
Material Properties		Oxide	Transitional	Fresh	Oxide	Transitional	Fresh	
• Density	(t/m³)	2.41	3.41	4.41	2.5			3.5
Input Parameters								
• Bench Height	(m)	10	10	10	10	10	10	20
• Hole Diameter	(mm)	203	203	203	152	152	152	102
• Sub-drill	(m)	1.00	1.00	1.00	0.00	1.20	1.20	0.00
• Spacing	(m)	9.00	7.10	6.67	8.00	5.80	5.41	1.20
• Burden	(m)	7.76	6.12	5.75	5.00	4.80	4.70	0.00
• Stemming Height	(m)	5.50	4.30	3.90	4.30	3.50	2.90	0.00
• Re-drill/Drilling Overlap Factor	(%)	10	10	10	10	10	10	10
Drilling								
• Hole Depth	(m)	11.00	11.00	11.00	10.00	11.20	11.20	20.00
• Volume Rock per Hole	(m³)	698.28	434.57	383.53	400.00	278.40	254.27	24.00
• Quantity Rock per Hole	(t)	1173.10	1082.08	1077.71	656.00	690.43	724.67	68.40
• Yield of Rock	(m³ rock/m drilled)	63.48	39.51	34.87	40.00	24.86	22.70	1.20
• Yield of Rock	(t rock/m drilled)	106.65	98.37	97.97	65.60	61.65	64.70	3.42
• Penetration Rate	(m/h)	30	30	30	30	30	30	30
• Drill time per Hole	(min)	25.87	25.87	25.87	23.83	26.27	26.27	49.60
• Productivity per metre	(m/doh)	24.91	25.41	25.88	25.17	25.58	25.58	24.19
• Productivity per tonne	(t/doh)	2656.33	2499.89	2535.42	1651.47	1576.73	1654.92	82.74
	(Mt/a)	13.22	12.44	12.61	8.22	7.84	8.23	0.41

Table 16-11: Drill and Blast Productivity

Project Assumptions	Units	Large Drill Epiroc 271	Large Drill Epiroc 271	Large Drill Epiroc 271	Medium Drill Epiroc D65	Medium Drill Epiroc D65	Medium Drill Epiroc D65	Small Drill Epiroc T45
Blasting								
• Stemming Volume	(m^3)	0.18	0.14	0.13	0.10	0.14	0.15	
• Volume of Charge	(m^3)	0.18	0.22	0.23	0.10	0.14	0.18	
• Charge Height	(m)	5.50	6.70	7.10	5.70	7.70	8.30	
• Charge per Hole	(kg)	222.51	271.06	287.24	129.29	174.65	188.26	
• Powder Factor	(kg/m^3)	0.32	0.62	0.75	0.32	0.63	0.74	
• Powder Factor	(kg/t)	0.19	0.25	0.26	0.20	0.25	0.26	

16.11 Equipment Requirements and Replacement

Based on the productivity figures outlined in Section 16.8, equipment requirements throughout the LoMp, based on the required fleet totals are presented in Table 16-12, following. In addition, equipment replacement was calculated based on the direct operating hours in relation to the equipment life cycle hours.

Table 16-12: Equipment Requirements per period

Equipment	Units	Max.	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032
Waste Shovel	(#)	**5**	-	5	5	5	5	5	5	5	4	4
Ore Shovel	(#)	**1**	-	1	1	1	1	1	1	1	1	1
ROM Loader	(#)	**1**	-	1	1	1	1	1	1	1	1	1
Stock Loader	(#)	**1**	-	1	1	1	1	1	1	-	-	1
Waste Truck	(#)	**23**	-	16	16	17	19	21	21	23	19	20
Ore Truck	(#)	**8**	-	4	5	4	6	5	5	6	7	6
Large Drill	(#)	**4**	-	4	4	4	4	4	4	4	3	3
Medium Drill	(#)	**3**	-	3	3	3	3	2	2	2	2	2
Small Drill	(#)	**1**	-	1	1	1	1	1	1	1	1	1
Track Dozer	(#)	**10**	-	7	7	7	8	9	9	10	9	9
Backhoe	(#)	**2**	-	2	2	2	2	2	2	2	2	2
Compactor	(#)	**1**	-	1	1	1	1	1	1	1	1	1
Motor Grader	(#)	**4**	-	3	3	3	3	4	4	4	4	4
Water Truck	(#)	**4**	-	3	3	3	3	4	4	4	4	4
Tire Handler	(#)	**1**	-	1	1	1	1	1	1	1	1	1
Fuel/Lube Truck	(#)	**2**	-	2	2	2	2	2	2	2	2	2
Service Truck	(#)	**1**	-	1	1	1	1	1	1	1	1	1
Lighting Plant	(#)	**21**	-	21	20	20	20	20	20	19	16	17
Light Vehicle	(#)	**10**	-	10	10	10	10	10	10	10	10	10

16.12 Labour

A full calendar (four panel shifts) was selected for the operation, considering equipment utilisation and fatigue management. Production personnel numbers for the mining contractor are based on a 6 day on 2 day off roster, with management based on a 42 day on 14 day off roster.

The mining contractor labour requirement over the LoM reaches a maximum of 699 personnel at steady state, whilst the blasting contractor will employ some 37 personnel. The owner mining personnel numbers reach 47 and comprise of people/teams allocated to Mineral Resource Management, mine production/contract management and Mining Technical Services.

A breakdown of labour numbers by business function is presented in Section 24 of this Report.

16.13 Mine Infrastructure

Mine infrastructure for the Project is discussed in Section 18 of this Report.

16.14 Mine Schedule and Associated Assumption

16.14.1 Introduction

This section discusses the production ramp-up, steady state and plant feed. A number of schedules have been developed for the Project, which has built up a considerable understanding of the most suitable approach for mining the Lafigué deposit.

Various production schedules were compared and traded off based on the following requirements:

- mining ramp-up and pre-strip requirements;
- plant feed tonnages and grade;
- stockpile movement and inventory;
- waste tonnes and total material mined;
- number of active benches and vertical advance; and
- optimised overall NPV for the Project.

The schedule was developed using Geovia's Minesched® scheduling software as the primary scheduling tool. Schedules were analysed on the cost of mining capital (equipment and pre-strip requirements) whilst considering unit mining operating costs and vertical pit advance rates to sustain plant throughput requirements and optimise project value.

The production schedule was developed monthly for 2024 and 2025, quarterly for 2026, 2027 and 2028, and annually thereafter.

16.14.2 Schedule assumptions and parameters

Adopting a rules-based iterative approach through Minesched allows one to determine an appropriate production schedule which provides the optimal economic value, whilst balancing practical mining constraints, particularly bench turnover rates and capital expenditure during ramp-up periods.

MineSched® develops schedules based on the following:

- A set of user definitions and objectives such as material types, movement and priorities, fleet capacity, etc.
- On a block-by-block as opposed to a bench-by-bench scheduling approach. Consequently, the software does not need to utilise bench averaging. Instead, the actual grades and actual strip ratios are reported in any reporting period.
- Any schedule generated by Minesched adheres to set Parameters, Precedences and special relationship rules to improve the practicality of the schedule.
- Capacity constraints and targets can be used to control the tonnes, volumes of content being mined for Flagged material types.

16.14.3 Defined Grade Envelopes

Various material types were developed to improve material blending to the process plant during scheduling. Materials were based on weathering type; Laterite, Saprolite, Transitional and Fresh rock. Ore material was based on classification, and CoG was split into various grade envelopes or bins. These material types are shown in Table 16-13.

Table 16-13: Defined Grade Envelopes

Material type	Description	CoG (g/t Au)
Laterite		
• LT_HG	High Grade	>= 1.5
• LT_MG	Medium Grade	>0.8 & <=1.5
• LT_LG	Low Grade	>0.4 & <=0.8
• LT_MO	Marginal Ore	na
Saprolite		
• SP_HG	High Grade	>= 1.5
• SP_MG	Medium Grade	>0.8 & <=1.5
• SP_LG	Low Grade	>0.4 & <=0.8
• SP_MO	Marginal Ore	na
Transitional		
• TR_HG	High Grade	>= 1.5
• TR_MG	Medium Grade	>0.8 & <=1.5
• TR_LG	Low Grade	>0.4 & <=0.8
• TR_MO	Marginal Ore	na
Fresh		
• FR_HG	High Grade	>= 1.5
• FR_MG	Medium Grade	>0.8 & <=1.5
• FR_LG	Low Grade	>0.5 & <=0.8
• FR_MO	Marginal Ore	>0.4 & <=0.5
Waste		
• LT_WST		Any
• SP_WST		Any
• TR_WST		Any
• FR_WST		Any

16.14.4 Material Movement

The current schedule locations were divided into various sub-sections based on elevation. Splitting the locations on elevation was primarily done to refine the waste schedule to improve haulage distances, rather than manage the strip ratios.

Stockpiles were added for each material type and grade bin, with all material scheduled from pit to stockpile and then re-handled to the process plant (Figure 16-7). This will not be the case in reality, but was done to allow for flexibility to manipulate re-handling later. Not simulating the re-handling was required to compare the Cat 777 and Cat 789 fleet options accurately, which may require changes to the Process plant's RoM feed bin.

Stockpile priority was assigned the highest priority on HG and systematically working down to MO that was left till the end of mining life. This strategy ensures that the highest potential feed grade is maintained.

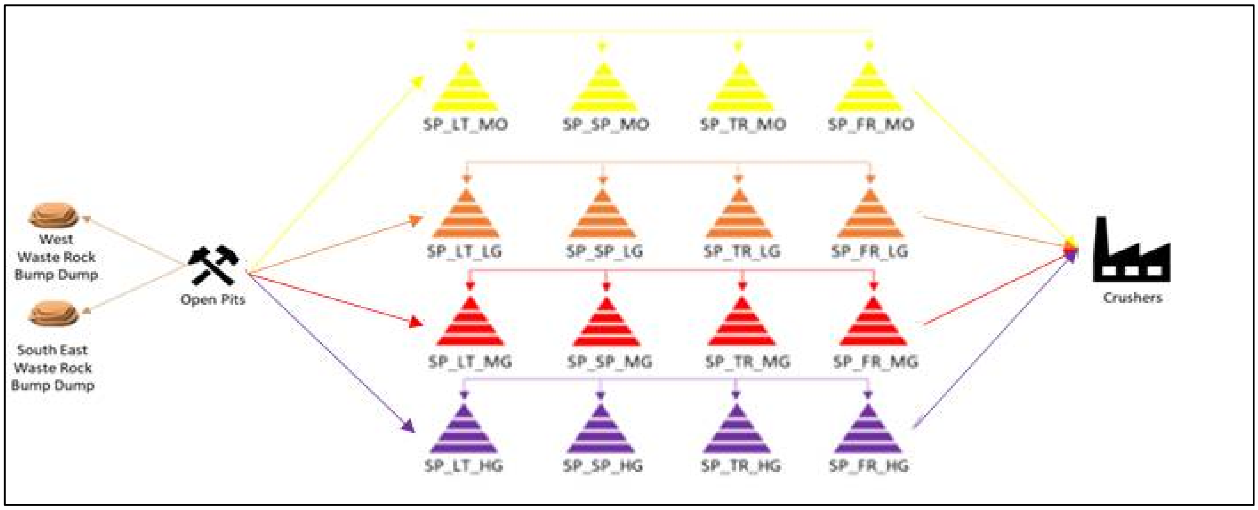


Figure 16-7: Material Movement

16.14.5 Schedule Parameters

Two primary schedule parameters were applied:

- Maximum active benches are limited to between four and six mining benches, depending on the material type. This function also limits the number of benches that can be mined in any one location per period and controls the mining activities within a specific area.
- Maximum benches per period, limit the number of benches that can be active per period, thereby controlling the bench turnover rate. When the maximum number of benches is reached, the schedule will not start a new bench till the next period. As a result, the bench turnover rate was limited to 80 and 100 m per year.

16.14.6 Schedule Precedence's

Two primary schedule precedences were applied:

- Horizontal lag in all directions controls the shape of the mining face as a bench advances. Schedule horizontal lag was set between 100 to 250 m opening benches, to accommodate multiple excavators on a bench.
- Vertical lag in all directions controls the minimum distance between adjacent advancing benches. Vatical lag was set at 60 m, allowing enough space on an advancing bench to conduct multiple activities safely.

16.14.7 Spatial Relationships

Spatial relationships are used to define the relationship between two mining locations. This assists the scheduling software to know how various locations should interact. Spatial relationships ensure no undermining between two sequential pushbacks occur and connect the schedule precedence between the various mining locations.

16.14.8 Max Capacity and Mining Ramp-up

The maximum required capacity was tested between 50 and 55 Mt/a (db) to sustain plant throughput requirements. As a result, the current ramp-up rate equates to 74% of total capacity based on 55 Mt/a in year 1. The monthly percentage of total capacity is illustrated in Table 16-14, following.

For the DFS, the focus was to determine the required pre-strip to sustain the process plant throughput and blend requirements in the first year. The required ramp-up rate and duration were determined based on the pre-strip needs. The current projected timeline for mobilisation will form part of the Project's critical path and is a risk to the first year of production.

Table 16-14: Mining Ramp-up

Month	Percentage of Full Capacity (%)
1	20
2	58
3	76
4	87
5	90
6	100

16.14.9 Process Blend and Throughput Rate

Process throughput rate was increased from the original PFS of 3.3 Mt/a (db) to 4 Mt/a (db) to maintain relatively the same ounce profile due to the drop in average grade, within the revised Resource model. Plant ramp-up is planned as illustrated in Table 16-15, with the first month at 80% throughput and maintaining a full capacity of 4 Mt/a (db) thereafter. The minimum turn-down capacity was capped at 3.8 Mt/a (db) if insufficient fresh material was available to maintain the feed blend.

The Oxide process blend requirements must be restricted to less than 20 to 25% of total feed to ensure optimal performance of the High-Pressure Grinding Rolls ('HPGR'). The HPGR needs resistance to work against, and with no rocks in the feed, cannot be pushed apart. To overcome the above-mentioned constraint, a mobile crusher will be acquired (the same one that will be used for the blast hole stemming material), oxide and transitional material will be sent through the mobile crusher directly into the ball mill, thus by-passing the HPGR. Bypassing the HPGR allows the total oxide material in the feed blend to be increased to 200 t/h (db) or 2 Mt/a (db).

Table 16-15: Processing Ramp-up

Year	Percentage of Full Capacity (%)	Activity
1	0	Pre-production
2	98	Commissioning one month at 80% before total production
Remaining	100	Production

16.14.10 Gold Production

Various schedule scenarios were developed to derive the optimal schedule selected and discussed in more detail later within this section. All the strategic Whittle schedule scenarios and the initial tactical schedule following the guidance from the strategic schedules, resulted in a significantly fluctuating ounce profile ranging between 130 and 350 koz/a. The fluctuating ounce profile is primarily due to the nature of the deposit, which comprises of disseminated HG lenses, with LG gaps in between, as illustrated previously in Figure 15-11. This occurrence of HG and LG lenses results in large quantities of HG material being mined in one period followed by LG material, which results in LG stockpile material being fed until the next HG lens is intersected.

As a result, gold production was limited to between 200 and 250 koz/a, so as to present a more conservative and sustainable mining schedule. For this reason, a direct comparison just on NPV between the various schedule scenarios will not favour the sustainable scenario.

16.15 Mining Production Schedule

16.15.1 Introduction

Various schedule scenarios were developed during the DFS, but during the DFS, multiple aspects changed, making a direct comparison between scenarios challenging. The different schedule scenarios along with the main differences between each, are as follows:

- The Base Case used the DFS block model (June 2021) and designs with the same pre-strip, process capacity and mining rates as the PFS.
- Scenario 1 assumed that all oxide material would be fed at process start-up before the commissioning of the HPGR. No gold capping was applied.
- Schedule 2 to 5 was based on an updated September 2021 block model and pit designs, with restricted levels of oxide feed to the HPGR. These schedules also tested various pre-strip and mining rates, while also applying a limit to the gold production of 230 koz/a, with 10% variation allowed.
- Scenario 6 incorporated some design changes to the ramps of main pit pushback 2, to gain quicker access to fresh material to sustain the plant feed during 2024 and 2025. Oxide is still limited in the feed blend.
- Scenarios 7 and 8 incorporated an updated May 2022 block model, while moving the process start date from January 2024 to April 2024 in scenario 7, and in scenario 8, to the confirmed date of June 2024.
- Scenario 9 incorporates oxide feed through the mobile crusher, increasing the oxide feed in 2024 and 2025.
- Scenario 10 increased mining in 2024 and 2025 to achieve 200 koz in 2026.
- Scenario 11 did not mine the high strip Pit B pushback 3.
- Scenario 13 updated design is based on a USD 1175/oz Au pit shell, with the latest contractor costs incorporated.

The main finding throughout the various schedule scenarios was that during the second to third year of production, while mining the main pit pushback 2, the grades decrease while waste stripping continues to access the next higher-grade lens. This resulted in lower ounces being produced during this time, which was consistent throughout all scenarios.

Various sequence changes were assessed, but any change in sequence resulted in a delay in accessing the higher grades. The only way to overcome this was by increasing the pre-strip tonnes or increasing the total tonnes mined during the first two years.

With the restriction on oxide material in the feed, the pre-strip tonnes had to be increased from the base case (PFS) 15 Mt to 22 Mt, so as to sustain the plant throughput rate. With the increase of oxide material in the feed blend, the required pre-strip tonnes were reduced back down to 15.6 Mt.

The decrease in the pre-strip volumes, delayed access to the higher grades, with a gold production decrease still occurring. To overcome this decrease, an additional 6 Mt of total material mined is required to access the HG material earlier and improve the ounces produced. As a result, schedule scenario 10 with an increased mining rate in 2024 and 2025 (65 Mt/a) showed a lower overall NPV, as the increased ounces do not offset the cost of the additional mining.

Due to the differences in schedule scenarios 1 to 9 and inaccuracy of direct comparisons, only the final schedule scenarios are compared in Table 16-16 (NPVs are indicative, actual evaluation can be in found in Section 21, Capital and Operating Cost Estimate). Therefore, only the final selected schedule, scenario 13, will be discussed in detail further in this section.

Table 16-16: Mine Schedule Comparison

Item	Units	Schedule 9	Schedule 10	Schedule 11	Schedule 13
Contained Gold Processed	(koz)	2742	2742	2659	2587
LoM Head Grade	(g/t Au)	1.68	1.68	1.67	1.69
Maximum Throughput	(Mt/a db)	4.0	4.0	4.0	4.0
LoM Recovery	(%)	95.0%	95.0%	95.0%	95.0%
Lom Strip Ratio	(t:t)	9.16	9.16	8.91	8.87
LoM Mined Grade	(g/t Au)	1.68	1.68	1.67	1.69
Gold Production					
• 2024	(koz)	139	145	145	139
• 2025	(koz)	213	212	212	223
• 2026	(koz)	169	196	196	177
• 2027	(koz)	223	209	209	230
• 2028	(koz)	239	220	220	233
Total Gold – first 3 Years	(koz)	521	553	553	539
Total Gold – first 5 Years	(koz)	983	982	982	1002
Mine Life	Years	13.0	13.0	12.5	13.0

16.15.2 Schedule Results

The final selected schedule for the DFS is discussed in more detail in this section. All production data is depicted as dry tonnes (dry metric tonnes or t (db)) unless stated otherwise.

Figure 16-8 shows the ex-pit movement by location. It peaks at around 55 Mt/a (db) with the variation in part due to different productivities by weathering type in Figure 16-9. Mining typically occurs in multiple stages and is primarily driven by high tonnage movement and slow sinking rate (maintaining enough working space). Figure 16-10 following shows ore tonnes mined by location, and in Figure 16-11, total ex-pit inferred tonnes.

Figure 16-12 shows the average sink rate per period. Depending on material weathering, sink rates were restricted to between 80 and 100 m. The initial sink rate is high during pre-strip, due to the initial topography and high percentage of oxides being mined. Other spikes in 2027 to 2030 are also due to oxides being mined during the start of pushback 3 and 4.

Table 16-17 summarises the annual mining production schedule, with Table 16-18 illustrating the annual mine progression.

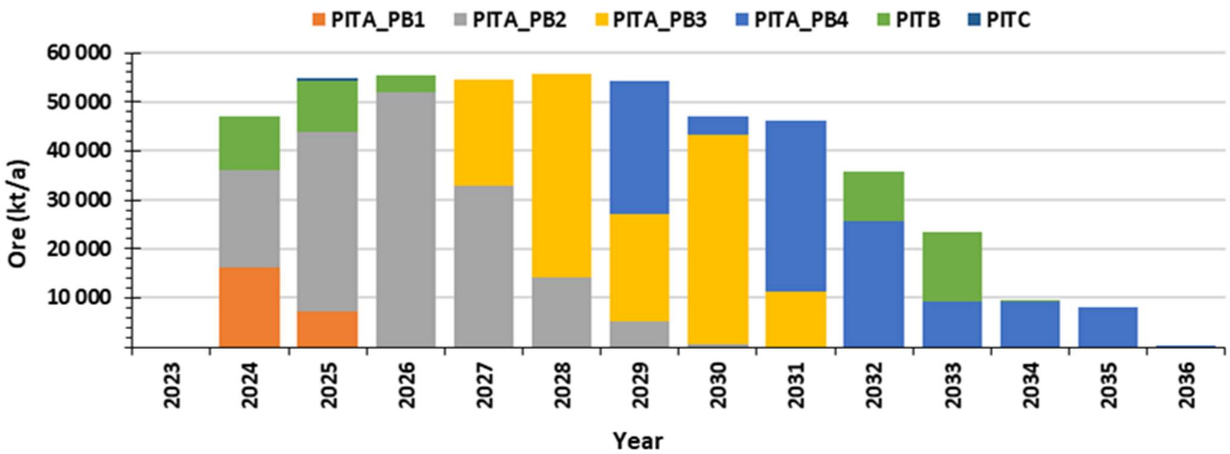


Figure 16-8: Total Ex-Pit Movement by Stage

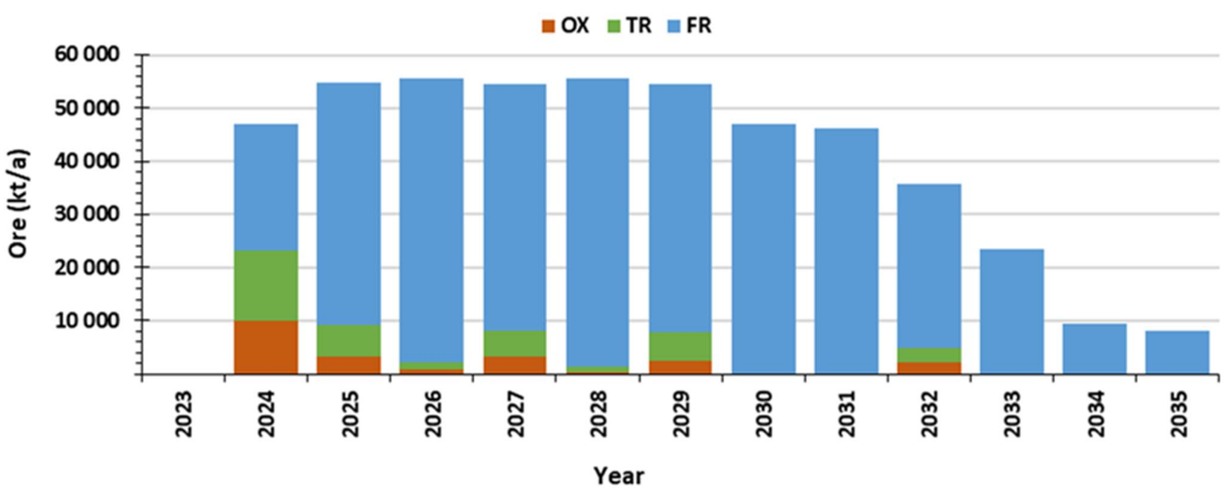


Figure 16-9: Total Ex-Pit Movement by Weathering

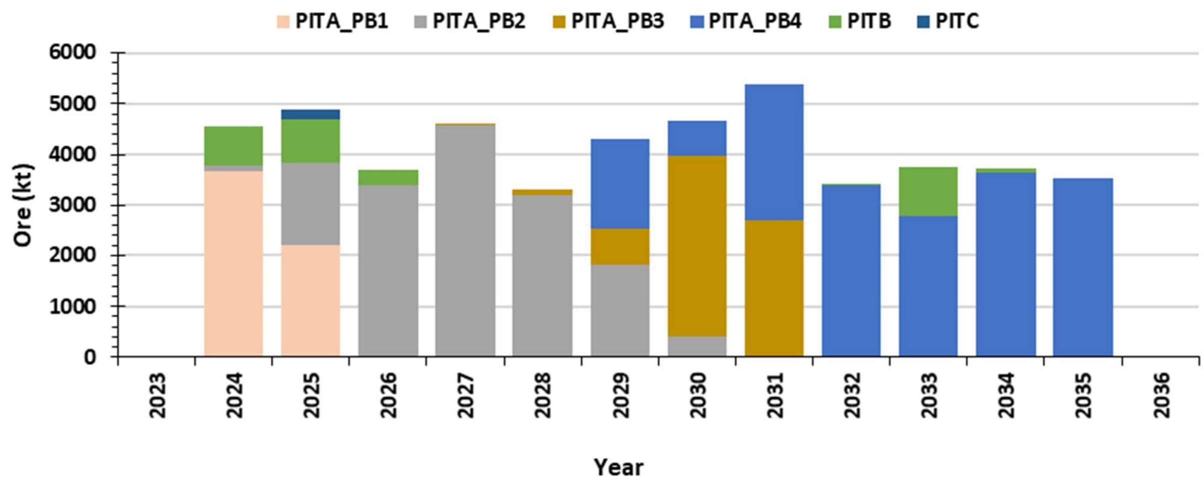


Figure 16-10: Total Ex-Pit Ore Movement by Location/Pit

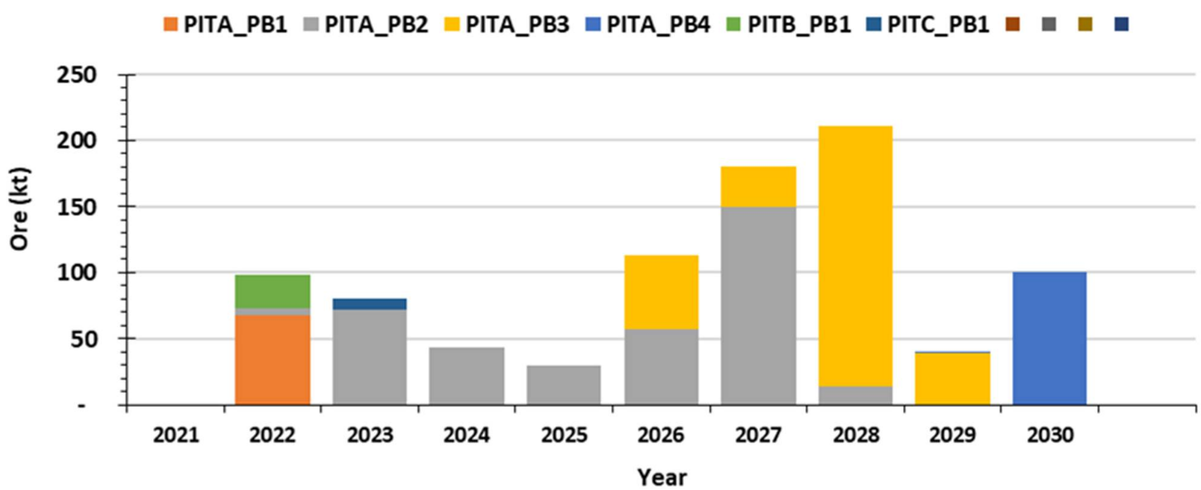


Figure 16-11: Total Ex-Pit Inferred Ore Mined by Location/Pit

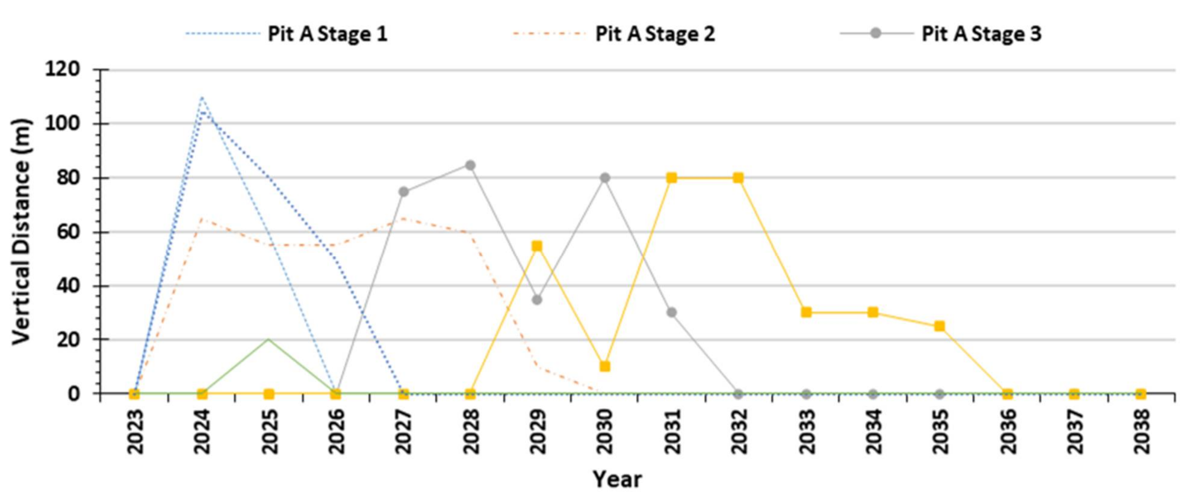


Figure 16-12: Bench Sink Rates

Table 16-17: Annual Mining Schedule

Mining Schedule	Units	Totals	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Total volume	(kbcm)	180 584	19 957	20 385	19 779	20 305	19 633	19 982	16 538	16 185	13 293	8250	3340	2924	14
Total Tonnes	(kt)	491 615	47 072	54 750	55 480	54 518	55 632	54 385	46 926	46 070	35 816	23 430	9401	8097	37
• Oxide	(kt)	22 390	10 085	3381	835	3151	258	2351	0	0	2330	0	0	0	0
• Transition	(kt)	34 248	13 092	5813	1276	4887	1167	5369	10	41	2496	96	0	0	0
• Fresh	(kt)	434 977	23 895	45 557	53 369	46 480	54 207	46 665	46 916	46 029	30 990	23 333	9401	8097	37
Waste mined	(kt)	441 802	42 529	49 878	51 779	49 941	52 322	50 079	42 263	40 683	32 409	19 677	5671	4562	10
Operating strip ratio	(t:t)	8.87	9.36	10.24	13.99	10.91	15.81	11.63	9.06	7.55	9.51	5.24	1.52	1.29	0.35
Feedable ore mined	(kt)	49 813	4543	4872	3701	4577	3310	4306	4664	5386	3407	3753	3731	3535	28
Feedable ore grade	(g/t)	1.69	1.37	1.57	1.40	1.70	2.16	1.78	1.57	1.44	2.18	1.89	1.90	1.71	1.96
Total ore mined	(kt)	49 813	4543	4872	3701	4577	3310	4306	4664	5386	3407	3753	3731	3535	28
Total ore grade	(g/t)	1.69	1.37	1.56	1.40	1.70	2.16	1.78	1.57	1.44	2.18	1.89	1.90	1.71	1.96
Ounces mined	(koz)	2714	200.52	245.10	166.59	249.88	230.34	245.85	235.75	249.36	238.28	228.24	228.07	194.60	1.76
MI	(koz)	2715	200.66	245.16	166.59	249.88	230.34	245.85	235.75	249.36	238.28	228.24	228.07	194.60	1.76
MIF	(koz)	38	3.07	2.23	0.96	0.64	4.09	12.78	6.57	1.99	4.36	0.83	0.41	0.09	0.00
Gold recoverable	(koz)	2584	190.48	233.19	158.39	237.84	219.71	234.27	224.30	236.94	227.52	217.44	217.41	185.31	1.68
HG - Ore tonne	(kt)	18 328	1522	1641	1012	1849	1656	1575	1627	1732	1381	1622	1450	1252	10
HG - Ore grade	(g/t)	3.23	2.50	3.09	3.05	3.02	3.46	3.52	3.03	2.75	4.22	3.30	3.63	3.41	3.87
MG - Ore tonne	(kt)	15 413	1559	1540	1223	1283	980	1193	1376	1941	1006	1115	1087	1100	12
MG - Ore grade	(g/t)	1.04	1.03	1.04	1.05	1.06	1.07	1.03	1.07	1.04	1.01	1.05	1.06	1.02	1.16
LG - Ore tonne	(kt)	12 179	1335	1276	1094	1077	533	1141	1193	1261	713	771	911	869	3
LG - Ore grade	(g/t)	0.61	0.58	0.61	0.61	0.63	0.62	0.61	0.61	0.62	0.61	0.61	0.61	0.61	0.55

Table 16-17: Annual Mining Schedule

Mining Schedule	Units	Totals	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Marginal-Ore - Ore tonne	(kt)	3892	127	415	373	368	141	398	468	452	307	245	282	314	3
Marginal-Ore - Ore grade	(g/t)	0.43	0.43	0.43	0.43	0.43	0.43	0.42	0.43	0.43	0.43	0.42	0.43	0.43	0.42
NF-Inferred tonne	(kt)	953	98	81	43	30	113	180	211	40	100	34	20	2	0
NF-Inferred graded	(g/t)	1.24	0.97	0.86	0.68	0.66	1.12	2.20	0.97	1.56	1.36	0.77	0.65	1.18	0.00
NF-Inferred ounce	(koz)	38	3.07	2.23	0.96	0.64	4.09	12.78	6.57	1.99	4.36	0.83	0.41	0.09	0.00
Mining+GC for forecast	(USD M)	1236	108	121	126	131	137	137	131	123	98	65	32	28	0
Mining+GC Unit Cost	(USD/t)	2.51	2.31	2.22	2.27	2.40	2.47	2.51	2.77	2.67	2.74	2.80	3.30	3.33	2.71

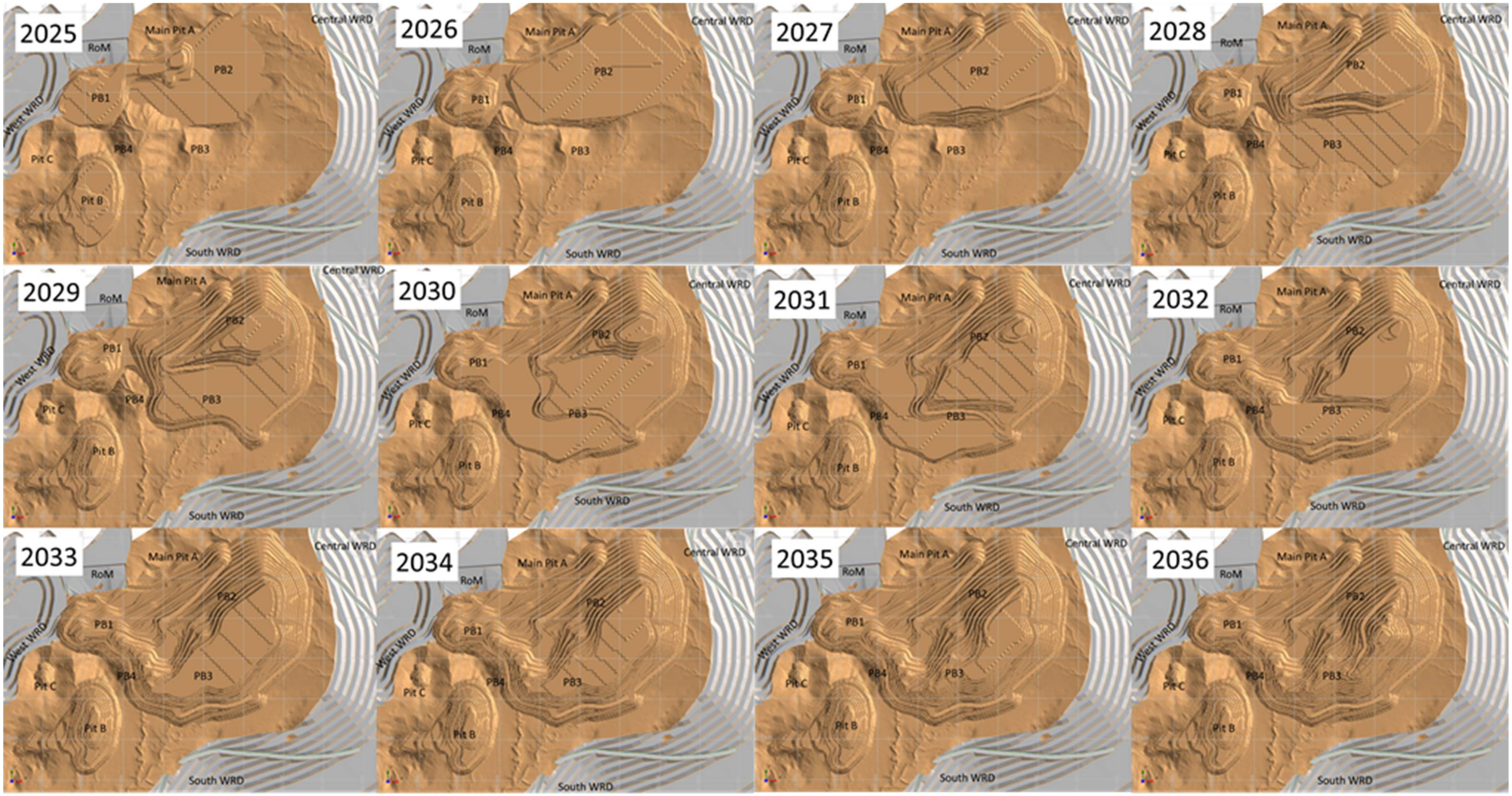


Figure 16-13: Annual Mine Progression

Figure 16-14 and Figure 16-15 following, illustrate the stockpile balance and movement by grade envelope and weathering type. The grade is shown as Fully Graded Ore ('FGO') above economic cut-off and FGO with marginal ore ('MO'). The MO ore stockpiles are only processed at the end of the mining life, when mining stops and the overall cost structure decreases. The peak of 4.9 Mt at 0.48 g/t Au, occurs in 2031. Therefore, the only material in the stockpile is LG and MO ore material.

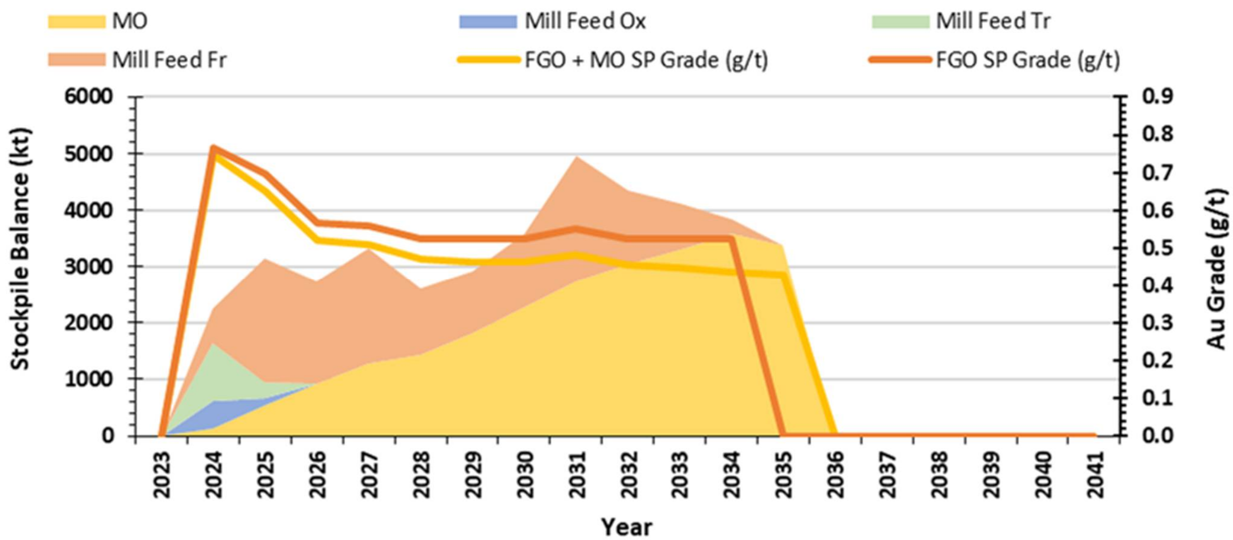


Figure 16-14: Long Term Stockpile Balance by Weathering Type and Grade

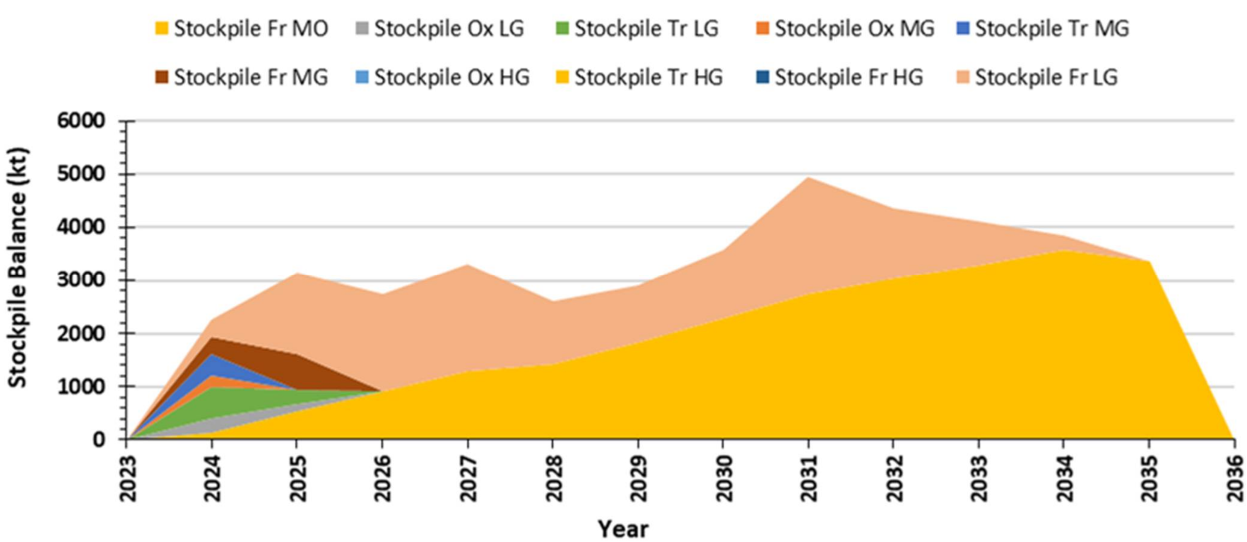


Figure 16-15: Long-Term Stockpile Balance by Grade Envelope

16.15.3 Processing Schedule

Figure 16-16 shows the ore mined and mined grade, process feed and feed grade. The feed grade is predominately between 1.4 and 2.0 g/t Au and decreases at the backend of the LoM, as feed over this period, is sourced from the MO stockpile. RoM feed grade is predominantly above the mined grade, apart from when insufficient ore is mined to sustain plant capacity, and feed is sourced from stockpile material.

Figure 16-17 shows the plant usage and ore feed to the processing plant by weathering type. Figure 16-17 summarises the annual Processing schedule.

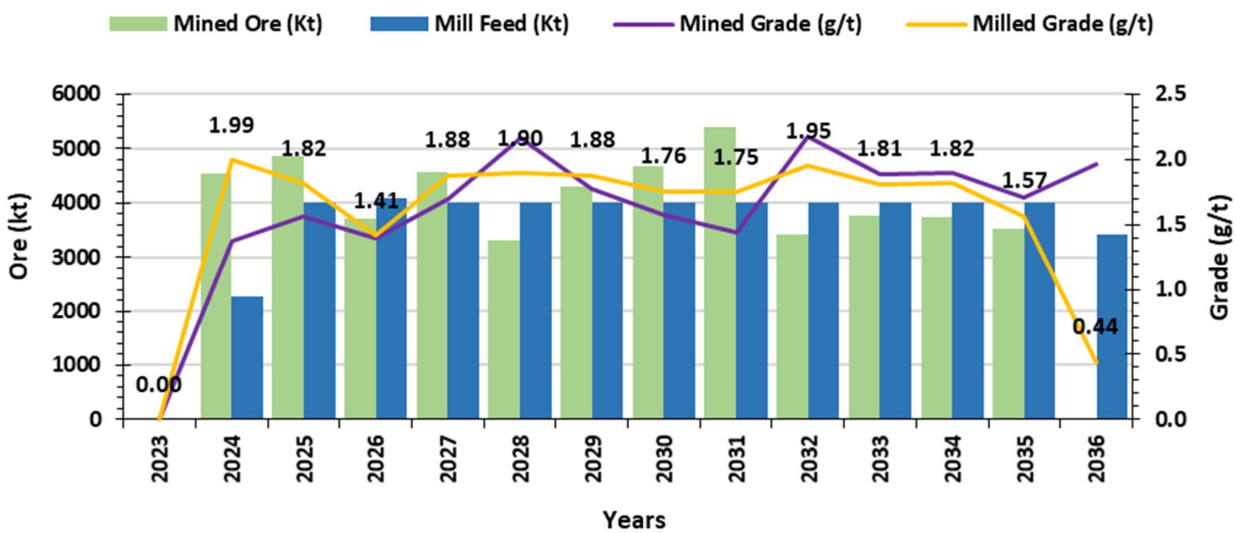


Figure 16-16: Ore Feed by Material Type

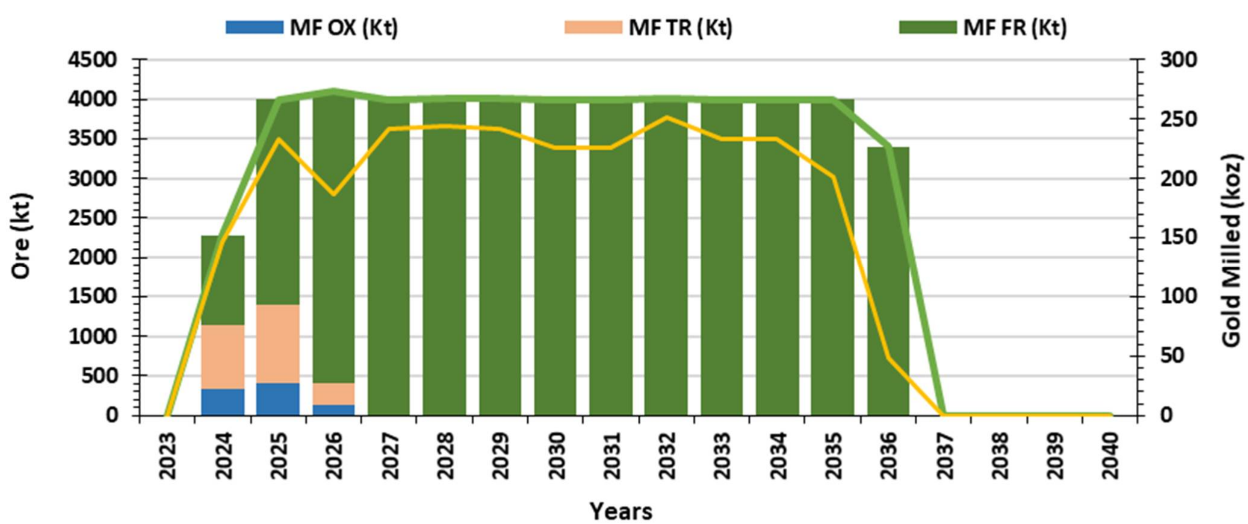


Figure 16-17: Ore Feed by Weathering Type and Gold Milled

16.15.4 Gold Product Schedule

Figure 16-18, following, illustrates the weighted average metallurgical recovery and gold produced by year. Production during the first three years is the most constrained, with 2026 not exceeding 200 koz. After 2026, production improves with higher grades becoming available as Pit A pushback 2, reaches a thicker portion of an HG lens. After that, gold production fluctuates between (210 and 240) koz/a as targeted, with the tail-end decreasing as MO becomes the primary feed material.

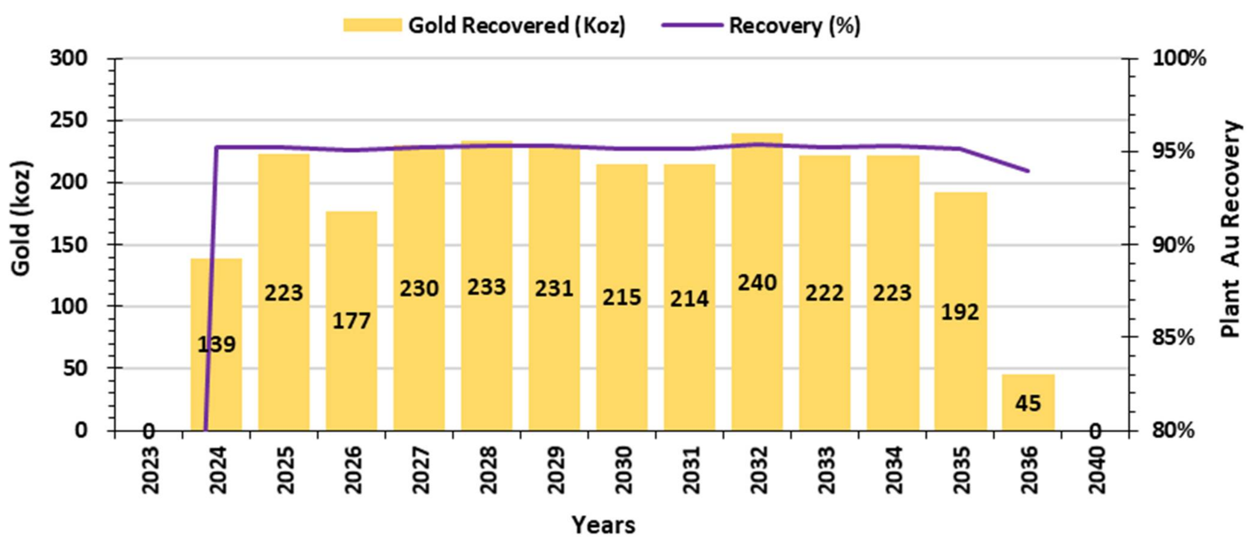


Figure 16-18: LoM Gold Production and Metallurgical Recovery by Year

Figure 16-18 notes: Gold production is reported by calendar year, with first gold being poured in Q2 2024. The reporting periods within the financial model was adjusted to first gold pour date, with year one running from Q2 2024 to Q2 2025.

Table 16-18: Annual Processing Schedule

Description	Units	Totals	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Ore Processed	(kt)	49 813	2279	4000	4098	4000	4011	4008	4000	4000	4011	4000	4000	4000	3405
Grade	(g/t)	1.69	1.99	1.82	1.41	1.88	1.90	1.88	1.76	1.75	1.95	1.81	1.82	1.57	0.44
	(kt)	867	342	400	125	0	0	0	0	0	0	0	0	0	0
Oxide	(%)	1.7	15	10	3	0	0	0	0	0	0	0	0	0	0
	(kt)	2079	798	995	279	0	0	8	0	0	0	0	0	0	0
Transitional	(%)	4.2	35	25	7	0	0	0	0	0	0	0	0	0	0
	(kt)	46 866	1140	2605	3694	4000	4011	4000	4000	4000	4011	4000	4000	4000	3405
Fresh	(%)	94.1	50	65	90	100	100	100	100	100	100	100	100	100	100
	(kt)	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Inferred	(%)	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Contained Gold	(koz)	2714	146	234	186	242	245	242	226	225	251	233	234	202	48
Gold Recovery	(%)	95.2	95	95	95	95	95	95	95	95	95	95	95	95	94
Gold Recovered	(koz)	2584	139	223	177	230	233	231	215	214	240	222	223	192	45

16.16 Mining Costs

Mining costs are presented in Section 21.

16.17 Data Verification

The approach to verifying, using, and presenting data from others in Section 16, is discussed in Section 12.

16.18 QP Comments on Section 16

The QP confirms that the reserve estimate and mining study has been undertaken to a DFS level of accuracy in accordance with the Canadian National Instrument 43-101, and adhering to the CIM Definition Standards guidelines (CIM, 2014). The QP notes that more work is required to bring the hydrogeology up to FS level but that it should not influence the reserve estimate if recommendations are followed.

16.19 Interpretations and Conclusions

Interpretations, conclusions, and risks pertaining to Section 16 are presented in Section 25 of this Report.

16.20 Recommendations

Recommendations/forward work programme activities pertaining to Section 16 are presented in Section 26 of this Report.

16.21 References

References cited in the preparation of Section 16, are presented in Section 27 of this Report.

17. RECOVERY METHODS

17.1 Process Design

17.1.1 Overview

Based on Mine Schedule 13k (SRK, 2022) and previous mine schedule iterations, the Lafigué Process Plant (the 'Plant') has been designed to process 4.0 Mt/a (dry basis (db)) of fresh ore, over a 13-year life of mine (LoM). The weighted average LoM gold feed grade to the plant is 1.69 g/t, with a mean monthly range of (0.42 to 6.75) g/t. The mine plan was developed to suit the Plant capacity/configuration and will produce a LoM weighted average of 208 koz/a of gold, with a LoM production range of (177 to 240) koz/a over the years of full production from year 2 to year 11[84]. Given low silver grades in the RoM ore, the gold doré produced is likely to contain in excess of 92% w/w gold. Metallurgical recoveries of gold are generally expected to exceed 96%.

At the plant front-end, a two-stage crushing/HPGR circuit was selected on the basis that the ore is essentially 95% unweathered rock, with only minor transitional and oxide components, and ore characteristics which indicate that the fresh ore will have high comminution energy requirements. The downstream circuit comprises a conventional ball milling and gravity/hybrid CIL treatment plant.

The Process Design Criteria (PDC) document (Lycopodium, 2022c; Lycopodium, 2022d) details the key engineering and metallurgical design criteria used to develop the plant design and CAPEX and OPEX cost estimates. The data presented in Section 13, summarises the key input parameters to said PDC.

The Process Flow Diagrams (PFDs), process design criteria (PDC) and mass and water balances, along with the plant services required, form the basis for process design and the equipment selected.

Value engineering assessments (VEA) were completed to assess the benefits of key processing flowsheet options for the Lafigué Project (the 'Project'). The selected options are reflected in the flowsheets and design criteria presented for the DFS.

17.1.2 Plant Design Philosophy

17.1.2.1 Mine Plan

The mine plan (SRK, 2022) was developed to align with the Plant nameplate throughput and constraints in terms of limiting the oxide/transition content of the feed blends in the early years of operation.

The combined oxide/transition component of the blend will typically be maintained below 30%, to minimise the impact of fine ores on the downstream ore processing circuit. This oxide/transitional material feed constraint will necessitate the stockpiling of this ore for later blending, rather than direct RoM tipping.

The selection of a two-stage closed circuit crushing flowsheet followed by HPGR crushing of the ball mill feed is not compatible with feed blends comprising high fractions of low competency ore or oxides with poor material handling characteristics.

The mine plan LoM diluted ore distribution fed to the Plant is 1.7% oxide, 4.1% transitional and 94.2% fresh ore. The LoM annual processing feed make-up and weighted average gold grade are illustrated in Figure 17-1.

[84] Based on a calendar production schedule, not on a yearly production schedule from first gold pour, in Q2 2024.

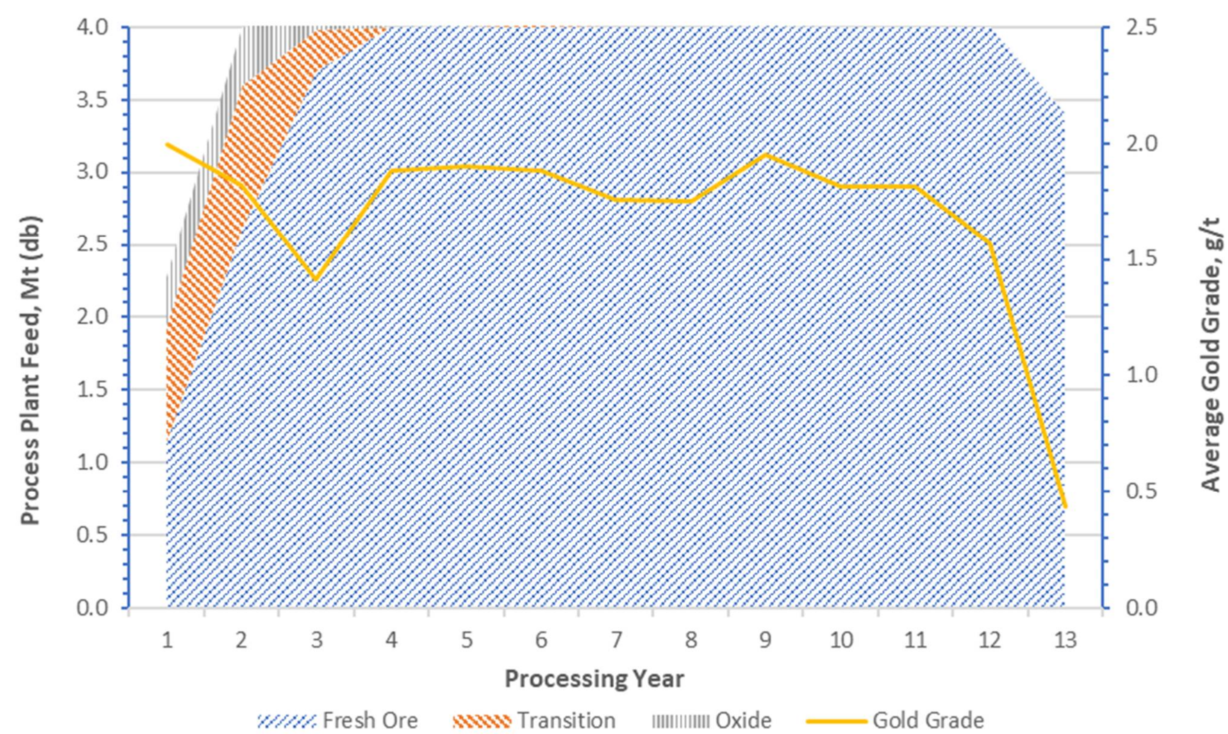


Figure 17-1: Mine Plan/RoM Feed by Year (SRK, 2022)

17.1.2.2 General Design Principles

The design of the plant is based on achieving the following:

- A fit for purpose plant with low operating costs and high metal recoveries.
- Optimum use of recycle water and, as a result, minimisation of the raw water make-up required.
- Reduced energy consumption where possible (low CO_2 footprint), without compromising process or project economic viability.
- Adherence to regulations and accepted best practise with respect to Occupational Health and Safety (OHS) and environmental standards and monitoring.
- Utilisation of appropriate and proven technologies and equipment.
- Minimisation of labour requirements and optimisation of process performance through appropriate levels of automation.
- Practical metallurgical accounting and gold reconciliation sampling and monitoring instrumentation provisions.

The process design selected for capital and operating cost estimation is based on standard unit operations, and equipment from reputable vendors that is well proven in the industry.

The Plant will process mainly fresh ore, with the early operation processing blended oxide/transition/fresh feeds. The combined oxide/transition component of the blend will typically be maintained below 30% to minimise the impact on the downstream ore processing equipment. The fresh ore is significantly more competent than the weathered oxides but has free flowing material handling characteristics and following grinding (P_{80} = 106 µm), the slurry has low viscosity and is readily dewatered.

The key project and ore specific design criteria that the plant design must meet are as follows:

- 4.0 Mt/a (db) of fresh ore.
- Mechanical overall plant availabilities of:
  - Closed circuit secondary crushing plant (70%).
  - Closed circuit high pressure grinding rolls (HPGR) circuit (86.7%).
  - Remainder of the downstream plant (91.3%).
- Intermediate crushed ore storage and provision of standby equipment for critical duties have been included, to ensure the overall availability and nameplate throughput can be met on a sustainable basis.

17.1.3 Selected Process Flowsheet

The selected treatment plant flowsheet incorporates the following unit process operations in accordance with the PDC and the plant design philosophy outlined in Section 17.1.2:

- Ore receipt at a RoM bin loaded by direct tip from haul trucks or front-end loader (FEL). Mine operations will stockpile oxide/transition and low-grade ores on the RoM pad to allow controlled blending of the plant feed.
- Fixed grizzly protection to prevent oversize blockages and vibrating grizzly screening to bypass fines ahead of jaw crushing.
- Primary jaw crushing to produce a coarse crushed product.
- Secondary cone crushing in closed circuit with a dry sizing screen to produce an intermediate crushed product.
- A live secondary crushed ore stockpile providing buffer storage of crushed ore with continuous reclaim to feed the HPGR circuit.
- HPGR operation in closed circuit with a wet sizing screen with undersize slurry reporting to the ball milling circuit via the mill discharge hopper and classification hydrocyclones.
- A ball mill in closed circuit with cyclones to produce a grind size of 80% passing (P_{80}) 106 micron (µm).
- Gravity concentration and recovery of coarse gold from the milling circuit, with treatment of the gravity concentrate by intensive cyanidation and electrowinning of the pregnant solution to recover gold doré.
- Trash screening to remove any wood trash or grit/oversize material from the cyclone overflow ahead of the carbon-in-leach (CIL) circuit.
- Pre-leach thickening of the trash screen underflow to dewater the leach feed to reduce reagent consumption and the leach and adsorption tankage volume required. Pre-leach thickening also recovers much of the essentially cyanide free water for recycle to the mill circuit, to minimise leaching of the gravity gold and OHS issues associated with cyanide in the mill.

- A leach tank ahead of the CIL tanks to maximise the gold solution grade feeding the adsorption tanks and to cater for pre-aeration of the slurry, should some ores consume the available dissolved oxygen provided by standard air sparging. The CIL circuit will continue leaching the gold in parallel with adsorption of the gold in solution onto the activated carbon.
- A split AARL elution circuit, electrowinning, and gold smelting operations to recover gold from the loaded carbon to produce doré.
- Thickening of the CIL tails slurry to maximise the tails solids concentration, minimise gold solution losses, and to recover process water and cyanide.
- Dilution of the tails thickener underflow with decant return/raw water in order to meet the target cyanide discharge level to the tailings storage facility (TSF).
- Tailings pumping to the TSF.
- Reagent mixing, storage, and distribution facilities.
- Provision of water treatment as required with storage and distribution of the various water services throughout the Plant.
- Generation of compressed air required and distribution through the circuit.

A process Block Flow Diagram (BFD) depicting the sequence of the unit operations for the selected process flowsheet is illustrated in Figure 17-2 following.

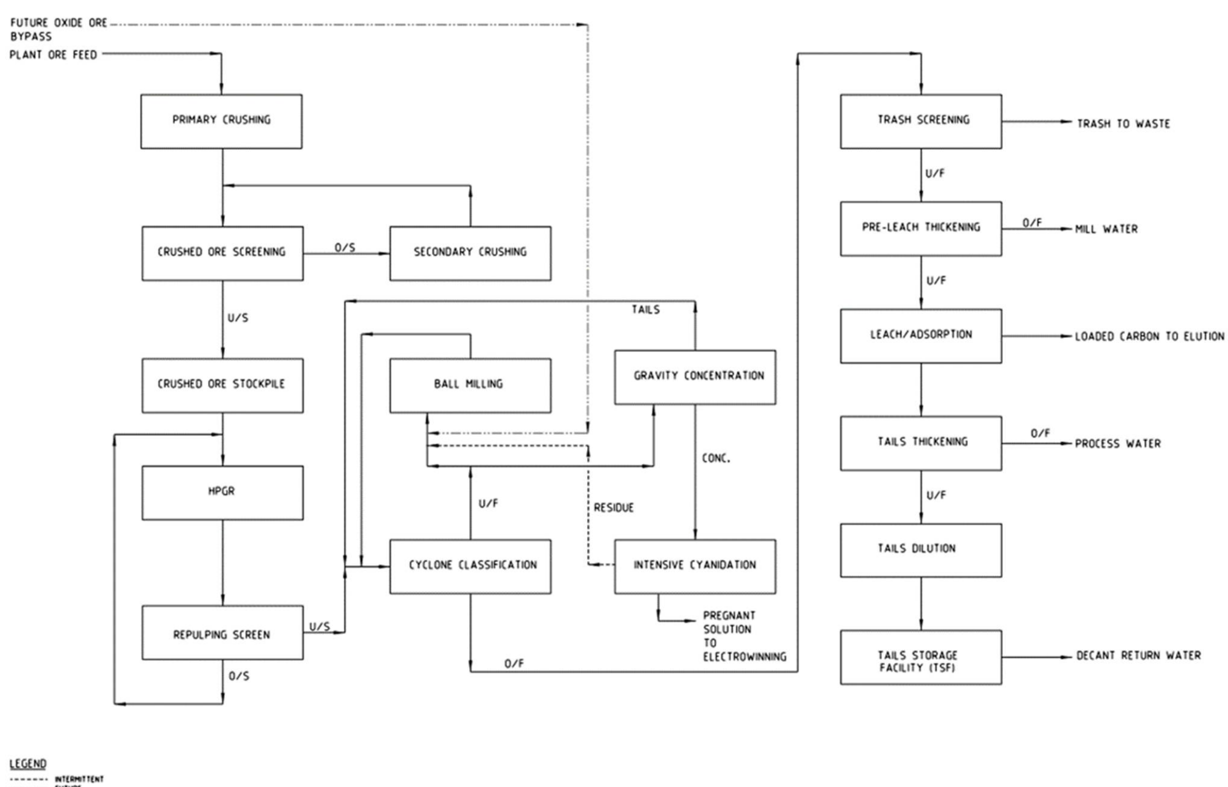


Figure 17-2: Overall Process Flow Block Diagram for the Plant

17.1.4 Plant Design Basis

The plant design basis and key issues considered in the process and equipment selection are outlined in this section. The Plant design is based on a nominal capacity of 4.0 Mt/a (db) of fresh ore feed. The plant will process the relatively low tonnage of oxide/transition ore overlying the fresh rock by blending this with the fresh ore up to a 30% fraction of the new plant feed ore. The plant feed schedule indicates that the oxide and transition ore make up 1.5% (oxide) and 4.1% (transition) of the life of mine (LoM) plant feed. The majority of the oxide and transition ore will be processed in the first two to three years of operation. Much of the original oxide resource has already been depleted by artisanal mining activities.

17.1.4.1 Comminution and Metallurgical Testwork Outcomes

The key comminution and metallurgical testwork outcomes influencing the process flowsheet selection and design criteria are summarised below.

- The comminution testwork results indicated the fresh ore is very competent with a consistently high breakage energy requirement for the coarse particles. This resulted in high specific energies for SAG milling and motivated the selection of the HPGR flowsheet, as a more energy efficient comminution alternative. Low oxide fractions in the resource along with more moderate ball milling work indices and low to moderate abrasion indices further supported the multistage crushing option.
- Gravity testwork and mineralogical investigations indicated that much of the gold occurs as coarse, free grains. Recovery of the gravity gold for separate intensive cyanidation is considered essential, since leach kinetics can be slow, and overall gold extraction reduced if the gravity recovery stage is bypassed. The high gravity gold content will result in a degree of gold assay variability throughout the mine and Plant.
- Good agreement between preliminary BLEG assay results and gold fire assays indicated that the ore is free milling with high gold extractions. Subsequent multi-element head assays indicated that there are few deleterious elements for gold leaching or tailings management, with low levels of base metals and arsenic.
- Grind optimisation testing indicated that the ore is relatively insensitive to grind. A P_{80} grind size of 106 µm was selected for design, following an economic evaluation of optimum grind size.
- Leach optimisation testing indicated that high gold extractions were achieved with air only sparging (no high purity oxygen required), relatively low cyanide dosing and high fresh ore slurry densities (up to 55% solids w/w).
- Variability leach testing indicated that although gold extractions were generally very high (>95%) approximately 10% of the samples tested displayed slightly lower extractions. Increased cyanide addition (maintaining a higher free cyanide excess concentration since consumption remained low) and extra leach residence time (36 h and 200 ppm free/excess NaCN) improved extractions to expected levels. No definitive explanation for this behaviour was identified, but slower leach kinetics were observed in a number of samples (this was frequently masked by high gravity gold extraction). The mineralogical investigations noted occasional associations of gold grains with pyrrhotite and silver tellurides. The presence of these elements is known to slow gold leaching. These effects are minor for the Lafigué ores, and the increased leach residence time and higher free cyanide should be sufficient to ensure high gold extractions. As a further mitigation, provision for lead nitrate (100 g/t) addition to the leach will be made.

- Slurry rheology testing indicated that the fresh ores display low viscosities up to high operating densities with fast settling rates. Results for the oxide ores indicated that although one sample tested contained viscous clay, the balance presented no material handling issues. Oxide/transition ore blends with the fresh feed will need to be managed to avoid excess fines presentation to the HPGR, but low oxide blends will also have benefits for downstream operation since the oxides may be slower settling with a tendency to form more viscous slurries.
- Cyanide and lime consumption will be very low, motivating the use of hydrated rather than quick lime. Cyanide usage will be further reduced by recovery and re-use of much of the excess free cyanide in the CIL tails.

17.1.4.2 RoM Pad and Ore Delivery (by Mining Contractor)

The RoM pad will be used to build buffer stockpiles between the mine and the plant to ensure there is always feed material available and to facilitate the planned blending of the fresh and weathered ores. The RoM stockpiles will further allow blending of feed stocks to ensure more consistent feed ore competency, or gold grade to the Plant to best suit processing requirements.

The RoM bin will be designed to accommodate direct tipping from the haul trucks (CAT 789 with a 177-t payload) and RoM stockpile reclaim by FEL. A fixed rock breaker adjacent to the RoM bin fixed grizzly will be used to clear the grizzly and break any retained oversize rocks.

17.1.4.3 Comminution Circuit Selection

Following a review of various comminution circuit alternatives, Endeavour selected an HPGR/ball mill circuit on the basis that this is the most energy efficient option for the very competent fresh ore. The higher energy efficiency HPGR technology also resulted in the lowest life cycle cost of the options considered. Comminution circuit testwork including HPGR testing was completed as part of the testwork programmes, as discussed in Section 13. The comminution data was provided to Orway Mineral Consultants (OMC) for comminution circuit modelling and equipment sizing.

The comminution data was also benchmarked against other ores where HPGR testwork results were available to support the recirculating load estimates and modelled product size distributions.

The OMC modelling was completed using their typical design point of the 85th percentile ore competency results with a nominated target P_{80} grind size of 106 µm selected, based on the optimum grind metallurgical testwork and economic evaluation.

The comminution circuit design basis and selected comminution circuit are described in the OMC Report 7441.45-02, 'Lafigué 4 Mtpa Circuit Update', October 2021. The comminution circuit design was based on a fresh ore feed, given that oxide and transitional ore is such a low proportion of the total feed over the life of mine. The plan is to blend in small amounts of oxide/transitional material which can be accommodated in the standard circuit design.

The key design criteria used in modelling of the comminution circuit and the key results for the selected circuit are summarised Table 17-1 and Table 17-2 following.

Table 17-1: Comminution Circuit Design Parameters

Parameters		Units	Fresh Ore
Ore Design Parameters			
• CWi (Derived from SMC testing)	85th %ile	kWh/t	28.4
• RWi	85th %ile	kWh/t	19.5
• BWi	85th %ile	kWh/t	16.9
• Ai	85th %ile/Average	g	0.251/0.163
• Mih	85th %ile	kWh/t	20.0
• Axb	15th %ile		26.0
• SG	Average	t/m^3	2.8
• HPGR Specific Pressure*		N/mm^2	3.5
• HPGR Specific Energy*	Per Pass	kWh/t	1.40
• HPGR Specific Throughput (m dot)*		ts/hm^3	300
• HPGR Product P_{80}		mm	2.1
Comminution Circuit Design			
• Plant Throughput		Mt/a (db)	4.0
• Crushing Feed F_{100}		mm	800
• Primary & Secondary Crushing	Availability	%	70%
	Throughput	dry t/h	652
• HPGR (Tertiary) Crushing	Availability	%	86.7%
	Throughput	dry t/h	527
• Grinding Circuit	Availability	%	91.3%
	Throughput	dry t/h	500
• Milling Feed F_{80}		mm	2.1
• Milling Product P_{80}		µm	106

Table 17-1 notes:

- *Testwork scaled to align with operating industry norms
- Low levels of oxide/transition ore were not considered when selecting the comminution equipment.

Table 17-2: Summary of Selected Comminution Equipment

Equipment	Unit	HPGR Crush/Ball Milling
Primary Crushing		
• Type		Jaw Crusher
• Model		Metso C160 or Equiv.
• Closed Side Setting	mm	150
Secondary Crushing		
• Type		Cone Crushers
• Model		2 x Metso HP6 or Equiv.
• Closed Side Setting	mm	35
• Machine Load	%	79
Crushing Screen		
• Type		Double Deck Dry Multi-slope
• Size (Width x Length)	m x m	3.0 x 7.3
• Top Deck Aperture	mm	65
• Bottom Deck Aperture	mm	38
• Crushing Product Size P_{80}	mm	25
HPGR		
• Size (Diameter x Width)	m x m	1.76 x 1.38
• Net Throughput	t/h	527
• Total Throughput	t/h	1048
• Speed	m/s	1.38
• Power - Duty	kW	2700
• Power - Installed	kW	2 x 1350
Milling Screen		
• Type		Double Deck Wet Multi-slope
• Size (width x length)	m x m	4.3 x 8.5
• Top Deck Aperture	mm	8
• Bottom Deck Aperture	mm	4
• Milling Screen Undersize P80	mm	2.1
Ball Mill		
• Mill Diameter x length (EGL)	m x m	6.40 x 10.70
• Discharge Arrangement		Overflow
• Mill Speed Range	% Nc	60 to 80
• Mill Speed	% Nc	75
• Top Ball Size	mm	60
• Ball Charge - Duty	% Vol	27
• Pinion Power - Duty	kW	6364
• Installed Power	kW	7700

The average 'unit' power and consumables requirements for the comminution circuit are summarised in Table 17-3 following and are used in the estimation of the processing operating cost, as detailed in Section 21. Annual grinding media consumption and power usage is summarised in Sections 17.1.6 and 17.3.3.2, respectively.

Table 17-3: Comminution Consumables, 4 Mt/a (db)

Equipment/Parameter	Unit	Fresh
Abrasion Index (average)		0.163
Primary Crusher		
• Liner Consumption - Fixed	h	2500
• Liner Consumption - Swing	h	3333
• Gross Power Consumption	kWh/t	0.2
Secondary Crusher		
• Liner Consumption - Bowl	h/unit	1677
• Liner Consumption - Mantle	h/unit	1677
• Gross Power Consumption	kWh/t	0.9
HPGR		
• Liner Consumption	hours/unit	8000
• Gross Power Consumption	kWh/t	2.8
Ball Mill		
• Media Consumption	kg/t milled	0.581
• Liner Consumption	kg/t milled	0.078
• Gross Power Consumption	kWh/t milled	12.6

17.1.4.4 Circuit Availabilities

Equipment sizing for the open-circuit primary crushing and closed-circuit secondary crushing and screening circuits, is based on an overall availability of 70% when processing fresh ore. This was driven by the lower availability of the secondary crushers, due to the expected higher relining/maintenance requirements.

The crushing circuit will be decoupled from the downstream plant by the crushed ore stockpile, which provides surge capacity (20 hours/10 000 t live capacity) between the lower availability crushing circuit and the higher availability downstream plant.

The sizing of the closed-circuit HPGR crushing circuit, is based on an overall availability of 86.7% on fresh ore. The HPGR circuit is likely to achieve higher operating availabilities, similar to those for the milling circuit, however the HPGR and mill downtime frequencies and durations are generally not aligned. The HPGR will require more frequent, shorter downtimes (typically two to four hours bi-weekly) for routine inspection/preventative maintenance, with one major downtime event to change the rolls at 12-to-18-month intervals. Unforeseen maintenance downtime will also occur at random intervals. The ball milling circuit will typically require less frequent, but longer maintenance downtime events. HPGR operation will be tied to the mill running (unless product stockpiling is occurring). Therefore, the overall HPGR operating availability will be lower than that of the milling circuit. The slightly higher HPGR circuit operating throughput will allow the build-up of sufficient surge capacity (in live storage and dead stockpiles) to sustain the milling circuit feed when the HPGR is off-line. Planned downtime events for the milling and HPGR circuits will be aligned when possible.

The HPGR will be decoupled from the milling circuit by the HPGR product surge bin which will provide greater than two hours surge capacity between the HPGR and the downstream plant. An extended HPGR product stockpile will provide up to 12 hours of emergency feed capacity, which can be extended to 72 hours for planned longer shutdowns such as the rolls change-out.

The milling circuit and downstream plant are sized on 91.3% availability (8000 operating hours at 500 t/h (db)) when treating fresh ore.

17.1.4.5 Primary Crushing, Secondary Crushing and Dry Screening

Closed circuit secondary crushing will be required to ensure that the maximum crushed product particle size is suitable for feeding the HPGR. HPGRs are sensitive to feed size, and it is important that the top feed size is significantly less than the HPGR operating gap to minimise stud wear and potential for stud breakage. A consistently finer feed size to the HPGR will improve the HPGR tyre wear life and comminution efficiency.

Modelling of the closed-circuit crushing circuit based on primary crushed product distribution and ore hardness was used to determine the likely secondary feed recirculating load for design.

Metal tramp detection and removal will be required to protect the secondary crushers and HPGR downstream. At least two stages of metal removal are required to improve the probability of recovering the tramp metal, particularly in the early operations phase where residual implements from artisanal mining may be mixed with the ore.

17.1.4.6 HPGR Circuit

The HPGR product size distribution and recirculating load were determined by OMC based on the closed circuit testwork, benchmarked operating parameters and repulping/wet screening efficiencies assuming a 4 mm closing screen size.

The HPGR product will contain considerable oversize due to the pressure profile across the roll width, with little crushing occurring near the roll edges. Wet screening is necessary to achieve efficient screening down to fine sizes. A finer closing size increases the work done by the HPGR, making the overall comminution process more energy efficient by reducing the ball milling specific energy requirement. The screen aperture size selected is a practical trade-off given increasing equipment size and moisture carryover (recycled to the HPGR feed) with decreasing screen open area. Site trials may determine that operation at a finer screening size is possible, allowing further reduction in overall circuit power draw.

The optimum HPGR feed should contain some graded fines to increase close packing in the ore bed and some moisture to improve binding and integrity of the autogenous wear protection layer. Too many fines or too much moisture can wash out the autogenous layer or result in relative slippage between the material bed and the roll increasing stud and tyre surface wear.

The HPGR product will be mixed with water in the repulping screen feed box to maximise the de-agglomeration of the HPGR flake product ahead of the screen. Low cyanide mill water will be used in this area to minimise OHS issues with potential hydrogen cyanide (HCN) release with the large screen surface area and spray mist. Wet screening will be conducted at less than 50% feed solids w/w, to ensure high screening efficiency and minimum undersize recycle.

17.1.4.7 Milling and Classification

The milling screen undersize will gravitate directly to the mill discharge hopper. The ball mill will be reverse fed via the cyclone underflow. This will remove final product size material generated by the HPGR directly, effectively reducing the new feed rate to the mill. This HPGR advantage (in terms of finished product generation relative to tertiary crushing) has been considered when sizing the ball mill.

The ball mill will be equipped with a variable speed drive and will typically be operated between 60% and 78% of critical speed. The variable speed drive will allow management of the energy input when treating blended fresh ore feeds with less competent oxide/transition fractions reducing the specific energy required. A lower ball charge will be utilised for start-up operations but managing ball charge level is too inflexible to be used for optimising control to match the power drawn to the degree of energy required for size reduction.

Alkalinity from the decant return water will build up in the mill water circuit over time, but should the ore tend to lower the pH in the mill water, provision will be made for adding hydrated lime slurry to the ball mill feed.

The ball mill will discharge via a trommel screen into the mill discharge/cyclone feed hopper for normal closed-circuit operation. The flow to the gravity concentrators will be managed by dedicating a number of cyclones to feeding the gravity circuit. For a given inlet pressure and density range, the mass flow to underflow will be predictable and relatively constant, ensuring a correct and known feed rate to each concentrator based on the number of feed cyclones selected. Gravity tails will be returned to the mill discharge hopper to avoid water balance issues in the mill.

The balance of the cyclones will classify the re-circulating mill load with the underflow reporting to the ball mill. Vendor modelling of the cyclones suggests that a smaller diameter cyclone (500 mm, OMC recommended 650 mm) will be best suited to the target cut-point and lower density feed conditions. The relatively low cyclone feed density results from the extra water addition from the repulping screen and gravity circuits. The greater number of cyclones at the lower diameter has the advantages of increased flexibility with the gravity split and reduced magnitude of a step change if a cyclone is brought on or taken off-line.

17.1.4.8 Pre-Leach Thickening

The dilute operating conditions in the cyclone circuit result in highly efficient classification, with high cyclone underflow densities and minimum fines recycle. This does also mandate pre-leach thickening to increase leach feed density. This is used to benefit the processing outcomes since it facilitates recovery of an essentially cyanide-free water stream to avoid leaching of the gravity gold in the milling circuit and having dilute gold values in this large circulating water system.

The relatively fast settling rates for the fresh and blended ores and high underflow densities produced also allow over-thickening to minimise raw water make-up to the mill water and provide the opportunity to reuse the recovered cyanide solution from the tails stream as leach feed dilution.

17.1.4.9 Gravity Concentration

The very high gravity recoverable gold fraction from the Lafigué ores require that an efficient gravity gold recovery circuit will be operating at all times. This has the advantages that:

- The feed grade to the leaching circuit will be smoothed by removing the high-grade spikes from the mill feed stream. This will assist with metallurgical accounting as the high-grade spikes can completely misrepresent the gold present when they occur and equally fail to record significant gold if they are missed. Gravity gold will be separately accounted for by independently electrowinning the intensive cyanidation solution.
- The bulk of the gold will be concentrated in a small mass for intensive cyanidation with inherently higher leach recoveries and little opportunity for potential interference from deleterious minerals such as pyrrhotite and silver tellurides, which have been identified as being associated with some of the gold mineralisation.
- Gold leach kinetics will be significantly faster without the coarse gravity gold content in the gravity tails leach, making for improved CIL solution and carbon profiles and reduced gold lock-up.
- It is likely that the deleterious elements that are associated with the gold will mostly report to the gravity concentrate such that the highly oxidising intensive cyanidation conditions will reduce their downstream impact when they are returned in the intensive cyanidation tails.

If the gravity circuit is bypassed, leaching reagent dosing rates would need to be significantly increased to ensure residues are minimised and faster leach kinetics achieved. This is despite evidence that reagent consumption is little affected by the coarser gold or deleterious elements/minerals, but the presence of increased free cyanide and lead nitrate in solution is clearly beneficial.

17.1.4.10 Leach Circuit

The metallurgical testwork indicated the following:

- The Lafigué ores typically have high gravity gold content. Thus, gravity gold recovery with intensive cyanidation of the concentrate is very beneficial for overall gold recovery, with significantly faster leach kinetics for the gravity tail compared with whole of ore leaching.
- The initial leach kinetics for the majority of the samples tested were rapid following gravity gold recovery, with between (95 and 100)% of total gold extraction being achieved in the first twelve hours of leaching. A few gravity tails samples displayed slower leach kinetics, however, and benefitted from additional leaching time through to 36 hours. This was subsequently addressed by increasing the cyanide concentration and dosing with lead nitrate. The combined effect tended to normalise the leach kinetics for these samples. Almost all test samples achieved very high overall gold extractions with the enhanced leach conditions. Provision for the longer residence time is allowed in the design to ensure that maximum gold extraction is achieved for all ores.
- Deleterious elements were rarely present in the Lafigué ores, and these appear to have minimal effect on the gold leaching with low cyanide consumption and oxygen uptake rates.
- High leach slurry density did not reduce gold extraction. A leach slurry density of 55% w/w solids was selected.
- Cyanide and lime consumption was typically very low for the fresh ores. The oxide/transition ores exhibited slightly higher lime consumption to maintain the pH.
- Very low oxygen requirements during leaching were measured with no benefit noted for oxygen sparging instead of air. Air sparging was selected for the flowsheet.

Endeavour agreed that the simplicity and cost effectiveness of a CIL circuit outweighed the metallurgical benefits of the originally nominated separate leach/carbon-in-pulp (CIP) circuits. The low gravity tails head grades and fast leach kinetics (for all ores having addressed the slow leaching instances) were factors in the decision to adopt a hybrid CIL circuit for the project.

The leach/adsorption circuit was based on a leach tank followed by a six stage CIL circuit providing 36-hour residence time when treating 4.0 Mt/a (db) (500 t/h (db)) fresh ore at 55% solids w/w.

17.1.4.11 Carbon Elution and Gold Recovery

A 12-t capacity split AARL elution circuit with nominally seven strips per week will be required based on the gravity tails CIL gold and probable silver loadings. This capacity allows for faster carbon movement (more frequent stripping) to catch up after periods of elution circuit downtime or if the gravity circuit is bypassed resulting in higher gold loadings.

Electrowinning requirements were calculated based on the modelled loaded carbon grades and the expected gravity gold pregnant solution grade. The goldroom will contain two electrowinning cells for the elution, with a third electrowinning cell dedicated to the gravity circuit.

17.1.4.12 Plant Tailings Treatment

The clean Lafigué ores have low propensity for weak acid dissociable cyanide (CN_{WAD}) formation such that most of the residual cyanide in the tails stream is CN_{Free}. A cyanide recovery solution, rather than destruction, was selected to minimise the cyanide reporting to tails. Tails thickening will be included in the flowsheet for water recovery given the scarcity of regional water sources and reliance on harvesting and storage of rainfall catchment during the wet season. The tails thickener will serve to recover the associated cyanide and the separation of the mill (low to no cyanide), and process water (cyanide containing) circuits will allow effective re-use of the residual cyanide in the leach feed dilution, CIL screen sprays and service points. Soluble gold loss will also be recycled in this water stream, adding further to process efficiency.

Although the quantity of cyanide in the tails stream is significantly reduced, the concentration will still exceed the International Cyanide Management Code (ICMC) guideline of 50 ppm CN_{WAD} for discharge from the plant. The natural breakdown of cyanide in the TSF supernatant water (and dilution by rainfall into the TSF) will allow the use of decant return water for dilution of the tail. The more dilute tails stream will settle faster and have higher water release rates, returning much of the decant water to the supernatant pond for further recycle as tails dilution water.

17.1.4.13 Raw Water

The majority of raw water for the project will be sourced from rainfall runoff to the water harvest dam (WHD). Additional make-up water will be available from groundwater bores, pit dewatering and rainfall within the storage catchments. Raw water will be supplied to the plant, mine and camps and will be the feed stream for the filtered water treatment plant.

Recycling of water within the Plant will be maximised in order to minimise raw water usage. For effective recycling it will be important to maintain awareness of the quality differences between the water types to ensure that the cleaner water is not degraded, but also that the impacted water use in the plant is maximised. The mill and process water circuits will be designed and implemented to ensure this general operating practise is readily maintained.

17.1.5 Key Design Criteria Parameters

The key process design criteria listed in Table 17-4 form the basis of the detailed process design criteria and mechanical equipment list.

Table 17-4: Summary of Key Process Design Criteria Parameters

Criteria	Unit	Fresh	Source/Basis
Plant Throughput	Mt/a (db)	4.0	EDV
Life of Mine (LoM) Ore Blend		94.2% fresh + 4.1% transition + 1.7% oxide[1]	SRK
Design Gold Head Grade	g Au/t	2.5	Agreed
Gravity Gold Recovery	%	70	Testwork
Design Overall Gold Recovery	%	99	Testwork
Primary Crushing Plant Utilisation	%	70	Lyco/OMC
Secondary Crushing Plant Utilisation	%	70	Lyco/OMC
HPGR Crushing Plant Utilisation	%	86.7	Lyco/OMC
Milling/Leaching Plant Utilisation	%	91.3	Lyco/OMC
RoM Ore Top Size	mm	800	OMC
Ore SG		2.80	Test/OMC
Comminution Circuit		Prim/Sec/HPGR Crush/Reverse fed Ball Mill	Endeavour/OMC
Crush Size, P_{80}	mm	2.1	OMC
Milling Circuit Top Size, P_{100}	mm	4	OMC
Target Grind Size, P_{80}	µm	106	Testwork/EDV
Cyclone Overflow Density	% solids w/w	19.0	Vendor
Pre-leach Thickener Solids Loading	$t.m^{-2}.h^{-1}$	0.75	Testwork/Lyco
Pre-leach Thickener Flocculant	g/t	30	Testwork
Pre-leach Thickener Underflow Density	% solids w/w	63.5	Testwork/Lyco
Leach Feed Slurry Density	% w/w	55	Testwork/Lyco
Leach Residence Time	hrs	36	Testwork/Lyco
Number of CIL Tanks		1 Leach/6 CIL	Lyco/EDV
Average Cyanide Consumption[6]	kg/t	0.17	Testwork
Average Hydrated Lime Consumption[7]	kg/t	0.29	Testwork
CIL Carbon Loading	g Au+Ag/t	3600	Lyco
Elution Circuit Type		Split AARL	EDV
Elution Circuit Capacity	t/strip	12	Lyco
Elution HCL	kg/strip	1860	Lyco
Elution NaOH	kg/strip	641	Lyco

Table 17-4: Summary of Key Process Design Criteria Parameters

Criteria	Unit	Fresh	Source/Basis
Strip Solution Heater Diesel	L/strip	2373	Lyco
Frequency of Elution	strips/week	7	Lyco
Tailings Thickener Solids Loading	$t.m^{-2}.h^{-1}$	0.75	Testwork/Lyco
Tails Thickener Flocculant	g/t	30	Testwork
Tails Thickener Underflow Density	% solids w/w	63.5	Testwork/Lyco

Table 17-4 notes:

- Plant feed mainly fresh ore, but with some blended feed up to 20% oxide or 30% oxide transitional.
- 'Testwork' refers to metallurgical testwork conducted.
- 'EDV' refers to advice/agreement from Endeavour Mining.
- 'Lyco' refers to Lycopodium first principles calculation/experience or generally accepted practise.
- 'OMC' refers to advice from Orway Mineral Consultants.
- Cyanide consumption makes allowance for 100 ppm residual NaCN in the CIL tail solution, (assume 50% recovery).
- Lime consumption based on 60% CaO.
- 'SRK' refers to advice from SRK Consulting.

17.1.6 Annual Reagent/Consumable Requirements

Plant: raw-water makeup, diesel, and power requirements are discussed in Sections 17.3.1.1, 17.3.2 and 17.3.3.2, respectively, whilst Plant consumption of key consumables/reagents per annum are summarised in Table 17-5 following.

Table 17-5: Plant Annual Consumption Rates for Consumables/Reagents (Lycopodium, 2022d)

Consumable/Reagent	Fresh (t/a)	LoM Average (t/a)
Grinding Media	2604	2526
Cyanide	844	853
Carbon	161	161
Lime (Ca(OH)2)	814	1041
Hydrochloric acid	677	677
Sodium Hydroxide	238	238
Flocculant	240	245

Table 17-5 notes:

- Fresh ore accounts for 94.2% of the ore processed over the LoM (Figure 17-1).
- Annual liner consumption/changes for comminution equipment is not shown here. Unit consumption rates are presented in Table 17-3.
- Annual consumption figures for consumable/reagents used in the gold room are not shown here (minor quantities).
- Lead nitrate is not shown on the basis that it is added on an ad hoc basis.

17.2 Plant Description

The Plant will mainly process fresh ore feed although a blended feed with up to 30% oxide/transitional ore will make up the plant feed for the first two to three years of operation, as per the mine plan (SRK, 2022). This is regarded as the maximum allowable finer feed content in the ore blend with the HPGR circuit requiring competent ore to maintain the HPGR gap setting between the rolls.

The Plant layout is illustrated in Section 18.

17.2.1 RoM Pad

Haul trucks operating from the open pit will deliver run-of-mine (RoM) ore to the RoM pad where it will be direct tipped to the RoM bin or dumped in blending 'finger' stockpiles arranged by ore gold grade and lithology. A front-end loader (FEL) will be used to reclaim and tram ore from the various stockpiles to the RoM bin.

Ore will be blended under the guidance of mine geologists and process personnel to maintain a relatively constant feed grade to the Plant and less than 20% oxide or 30% oxide/transitional blend in the feed.

17.2.2 Crushing Circuit

17.2.2.1 Primary Crushing

A horizontal static grizzly will be fitted to the RoM bin to prevent blockages in the chutes and to protect the downstream equipment from oversize material. A fixed rock breaker will be utilised to clear the grizzly and break oversize rocks on the grizzly or RoM pad.

Water sprays on the RoM bin will provide dust suppression during ore tipping.

RoM ore will be drawn from the RoM bin by a variable speed apron feeder and discharge onto a vibrating grizzly feeder. The grizzly feeder oversize will report to the primary jaw crusher. A belt feeder will be used to collect dribble from the apron feeder and discharge this to the crusher discharge conveyor CV1.

The grizzly undersize and primary crushed product will discharge onto CV1. Secondary crushed ore will also discharge onto CV1. A weightometer will indicate the combined crushed ore tonnage including the secondary recirculating load. Primary crushed ore tonnage will be inferred and totalised as the difference between the secondary crusher feed tonnes (CV5 weightometer) and the total tonnage recorded on CV1.

A self-cleaning head pulley magnet will be mounted above the CV1 discharge, to remove magnetic tramp for secondary crusher and HPGR protection. CV1 will discharge to the crushing screen feed conveyor (CV2) via a transfer station.

17.2.2.2 Screening and Secondary Crushing

The combined crusher products will be transferred via CV2 to the crushing screen. The crushing screen feed chute will spread the feed with a number of falls to ensure full width screen feeding. The screen will be a double deck unit, mainly to provide a protective upper deck given the large top size of the primary crusher product (P_{100} = 385 mm). The lower deck will do the sizing to ensure that the HPGR feed size is maintained below 38 mm.

Crushing screen oversize will report to the crushing screen oversize conveyor (CV4) and will be transferred to the secondary crusher feed conveyor (CV5). A metal detector located above CV4 will detect any remaining tramp metal in the secondary crusher feed and will bypass tramp metal via a diverter chute at the head pulley to a tamp bunker. A weightometer on CV5 will indicate the secondary crushing circuit recirculating tonnage.

CV5 will discharge into the secondary crusher feed bins. The bins will be centrally fed to achieve an even ore split to the two crushers based on the rill angle into each bin. If one crusher is down, that side of the bin will fill to the point where all feed is directed to the operating crusher. To feed the secondary cone crushers at a controlled rate, ore will be drawn from the bifurcated bin using the secondary crusher feeders. Secondary crushed ore will report to CV1 (along with primary crushed ore) feeding the crushing screen for undersize removal.

Crushing screen undersize, with a top size of 38 mm, will report to the stockpile feed conveyor (CV3) discharging onto the crushed ore stockpile. A weightometer on CV3 will indicate the crushed ore stockpile feed tonnage for crushing circuit throughput accounting.

A semi portal crane will service both the primary and secondary crushing areas to expedite relining and other maintenance activities in this area.

17.2.3 Crushed Ore Stockpile Reclaim, HPGR Crushing and Wet Screening

17.2.3.1 Stockpile Reclaim

Crushed ore will be withdrawn from the stockpile at a controlled rate by two variable speed apron feeders onto the HPGR feed conveyor (CV7). Each feeder will have the full plant feed capacity, so they can run independently if required, but normally they will run together to maximise the live capacity of the stockpile. A degree of size segregation in the stockpile feed can be expected. This segregation can be smoothed out by managing the relative feeder speeds, but the HPGR is less sensitive to feed size variation than a SAG mill.

The HPGR closed circuit screen (milling screen) oversize will be stockpiled adjacent to the crushed ore stockpile and reclaimed via a vibrating feeder onto CV7. Under normal operation it is not intended to stockpile any screen oversize; the stockpiling facility is provided to cater for HPGR product reclaim during periods of HPGR downtime. Reclaim tunnel ventilation and dust collection will be provided.

17.2.3.2 HPGR Feed Bin

CV7 will discharge to the HPGR feed bin. A metal detector located above the CV7 head pulley will detect any tramp metal in the HPGR feed and will activate a diverter gate to further protect the HPGR from any metal ingress. The bypass ore will be returned to the crushed ore stockpile following metal removal. It will be a priority to cross mix the bin feed to minimise size segregation in the HPGR feed bin. This bin is fed at right angles to the withdrawal direction and segregation across the width of the HPGR must be minimised to avoid causing skewing of the rolls.

17.2.3.3 HPGR

HPGR feed will be drawn directly from a feed bin and chute above the HPGR. The priority will be to maintain a level in the HPGR feed chute to minimise feed variation and associated tyre wear. A number of different HPGRs have been offered for this duty with varying width to diameter ratios, tyre wear and roll skew solutions as well as other features. Equivalence was based on specific throughput for each offer. The HPGR will be provided with variable speed drives on the rolls and variable pressing force to optimise size reduction at the nominated throughput rate. Installed power will cater for spikes in operating power draws, but normal operating power will only be 1.4 kWh/t.

The HPGR operates by applying a high pressing force to the material drawn into the operating gap between the rolls. Roll surface wear under these extreme abrasive conditions will be addressed by fitting tungsten carbide studs to the roll surface that allow an autogenous wear layer to build up between the studs, significantly increasing roll tyre wear life and reducing the reline frequency.

The proposed flowsheet addresses the downtime requirements of the HPGR with surge capacity between the HPGR and mill and the facility to create and reclaim extended HPGR product stockpiles as required to maintain mill operation.

17.2.3.4 HPGR Product Surge Bin and Extended Stockpile

The ore feed will be crushed in the HPGR and will discharge to the HPGR product conveyor (CV8) feeding the HPGR product surge bin. The surge bin will have a rill outlet to allow drive in access to clear the bin and also reclaim from the bin to build extended product stockpiles as required to ensure mill feed availability during planned, and possible unplanned, HPGR downtime periods.

HPGR product will be drawn from the surge bin via the HPGR product reclaim feeders one to three onto the milling screen feed conveyor (CV9) and will discharge to the screen feed repulping box. When the HPGR is offline, HPGR product will be reclaimed by FEL/haul trucks from the stockpiles to feed the milling screen via reclaim slots on the product bin apron using the HPGR product reclaim feeders four to six onto CV9 to maintain feed to the milling circuit.

17.2.3.5 Repulping Box and Milling Screen

Water will be added to the repulping box to de-agglomerate the HPGR product for presentation to the milling screen for efficient sizing. The water will also assist with spreading the screen feed across the full width of the screen in the feed box.

The repulped slurry will feed the wet milling screen. A double deck screen will be utilised for this duty, with the upper deck serving to:

- protect the lower deck from wear by the larger oversize particles;
- further break up agglomerates for better presentation to the lower deck;
- reduce the bed depth on the lower deck for improved screening efficiency; and,
- increase the overall deck area for dewatering and minimum moisture return to the HPGR.

Milling screen undersize slurry will report to the cyclone feed hopper, providing the new feed to the milling and classification circuit.

Milling screen oversize will report to the oversize stockpile via the milling screen oversize conveyor (CV10). Under normal operating conditions, the oversize will be reclaimed at the rate it is produced with the stockpile operating at minimum level, since it should be maintained empty to maximise the storage capacity when there is an HPGR downtime event. After filling the stockpile during a downtime event, reclaim of the dead material will be required to feed the HPGR and create capacity for future HPGR downtime events.

If the HPGR is offline for a lengthy shutdown, the oversize stockpile will need to be extended by reclaiming with FEL to create additional nearby stockpiles. The facility to recover the milling screen lower deck oversize to the screen underpan will be provided to reduce the amount of oversize generation during HPGR downtime periods. This material will be less than 8 mm, suitable for ball mill feed but not having the full benefit of the HPGR size reduction. This will slightly reduce the ball mill capacity at the same product grind, further reducing the product stockpile reclaim rate required for the HPGR shutdown duration.

17.2.4 Grinding and Classification Circuit

The HPGR crushed ore will be milled to achieve the nominated grind size (P_{80} = 106 µm) for effective gold leaching. The grinding circuit will comprise a ball mill in closed circuit with a cluster of classification cyclones with gravity gold recovery from a portion of the cyclone underflow stream.

The ball mill will effectively be reverse fed with milling screen undersize slurry reporting to the mill discharge/cyclone feed hopper and following classification and product size removal to the overflow stream, cyclone underflow will report to the ball mill feed. The ball mill will be equipped with a variable speed drive to assist with managing variations in ore blend and competency as well as feed tonnes.

When treating blended ore feeds on start-up, the ball mill will be operated at lower speeds, and a reduced ball charge to minimise power drawn for the target grind size. When treating fresh ore alone, the ball mill speed and ball charge will be increased to optimum levels for milling the competent fresh ore. Steel grinding media (mill balls) will be added to the ball mill feed hopper as required using a hoist and a ball kibble. Balls loaded into the kibble will be hoisted to the ball charging level and discharged into the ball mill.

The ball mill will discharge via a trommel to the cyclone feed hopper and be combined with the milling screen undersize slurry. Trommel oversize (worn steel grinding media) will report to the scats bunker for periodic clearing with a bobcat. Dilution water will be added to the mill discharge hopper for level control and the slurry will be pumped to the cyclones. Duty/standby pumps will be provided to maximise operating availability with variable speed pump drives to manage the cyclone feed flow and inlet pressure. The mill recirculating load will vary with ore competency, degree of fineness in the HPGR product and target grind size.

The cyclone cluster will be fitted with a number of spare cyclones to allow wear inspection and maintenance on-line. Cyclone underflow will report to the scalping screen feed splitter box while the balance of the underflow slurry will report to the ball mill feed box. Cyclone overflow will gravitate to the trash screen feed box.

Lime slurry addition to the ball mill feed box will be provided should the natural pH of the ore be acidic and require neutralisation. The mill water will acquire a degree of alkalinity over time and this requirement should diminish. Controlled additions from the lime ring main will be manually managed to maintain a near neutral solution.

17.2.5 Gravity Circuit

17.2.5.1 Gravity Circuit Feed

The gravity circuit will consist of two parallel trains each containing a feed scalping screen and a centrifugal concentrator. An intensive cyanidation reactor (ICR) will process the combined gravity concentrates from the two trains.

A portion of cyclone underflow will report to the scalping screen feed splitter box. This splitter box will allow even distribution between the two parallel circuits from a central stilling compartment. Dilution water will be added to the screen feed to reduce the slurry viscosity and improve screening efficiency. Launder gates will allow isolation of either circuit with an overflow to the mill feed.

17.2.5.2 Gravity Scalping Screens

The gravity scalping screens serve to protect the concentrators by removing the oversize from the feed that would otherwise fill the concentrate grooves and cause increased wear in the cones. The screen oversize products will be combined in a launder reporting to the ball mill feed to minimise wear from the coarse particles. The undersize from each screen will feed a centrifugal concentrator to capture the coarse gold from the milling circuit recirculating stream.

17.2.5.3 Centrifugal Concentrators

The concentrators will be operated on a semi-batch basis with periodic discharge of the gravity concentrates to the ICR. During the collection cycle, coarse and dense particles fill the grooves in the spinning bowl while the tails stream overflows the top of the bowl. Fluidising water is sprayed through ports in the back of the groove to slightly fluidise the bed and elutriate the fine light particles while allowing the denser particles to percolate deeper into the bed under the centrifugal action. The concentrate discharge cycle will require bypassing the concentrator feed, slowing of the spinning bowl, flushing out the concentrate from the collection grooves, accelerating the bowl to operating speed and reopening the feed to commence the next concentrate collection cycle. The tails slurry from the concentrators will gravitate to the cyclone feed hopper.

The concentrators will be controlled such that the concentrate discharge cycle for each concentrator occurs at the midpoint of the alternate concentrator cycle. If a concentrator is off-line, gravity concentrate loss will be minimal as the gold will remain in the recirculating load and the on-line concentrator will be fed at a higher grade. Minor gold particle flattening, and resultant reporting downstream may occur, but gold in this form will leach readily.

The gravity concentrators will be enclosed in a security area. Security close-circuit television (CCTV) surveillance will be used to monitor the secure area and access will be by swipe card permission.

Dedicated fluidisation water pumps will be provided to ensure constant pressure and flow to the concentrators as this is a key operating efficiency driver. Maintaining minimum lime additions to the mill water will also be beneficial with scaling of the fluidisation holes in the cone being a cause of downtime, but the likelihood of this occurring in the Plant will be much reduced.

17.2.5.4 Intensive Cyanidation Reactor

Gravity concentrator discharges will be stored in the ICR feed cone with excess water overflowing to grade. A batch of gravity concentrate will be leached daily under intensive cyanidation conditions. Caustic and sodium cyanide solutions from the respective storage tanks will be metered into the ICR leach solution and a leach accelerant/hydrogen peroxide will be metered into the solution to provide the required oxygen demand. Flocculant may be required to clarify the leach solution prior to pumping the gold-rich pregnant solution to the gravity electrowinning solution storage tank. ICR tails will be washed to remove the residual cyanide solution and pumped to the cyclone feed hopper for further processing.

The ICR will be located within a secure enclosure in the milling bunded area directly below the gravity concentrators. Security CCTV surveillance will be used to monitor the secure area and access will be by swipe card permission.

17.2.6 Trash Screening and Pre-Leach Thickening

Milled slurry will gravitate from the cyclone overflow to the trash screens where any oversize grit, wood chips and plastics will be removed to prevent this from becoming locked up in the activated carbon circuit. Trash material will report to the trash bunker for periodic clearing by FEL. The trash screen underflows will gravitate to the pre-leach thickener feed deaeration box.

The leach feed slurry will be thickened to reduce volume flows through the leach circuit to ensure the required leach residence time will be met. Leach reagent economy will also be improved with the required concentrations being achieved with lower additions with less solution.

Flocculant will be mixed with the pre-leach thickener feed slurry to facilitate effective settling. If fine clays are present, lime slurry addition to the thickener feed may also be required to serve as a coagulant. Flocculant will also be sparged into the thickener feed well. Flocculant will be diluted using a static mixer ahead of the dosing points to improve the dispersion into the process stream. Flocculant addition rate will be used to control the bed level of the settled solids and clarity of the overflow water.

The thickener underflow will gravity feed the underflow tank with a pinch valve controlling the outflow to ensure maximum settled density. Thickener underflow will be back diluted to (50 to 55)% w/w solids in the underflow tank with cyanide containing process water prior to pumping to the leach feed distribution box. An automatic two stage leach feed sampler will produce composite shift samples from incremental sample cuts at timed intervals over each shift.

If the thickener underflow density is too low or the downstream processes have stopped, the thickener can be put into recirculation mode.

Pre-leach thickener overflow will gravitate to the mill water tank for distribution as dilution water to the milling circuit. Decant water and raw water as required will be added to the milling water pond to make up the milling water requirements. Separation of the mill water from the process water allows for low cyanide water to be recirculated through the mill as well as maximising the benefit from the recovered cyanide in the process water.

Lime slurry, used for pH control in the leach circuit, will be added to the pre-leach thickener U/F tank from the lime ring main. The rate of lime addition will be ratioed to the dry solids leach feed tonnage rate, with the ratio being adjusted to achieve the target leach pH.

17.2.7 Leach/Adsorption

The leaching circuit will consist of seven tanks in series operating in a one pre-leach/six CIL tank configuration. The tanks will be interconnected with launders and slurry will flow by gravity through the tank train. Each tank will be fitted with an agitator to ensure the tank is well mixed. All tanks will be fitted with bypass facilities to allow any tank to be removed from service.

Pre-leach thickener underflow will be pumped to the leach feed box having been diluted with process water to the required leach slurry density. A facility to add lime slurry to the first two tanks will be provided to ensure that the slurry pH is suitable for cyanidation. Top-up may be required for some of the reactive sulphides present, where lime slurry addition in the pre-leach thickener underflow (U/F) tank may have been insufficient.

Sodium cyanide solution will be metered from the cyanide ring main based on solution concentration into the leach feed distribution box. Further addition points will be located down the leach train for use as required to maintain excess free cyanide. Lead nitrate solution will be added to the leach feed distribution box (or pre-leach thickener U/F tank) to passivate the sulphides present as it has been demonstrated to enhance the leach kinetics of the slow leaching ores. Procedures to identify slow leaching ores during grade control testing will need to be established.

Low pressure air will be added to each of the leach/CIL tanks, via air spargers below the lower agitator impeller, to satisfy the oxygen demand for the leach. Slurry from the last CIL tank will gravitate to the carbon safety screen via a two-stage slurry sampler.

17.2.7.1 CIL Circuit

Gold and silver will be leached from the milled ore using sodium cyanide and air. The dissolved gold and silver will be recovered from the leach solution by adsorption onto activated carbon. The carbon will be periodically removed from the slurry and eluted to recover the gold and silver.

The adsorption circuit will consist of six adsorption tanks (CIL tanks 2 to 7) which will be sized to provide the required total leaching residence time. Each CIL tank will be equipped with a mechanically swept intertank screen to retain carbon within the tank, and a recessed impeller carbon transfer pump to facilitate counter-current carbon transfer between adjacent tanks.

Regenerated barren carbon will be added to the final CIL tank and will be advanced counter-current to the slurry flow, allowing leached gold and silver values in solution to adsorb onto the carbon. Each carbon transfer step will result in the carbon being retained in the upstream tank by the intertank screen while the slurry will recycle back to the downstream tank. This ensures that higher grade carbon is moved to the head end of the adsorption circuit, whilst the lower grade carbon towards the tail end will more readily scavenge the low-grade solutions, minimising soluble gold losses from the circuit.

Carbon loaded with gold and silver (loaded carbon) will be recovered from the circuit by pumping slurry and carbon from the first CIL tank to the loaded carbon recovery screen. The loaded carbon will be washed on the screen to remove fine ore particles and report to the acid wash column in the elution circuit. The loaded carbon screen undersize slurry will gravitate back to the first CIL tank.

Barren tails from the last CIL tank will gravitate to the carbon safety screen to recover any carbon which may have passed through worn screens or overflowed the last CIL tank. Safety screen undersize slurry will gravitate to the tails thickener and the screen oversize (recovered carbon) will be collected in the carbon bin for subsequent return to the circuit. The facility to bypass the carbon safety screen will be provided for improved overall plant operating availability.

Barren carbon from the carbon regeneration kiln will be screened on the carbon sizing screen to remove fine carbon and quench/transfer water. The regenerated and screened carbon will discharge from the sizing screen directly into the last CIL tank.

17.2.8 Elution Circuit and Goldroom Operations

The following operations will be carried out in the elution and goldroom areas:

- acid washing of carbon;
- stripping of gold and silver from the loaded carbon using the split AARL method;
- regeneration of barren carbon;

- electrowinning of gold from elution pregnant solution;
- electrowinning of gold from gravity intensive cyanidation pregnant solution; and
- smelting of electrowinning products.

The stripping and goldroom areas will operate seven days per week if required, with the majority of loaded carbon preparation and stripping occurring during day shift. The split AARL stripping circuit will be fully automated and will contain separate acid wash and an elution column. Gravity gold electrowinning will be performed in parallel with the elution electrowinning.

17.2.8.1 Acid Wash

Loaded carbon will be received into the acid wash column from the loaded carbon screen. During acid washing, the column will be filled with a dilute solution of hydrochloric acid to remove contaminants, predominantly carbonates, from the carbon. After the soak period has elapsed, the loaded carbon will be rinsed with water. The dilute acid and rinse water will be pumped to the tailings hopper for disposal. Washed carbon from the acid wash column will be hydraulically transferred from the acid wash column to the elution column and the transfer water will be drained from the columns.

17.2.8.2 Pre-Soak and Elution

The split AARL elution process will be used to recover gold and silver from the loaded carbon in the elution column.

Strip solution will be pumped from the lean eluate tank through the inline heat transfer system (heat exchangers and strip solution heater) and injected into the base of the elution column. Caustic and sodium cyanide solutions will be pumped from the respective reagent storage tanks and injected into the suction line of the strip solution pump. The loaded carbon will be pre-soaked in the cyanide/caustic solution to prepare the carbon for gold desorption.

The carbon will then be eluted by hot strip solution which will pass out of the circuit to either of the two pregnant solution tanks for the first half of the stripping cycle, with the balance of the stripping solution reporting to the lean eluate tank. Outgoing strip solution will pass through the recovery heat exchanger to heat the incoming strip solution. The lean eluate produced in each cycle will be the first batch of strip solution in the subsequent cycle.

Once desorption is complete, the carbon will be cooled to <95°C (to avoid flashing of the solution) prior to transfer to carbon regeneration.

17.2.8.3 Electrowinning and Goldroom

Once the elution cycle has been completed, the gold and silver in the pregnant eluate solution will be recovered by electrowinning. Pregnant solution will be pumped from the selected pregnant solution tank through the parallel electrowinning cells. Direct current will be passed through stainless steel anodes and stainless-steel woven mesh cathodes within the electrowinning cells. Electrolytic action will cause the gold in solution to plate out as a gold rich sludge on the cathodes. Solution discharging from the electrowinning cell will return by gravity to the pregnant solution tank.

Solution will be recycled through the electrowinning cells to reduce the solution grade. Single pass electrowinning can be used to complete the cycle, with the barren electrolyte being pumped to the leach circuit. The electrowinning cycle duration will be timed to achieve the barren solution target gold grades.

An overhead crane within the goldroom will be provided to assist with handling of cathodes and anodes. The gold sludge will be removed from the cathodes by high pressure water washing. The resulting slurry will be filtered in a vacuum pan filter and the solids then dried in an oven. The sludge will then be direct smelted with fluxes in an induction heated furnace to produce doré bars. Slag from smelting operations will be returned to the milling circuit.

Fume extraction equipment will be provided to remove electrowinning cell gases and off gas from the drying oven and smelting furnace. In addition to this, ventilation fans will be provided to ensure there is adequate fresh air recycling inside the gold room. A wet scrubber on the smelting furnace fume extraction will be used to recover any gold particles and dust.

The gravity pregnant solution tank and electrowinning cell will be located adjacent to the elution pregnant solution tanks and cells. Pregnant solution will be pumped through the electrowinning cell with electrowon gold plating on the cathodes. The gold from the gravity electrowinning cell cathodes will be treated separately to the CIL/elution electrowinning cathodes to assist in metallurgical accounting. The gravity barren electrolyte will be combined with the elution barren electrolyte and pumped to the leach circuit.

17.2.8.4 Carbon Regeneration

The carbon regeneration circuit will consist of a dewatering screen, electrically heated regeneration kiln, quench tank and carbon sizing screen.

Barren carbon will be hydraulically transferred from the elution column to the carbon dewatering screen. The dewatering screen will remove the majority of the transfer water prior to discharging into the feed hopper of the regeneration kiln.

In the regeneration kiln feed hopper, residual and interstitial water will be drained from the carbon before it enters the kiln. In the kiln, the carbon will be heated to (650 to 750)°C to allow effective reactivation to occur. Reactivated carbon from the kiln will discharge to the quench tank where it will be cooled with raw water. The cooled carbon will be pumped to the carbon sizing screen, where fine carbon and the majority of transfer water will be discharged into the CIL tailings stream. The carbon will discharge to CIL Tank 7 at the end of the CIL train. Regeneration of back-to-back carbon batches can be conducted to minimise cooling of the retort between batches.

17.2.9 Tailings Disposal

17.2.9.1 Tails Thickening

CIL tails will gravitate to the tails thickener via the carbon safety screen. The tails thickener will be used to thicken the plant tails in order to recover free cyanide and gold in solution, thereby also reducing the amount of cyanide discharged with the plant tails. The thickener underflow will be diluted with decant return and/or raw water to reduce the cyanide concentration in the plant tails to the required discharge limit.

Carbon safety screen underflow will report to the tails thickener feed and be mixed with flocculant to facilitate effective settling. Thickener overflow containing the majority of the cyanide and soluble gold, will overflow to the process water tank to be mainly recycled as pre-leach thickener U/F dilution water for cyanide re-use in the leach circuit, and for distribution to the CIL screen sprays and service points.

Tails thickener underflow will be sampled to check final gold and cyanide values exiting the plant aligned with a mass flow measurement for metallurgical accounting purposes. The tails thickener overflow will also be sampled to allow modelling of the recovered gold and cyanide in this water stream.

Tails thickener underflow will be combined with any other plant waste streams in the tails tank. As required, the plant tails will be diluted with decant return and/or raw water to reduce the cyanide discharge concentration, prior to pumping to the TSF. The facility to recirculate the tails back to the thickener to build up the settled density will be provided, but this will only be used when the plant is down to avoid recycling the acid waste streams to the thickener feed. The facility to bypass the tails thickener will be provided to increase overall operating availability.

Tailings will be deposited into the TSF using established approaches to spigot discharge for cyclical beach deposition and drying phases as described in Section 18. Supernatant water (decant return) will be directed to a pond around the decant tower and will be pumped to the plant for re-use as tails dilution and mill water make-up when pond levels are high. UV degradation of the cyanide in the supernatant pond along with rainfall dilution and bacterial decay should ensure that very low levels of cyanide are present in the decant return water making this a good, low cyanide alternative to raw water make-up.

17.2.10 Reagents

17.2.10.1 Reagent Storage

Reagents will be received on site either in bulk or in shipping containers, with a minimum of twelve weeks capacity stored on site to ensure that supply interruptions due to port, transport or weather delays do not restrict production.

Reagent mixing design aims to cater for daily mixing of a reagent batch. No specific reagent operators have been appointed, so plant operations personnel will make up new batches as and when required. Mixing tank designs are based on the use of an exact number of bulk bags or IBCs to achieve the desired solution concentration. In some cases, this will result in longer times between mixes.

17.2.10.2 Lime

Hydrated lime slurry will be prepared in a vendor supplied mixing plant for use in the process. The lime mixing plant will consist of a lime hopper with bag breaker, dust collector, screw feeder and vortex wetter.

Hydrated lime will be delivered to the site as a dry powder in bulk bags within a container (24 t delivery size). Bulk bags will be split in an enclosed bag breaker discharging into the dry lime hopper and metered via the lime screw feeder to the vortex mixer. Raw water will be added to the vortex mixer in proportion to the dry lime feed, measured by screw rotations, to achieve a 20% w/w solids concentration in the slurry discharging to the agitated milk of lime tank.

The milk of lime slurry will be pumped from the lime mixing and storage tank via a ring main, to dosing take-offs at the pre-leach thickener U/F tank, the pre-leach thickener feed box, the ball mill feed spout, and leach/CIL tanks 1 and 2.

Lime usage will typically be very low, so the circulating rate in the ring main will be high relative to the dosing point flows. Dosing will be achieved using pneumatically pulsing diaphragm valves allowing full bore flow for brief intervals rather than restricted continuous flow which would be likely to block. pH control or a manually input set-point will dictate the opening frequency. The metering line and valves will be fitted with rod-out cleaning provisions.

17.2.10.3 Cyanide

Sodium cyanide will be delivered as dry briquettes in bulk bags (1 t). The cyanide bags will be added to the agitated mixing tank via a bag breaker and dissolved in filtered water to achieve the required solution strength.

The cyanide solution will be transferred to the storage tank for use in the process. Cyanide will be reticulated to the leach circuit via a ring main and dosed to the leach/CIL tanks as required. A dedicated pump will meter the cyanide solution to the elution circuit and ICR as required.

17.2.10.4 Caustic

Caustic soda (sodium hydroxide) will be delivered to site as dry 'pearl' pellets in bulk bags (1 t). The bulk bags will be added to the mixing tank via a bag breaker on the receiving hopper. The pellets will be discharged into the agitated mixing tank via a screw feeder providing a controlled addition rate and preventing splash back from the tank with the exothermic local heating on contact with the water. The caustic pellets will dissolve in the filtered water to the required solution strength.

The caustic dosing pump will meter the caustic solution to the elution/electrowinning circuits and ICR as required.

17.2.10.5 Hydrochloric Acid

Concentrated hydrochloric acid (HCL) will be delivered to site in Intermediate Bulk Containers (IBCs) (1 m^3). The concentrated hydrochloric acid will be pumped into the acid mixing tank where it will be diluted with filtered water to achieve the required acid wash solution concentration. The dilute acid solution will be pumped to the acid wash column as required.

Space will be allowed adjacent to the dilute acid mixing tank to allow for the use of concentrated HCL delivered in isotainers, being a safer, less labour-intensive handling solution.

17.2.10.6 Activated Carbon

Activated carbon will be delivered in bulk bags (0.5 t). Carbon will be added via the carbon quench tank as required for carbon make-up to the CIL inventory. This addition point will allow any fine carbon particles to be removed on the carbon sizing screen and combined with CIL tailings slurry for disposal. The first bulk fill of carbon to CIL will be lifted to the top of the CIL and the carbon dumped directly into the tanks.

17.2.10.7 Grinding Media

Grinding balls will be delivered to site in drums or bulk bags. The balls will be emptied into the ball storage bunker. From here, the balls will be loaded into kibbles which will be lifted and discharged into the ball charging hopper. The balls will gravity flow to the ball mill feed box. Ball charging will be initiated once or twice daily to maintain the target power draw. The mass of ball additions to the mill will be measured by the number of kibble loads.

17.2.10.8 Flocculant

Flocculant will be delivered to site in bulk bags (750 kg) as a dry powder and will be added to the flocculant plant feed hopper. The vendor supplied flocculant mixing plant will automatically mix batches of flocculant with raw water and transfer the mixed flocculant to an aging/storage tank after each mixing cycle is complete. Flocculant storage tanks will be located at each thickener, with make-up of new batches of flocculant being driven by which storage tank has the available volume for transfer of a new batch. Flocculant will also be transferred to a pre-used IBC for use in the ICR.

Dedicated dosing pumps will meter the flocculant solution to the pre-leach thickener and tails thickener as required. Flocculant will be diluted using a static mixer ahead of the dosing points to improve the dispersion into the process stream.

17.2.10.9 Lead Nitrate

Lead nitrate will be delivered to site in bulk bags (1 t) as a dry powder and will be added to the lead nitrate feed hopper and bag breaker to be broken into the agitated lead nitrate mixing tank. The lead nitrate bag feed hopper and bag breaker will be enclosed with a dust extraction and wet scrubber unit to ensure operators are not exposed to harmful lead particles.

Batches of lead nitrate solution will be mixed by adding the bulk bags to filtered water in the correct ratio as required. A buffer tank with a common suction to the mixing tank will allow for dosing of lead nitrate at the standard concentration during mixing cycles. Dedicated pumps will meter the lead nitrate solution to the pre-leach thickener U/F tank as required.

17.2.10.10 Hydrogen Peroxide

Hydrogen peroxide solution (50% w/w) will be delivered to site in IBCs (1 m^3). The hydrogen peroxide will be used in the ICR only (not added to CIL leach).

17.2.10.11 Fluxes

Sodium borate (borax), silica flour, sodium nitrate (nitre) and sodium carbonate (soda ash) will be used as fluxes for gold smelting. The fluxes will be delivered in 25 kg bags and mixed in small quantities with the gold sludge prior to smelting.

17.3 Plant Services Description

17.3.1 Water

17.3.1.1 Raw Water

Site raw water supply and management is discussed in Section 18, whilst the Plant's water requirements are discussed herein.

The plant raw water tank will have sufficient capacity to minimise the impact of short-term supply interruptions. The plant raw water pumps will distribute raw water to the plant and mine services area (MSA). Raw water make-up to the plant will be via the mill water pond supplied by overflowing the raw water tank or fed directly to the pond if the raw water tank level has been drawn down. Decant return water will be used to make-up any additional mill/process water requirements.

The plant will nominally require 247 m^3/h of raw water make-up, which equates to 0.42 t of water/t of RoM ore (db). Supplementing this make-up requirement is 351.4 m^3/h of tailings return water (Decant) which is used for diluting the tails prior to pumping to the TSF, with the remainder being pumped to the mill water pond. The Site water balance is described more fully in Section 18.

17.3.1.2 Fire Water

Fire water systems for plant and infrastructure are described in Section 18.

17.3.1.3 Mill Water

The pre-leach thickener will overflow to the mill water tank which serves to capture the bulk of any silt carryover and this tank will overflow (occasionally) in turn to the mill water pond (dirty side). The mill water pond will be split into two sections, the 'dirty' and 'clean' sides. The incoming make-up water will report to the dirty side to allow settling of entrained silt in this section, which will operate with a continuous overflow to the clean side. Both sections of the pond will be double lined with leak detection pumps for demonstrating cyanide compliance. The dirty side pond will be provided with a low point valved drain to facilitate removal of accumulated silt via connection to the mill water pump suction or direct outlet into the pre-leach thickener bund.

The mill water pumps will provide all the dilution, spray water and service water requirements for the milling, gravity, classification, and trash screening circuits. The mill water pumps draw directly from the mill water tank recirculating any silt that overflow the thickener to prevent accumulation in the mill water pond. These pumps will have a common suction that extends into the clean mill water pond section, so that they can draw from this larger inventory of stored water. With a normal shortfall in water supply from the thickener overflow, the balance of the mill water will draw from the pond. The level in the mill water tank will always match or be slightly higher than the mill water pond level due to the common suction header, and it is therefore critical that the pond low level is no lower than the mill water tank pump suction outlet nozzle. A check valve will be placed on the common suction between the mill water (dirty side) and gravity concentrator fluidisation (clean water side) pumps to provide further security that the thickener overflow will not flow to the pond via the clean side of the common suction. Instead, pond water will be drawn into the mill water pump suction in the event of the pond level exceeding the mill water tank level.

The gravity concentrator fluidisation water pumps will draw from the clean side suction line.

Decant water and raw water will be pumped to the mill water pond to provide the process water make-up required by maintaining the pond level.

17.3.1.4 Process Water

Plant process water will be sourced from the tails thickener overflow and will contain the recovered cyanide from the CIL tails stream. Maintaining the cyanide concentration to maximise its utility in recycling to the process will be a priority. This water will provide all the screen sprays, dilution, and service water downstream of the pre-leach thickener. Excess process water caused by temporary water circuit imbalance will be discharged to the tails tank. Minimum make-up water to the process water supply should be required, but raw water will be provided for start-up.

Antiscalant will be added as required to condition the process water and reduce fouling of the pipelines, spray nozzles and screen decks given the lime content.

17.3.1.5 Fluidising Water

Fluidising water will be provided by dedicated fluidising water pumps to the centrifugal concentrator. The pumps will draw water from the clean mill water pond for minimum fouling and consistency of supply flow and pressure in the centrifugal concentrators.

17.3.1.6 Filtered Water

A filtered water treatment plant will be installed to treat the raw water to remove any suspended solids and bacterial contaminants. The treatment plant will consist of clarification through flocculant addition, sand filtration, carbon filtration and biocide dosing.

Filtered water will report to the filtered water tank and will be distributed to the Plant by the filtered water pumps for use as gland water, elution, and reagent mixing. Other minor users will include dust suppression systems, cooling water make-up, feeds to the titration room and cyanide analysers, high pressure washers at the CIL and goldroom and non-potable water to the workshops and laboratory.

17.3.1.7 Gland Water

Water from the filtered water storage tank will be distributed as gland service water using gland water pumps.

17.3.1.8 Potable Water

Potable water supply and distribution for plant and infrastructure, is summarised in Section 18.

17.3.1.9 Cooling Water

Cooled water for the ball mill and HPGR will be provided using an evaporative water-cooling tower and heat exchanger. Filtered water will be added to the cooling tower sump to supplement the water loss from evaporation and blowdown. Antiscalant and biocide will be dosed into the cooling water system to maintain water quality.

The ball mill variable speed drive (VSD) will be chilled water cooled, using a dedicated closed circuit refrigeration unit. The ball mill main motor will be water jacket cooled, via open circuit raw water circulation using the raw water tank as a heat sink.

17.3.1.10 Leach, Plant, and Instrument Air

Low pressure (LP) air will be supplied using low pressure blowers synchronised to swing the loaded blower on and off-line to provide the required LP air volume. The air will be oil-free to avoid carbon fouling and will be reticulated to the leach and CIL tanks and injected into the slurry down the agitator shafts, with air spargers at the bottom for each tank.

Plant and instrument air will be supplied from air compressors. All compressed air will be filtered and dried before distribution to the various area specific air receivers which will supply the instruments and plant service air requirements. Dedicated instrument air receivers will be provided in key areas to ensure that a secure supply is available for the critical instruments and valves that cannot be drawn down via the air service points.

A dedicated plant air compressor and receiver will be located in the crushing area.

17.3.2 Diesel

Diesel supply, storage, and distribution for the mine as a whole is summarised in Section 18. From the bulk fuel storage facility located between the MSA area and the emergency backup power station. A site tanker will distribute diesel to the other various points of use, namely:

- Plant storage tank for gravity feed to the elution heater via head tank;
- Plant incinerator;
- assay laboratory storage tank if fire assays are adopted;

- accommodation and security camp emergency generators;
- water bore field generators;
- site fire water diesel pump tanks; and
- storage tank for refuelling of site light vehicles.

Diesel usage by process/facility area is indicated below (Endeavour, 2022):

- Elution: approximately 860 m^3/a
- Process/Infrastructure (plant vehicles, incinerator, and small diesel gensets): approximately 990 m^3/a

17.3.3 Electrical

17.3.3.1 Overview

The electrical power supply and associated power demand (grid and diesel generated) for the Plant and infrastructure (camps, airfields, water management infrastructure; general offices and buildings and mining facilities) is described in Section 18, whilst the power demand for the Plant is described in Section 17.3.3.2 following.

17.3.3.2 Plant Power Demand

Installed power, absorbed power and power consumed per annum for the Plant is summarised in Table 17-6 following.

Table 17-6: Plant Power Requirements (Fresh Rock), (Lycopodium, 2022b)

Plant Area	Installed Power Power kW	Peak Continuous Power Draw kW	Avg Continuous Power. Draw kW	Power Consumed MWh/a
Feed Preparation	2374	1864	1280	11 214
Milling	14 267	11 432	9516	83 359
Trash Removal and Thickening	288	162	134	1175
Leaching	1245	1045	871	7626
Elution and Gold Room	1842	1441	985	8633
Tails Thickening and Pumping	573	270	232	2030
Reagents	145	112	37	326
Water Services	2240	953	813	7119
Air Services	702	505	468	4097
Plant Total	**23 676**	**17 783**	**14 336**	**125 579**

17.3.3.3 Electrical Distribution

The electrical system design for the Mine/Plant is based on 11 kV distribution and 415 V working voltage. System frequency is designed at 50 Hz. The largest drives within the Plant will be the ball mill (7700 kW) and the two HPGR motors (2 x 1350 kW).

Power supply for the Plant will be from the site incomer HV switchyard, via the main 11 kV switchboard. The 11 kV supply will be reticulated from a tariff metered plant feeder to the Plant 11 kV main switchboard.

The main Plant distribution supplied from the Plant main 11 kV switchboard will comprise of nine containerised switchrooms located in following areas:

- Plant Main 11 kV Switchroom.
- Primary Crushing Switchroom.
- Secondary Screening Switchroom.
- Milling Area Switchroom.
- HPGR Product and Reclaim Switchroom.
- HPGR Switchroom.
- Pre-Leach and Services Switchroom.
- Elution and Goldroom Switchroom.
- CIL and Tailings Switchroom.

An 11 kV overhead power line fed from the Plant main 11 kV switchboard will provide power supply to remote and non-process infrastructure facilities as described in Section 18.

17.3.3.4 Emergency Back Up Power Supply

An emergency backup power station using diesel generators will be provided for the Mine/Plant to supply only critical power loads to the Plant during planned and unplanned outages on the grid. Additional detail is provided in Section 18.

17.3.3.5 Electrical Buildings

Electrical switchrooms will be designed to house the low voltage (LV) Motor Control Centres (MCCs), MV switchboards, VSD and Process Control System (PCS) hardware. The switchrooms will be sealed against dust ingress and be fitted with air conditioning, uninterruptible power supplies (UPS) and fire detection systems.

The switchrooms will be mounted on 2 m high steel pedestals to facilitate cable installation below the switchroom and bottom entry connection to the internal equipment through gland plates. Entry to the rooms will be via stairs and access platforms constructed at each end.

17.3.3.6 Transformers and Compounds

All 11 kV/415 V distribution transformers will be of ONAN (non-fan forced) cooling configuration and vector group Dyn11.

Fire rated concrete walls will be constructed around the pad mounted transformers where necessary.

Distribution transformers will be rationalised in the electrical design to minimise spares holding requirements.

17.3.3.7 MV Switchboards (11 kV)

One 11 kV Plant Main Switchboard has been allowed for in the plant. The indoor 11 kV switchboard will be a withdrawable design. The 11 kV switchboard will be supplied with protection, metering, and earthing facilities.

The design fault level and circuit breaker ratings adopted are:

- 11 kV switchboard busbar 2500 A, 25 kA at 3 seconds.
- 11 kV incomer circuit breakers 2500 A.
- 11 kV feeder circuit breakers 630 A.

Protection will be provided by microprocessor-based protection relays.

17.3.3.8 Motor Control Centres (MCCs)

The LV MCCs will be single-sided and housed in the LV switchrooms (listed in Section 17.3.3.2). Construction of all MCCs will have Form 4b segregation, Type 2 coordination. Starters in MCCs will have a demountable design and main incoming circuit breakers will have a withdrawable design complete with protection.

Motor starters up to 90 kW will be equipped with thermal overload protection and electronic protection for all larger drives. The LV MCCs will supply power to the LV motors, LV variable speed drives and LV distribution boards.

17.3.3.9 Variable Speed Drives and Soft Starters

Low voltage VSD units will be supplied from the LV MCC's. These units will be installed along the internal wall of the relevant LV electrical switchrooms. Soft starters where applicable, will be installed inside the starter modules in the MCC for each drive.

17.3.3.10 Fire Protection

All switchrooms will be provided with local fire detection systems consisting of Very Early Smoke Detection Apparatus (VESDA) sampling for the switchboard. Signals from the fire detection system will be wired to the respective Fire Indication Panel (FIP) in the switchrooms and all signals will be passed onto the control system for alarming on the plant SCADA system. Each FIP will also be wired to a local siren with beacon to warn staff of the fire detection.

17.3.3.11 Earthing System and Lightning Protection

The earthing system within the plant will be designed in accordance with relevant Australian Standards. The following method of system earthing will be implemented at various voltage levels:

- 11 kV earthed via earthing transformer or neutral earthing resistors (NER) at power station.
- 415 V solidly earthed system/Multiple Earthed Neutral (MEN)/T-N-C-S.

Earth stakes and grading rings will be provided around the switchrooms to mitigate against step and touch potential risks.

Lightning protection will be provided for buildings and structural steel as appropriate. Lightning protection systems will have their own independent earthing electrodes and will be interconnected with the power earthing system.

17.3.3.12 Electrical Field Installation

Cables up to 25 mm^2 will be PVC insulated and larger cables will be XLPE insulated.

VSD cables will be three phase, with three earth cables symmetrically laid out within an overall shielded cable.

In general, cables within the plant area will be installed above ground on cable ladders and follow the pipe racks wherever possible. Cables to equipment in open areas such as process water pumps and concentration pumps will be partially installed underground in conduits for ease of access and to minimise clashes with pipework.

Cable ladders will generally be laid horizontally, with vertical ladders only used in areas where regular spillage may occur. Hot dip galvanised cable ladder will be used. Ladder routes will in general follow the pipe racks.

Cables of different voltage groups will generally be installed on separate ladders. Where they need to be installed together, segregation in the form of barrier strips will be provided.

Sun covers will be provided over the top level of all cable ladder to provide protection against UV damage and plant spillage.

17.3.3.13 Lighting

Plant lighting will be designed in a fit for purpose manner to suit the operational requirements of the plant. LED luminaires will be used to maximise light spread and energy efficiency. Enclosed areas and staircases will be fitted with traditional swivel lighting poles. Vibration resistant fittings and auxiliaries will be used where required.

Flood lights and high-bay luminaires will be provided for perimeter, general area, and workshop lighting.

UPS maintained emergency light fittings will be installed as required throughout the plant to ensure that personnel can safely negotiate obstacles in substations, control rooms, stairways, access ways and safety shower locations.

17.3.3.14 Harmonic Filters

All LV VSDs considered for this study have active front ends to reduce harmonics. Therefore, no additional harmonic mitigation will be included at the LV MCCs.

17.3.3.15 Power Factor Correction Panel

A power factor correction panel of size 6.25 MVAr with 5 x 1.25 MVAr stages will be connected to the plant main 11 kV switchboard.

17.3.4 Control System

The process control system employed for both Plant and non-plant process infrastructure is summarised in Section 18.

17.3.5 Communications

Site wide communications infrastructure to be provided for the Project/Mine including; network topology, external connectivity, radio communications, security network and server/computer infrastructure, is described in Section 18.

17.3.6 Security and CCTV Systems

Security systems employed on the Mine including, access control and closed-circuit television (CCTV) systems are summarised in Section 18, whilst the process and operational control CCTV systems are described below.

A network of process CCTV cameras will be installed with viewing facilities for control room operator information as well as within the plant offices for process operations checks given the expansive plant layout and relatively low manning levels per shift. Note that the goldroom and gravity concentrate recovery and treatment areas are excluded from the process network, as these will be covered by security, and duplicate coverage may be considered a security risk.

Camera locations will facilitate remote monitoring of all operating areas where possible including aspects such as feeder and conveyor burdens, bin and stockpile levels and screen decks. Area overviews will also be facilitated, for example HPGR product stockpiling, thickener surfaces and overflow (O/F) clarity, CIL tanks and carbon screens.

17.4 Metallurgical Accounting

Provision will be made for monitoring instrumentation and field measurement totalisation to assist with metallurgical accounting for the operation. Automated sampling of key streams will also be provided to allow consistent incremental and composite shift samples to be prepared for assay, sizing, and SG measurements.

17.4.1 Weightometers

Weightometers will be located on the various conveyors throughout the plant:

- Crusher Discharge Conveyor, CV1- primary and secondary crushed ore. New crushing plant feed will be estimated from CV1 to CV5 tonnages. This value is not used for control as the circuit will be balanced to manage the secondary crusher feed bin level and recirculating load. The stockpile feed conveyor (CV3) will provide a direct measurement of crushing circuit throughput.
- Secondary Crushing Feed Conveyor (CV5) - crushing screen oversize material reporting to the secondary crusher feed bins (secondary crushing recirculating fraction).
- Crushed Ore Stockpile Feed Conveyor (CV3) - crushed ore (crushing screen undersize) reporting to the crushed ore stockpile.
- The secondary crushing recirculating load will be calculated as the ratio of the tonnage feeding the secondary crushing bins to the stockpile feed tonnage (crushing circuit product). Comparison of totalised tonnages or moving averages will be more informative than instantaneous tonnages, with feed rate variability and the impact of intermediate surge capacity being smoothed out over time.
- HPGR Feed Conveyor (CV7):
  - Crushed ore reclaimed from the crushed ore stockpile as new feed to HPGR crushing circuit.
  - HPGR total feed (combined stockpile reclaim and milling screen oversize) located after the milling screen oversize reclaim to the HPGR feed conveyor.
- Milling Screen Feed Conveyor (CV9) - HPGR product reclaimed from the HPGR surge bin (and the extended product stockpiles if applicable) reporting to the milling screen.
- Milling Screen Oversize Conveyor (CV10) - milling screen oversize recycled to the HPGR feed. New milling circuit feed can be estimated from the difference between the milling screen feed tonnage and oversize tonnage.
- The milling screen oversize tonnage can be cross checked by comparing the difference between HPGR Feed Conveyor A and B tonnages, and the measured oversize tonnage. Comparison of totalised tonnages over a period or moving averages will be more representative, with the intermediate surge capacities affecting instantaneous tonnages.

17.4.2 Mass Flow Measurements

Density and flow meters on the pre-leach thickener U/F and leach feed streams are required for control. This provides two similar measurements of mass flow allowing dry solids tonnes to be estimated using the assumed solids SG. Averaging of the two similar mass flow measurements will provide the most accurate direct continuous measurement of plant feed tonnage. Tying in this tonnage to the sample assays for this stream, provides the gold feed ounces excluding gravity recoverable gold.

Similarly, the density and flow metering on the plant tailings line (tails thickener U/F) will provide the dry tonnage of solids pumped to the TSF. The tails sample assays will allow determination of the residual gold loss from the circuit. The gold recovered in the leach/CIL circuit can be calculated from these data inputs.

The totalised mass flow measurements can be cross checked against the milling circuit feed tonnage estimates determined from the HPGR circuit weightometers.

Automated stream samplers on the leach feed and adsorption tails streams will ensure reliable and consistent composite shift samples for leach head grade and tails solution and residue grades.

17.4.3 Gravity Gold

A dedicated electrowinning cell will be provided for recovery of the gold leached by intensive cyanidation of gravity concentrate. Poppet samplers on the cell feed and barren solution lines will take timed samples over the duration of the electrowinning cycle and barren solution return to allow assessment of the gold recovered to the cathodes, and the additional gold recycled to the leach circuit. An accurate measurement of the net solution volume treated will be recorded for each electrowinning cycle. The gravity gold sludge recovered can be smelted separately to allow gold weight measurement for comparison with the calculated recovery. The average plant head grade over each period can be back-calculated from the gravity gold recovered and leach head grade.

Regular gold in circuit (GIC) surveys will allow reconciliation of precious metals in feed compared to doré production.

17.4.4 Other

Water supplied and used in the various areas will be continuously monitored and totalised.

Reconciliation of the amount of reagents used over relatively long periods will be achieved by delivery receipts and stock takes. On an instantaneous basis, reagent usage rates to unit operations will be measured (L/min) and accumulated (m^3) using flow meters.

17.5 Data Verification

The data verification process applied in ensuring that the data used and presented herein is valid and suitable for use, is discussed in Section 12.

17.6 Comment for Section 17

The QP for Section 17 considers that in the development of the Plant design and operating cost estimate, CIM best practise guidelines for mineral processing have been applied (CIM, 2011), and the level of technical development undertaken is in accordance with industry standards for a DFS.

17.7 Interpretations and Conclusions

Interpretations, conclusions and risks for Section 17 are presented in Section 25 of this Report.

17.8 Recommendation

Recommendations for Section 17 presented in Section 26 of this Report.

17.9 References

References cited in the preparation of Section 17 are presented in Section 27 of this Report.

18. PROJECT INFRASTRUCTURE

18.1 Overview

The Lafigué Project (the 'Project') is a greenfields development comprising of the following new infrastructure/facilities to support mining, processing, and waste management on the Lafigué Mining License (PE 58 or the 'Site'):

- Site roads (haul and general) and the public access road to the mine from the existing national road network.
- Airstrip.
- 225 kV power supply and transmission line (33 km) from a Côte d'Ivoire ENERGIES (CI-ENERGIES) switchyard at Dabakala.
- Power distribution on site (26 MWe installed load) and emergency power generation (3650 kVA installed capacity).
- Site facilities/services, including but not limited to; water harvesting/abstraction, storage, and supply/treatment; sewage treatment; general non-production waste management; communications; fuel supply; medical, analytical, cleaning, catering and laundry, security; and maintenance and transport services.
- Plant and general infrastructure buildings.
- Tailings Storage Facility (TSF).
- Surface Water Management and Sediment Control.
- Mine Services Area (MSA) consisting of infrastructure facilities to support the mining operation.
- Emulsion and explosive storage facilities.
- Accommodation and recreational facilities for non-local staff at the Permanent Camp, Gendarmes Barracks.

Figure 18-1 to Figure 18-3 following present a series of site layout drawings, for PE 58 and the mine and the plant respectively.

Offsite enabling infrastructure is discussed more fully in Section 5.

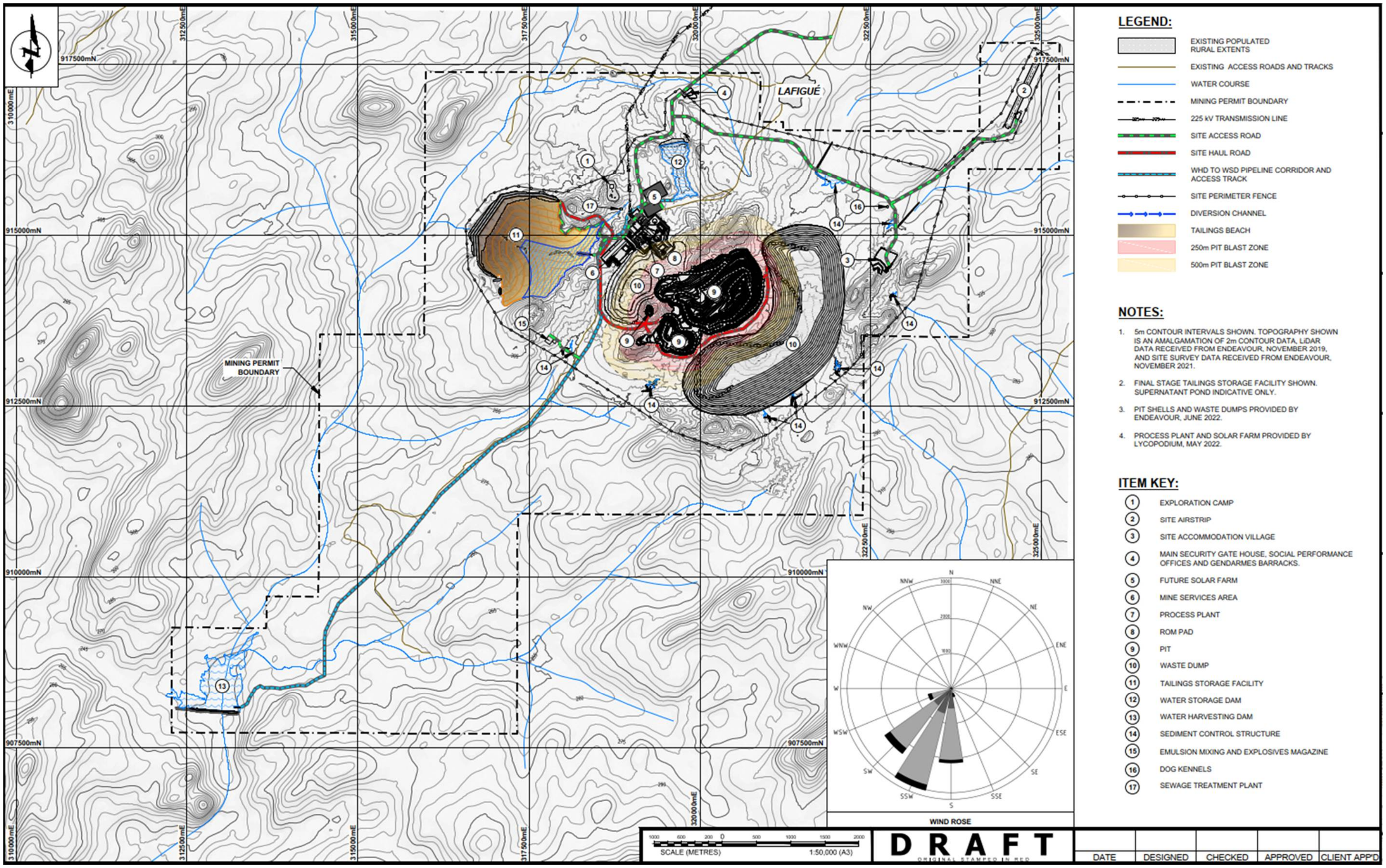


Figure 18-1: PE 58 License and Project Layout (KP, 2022b)

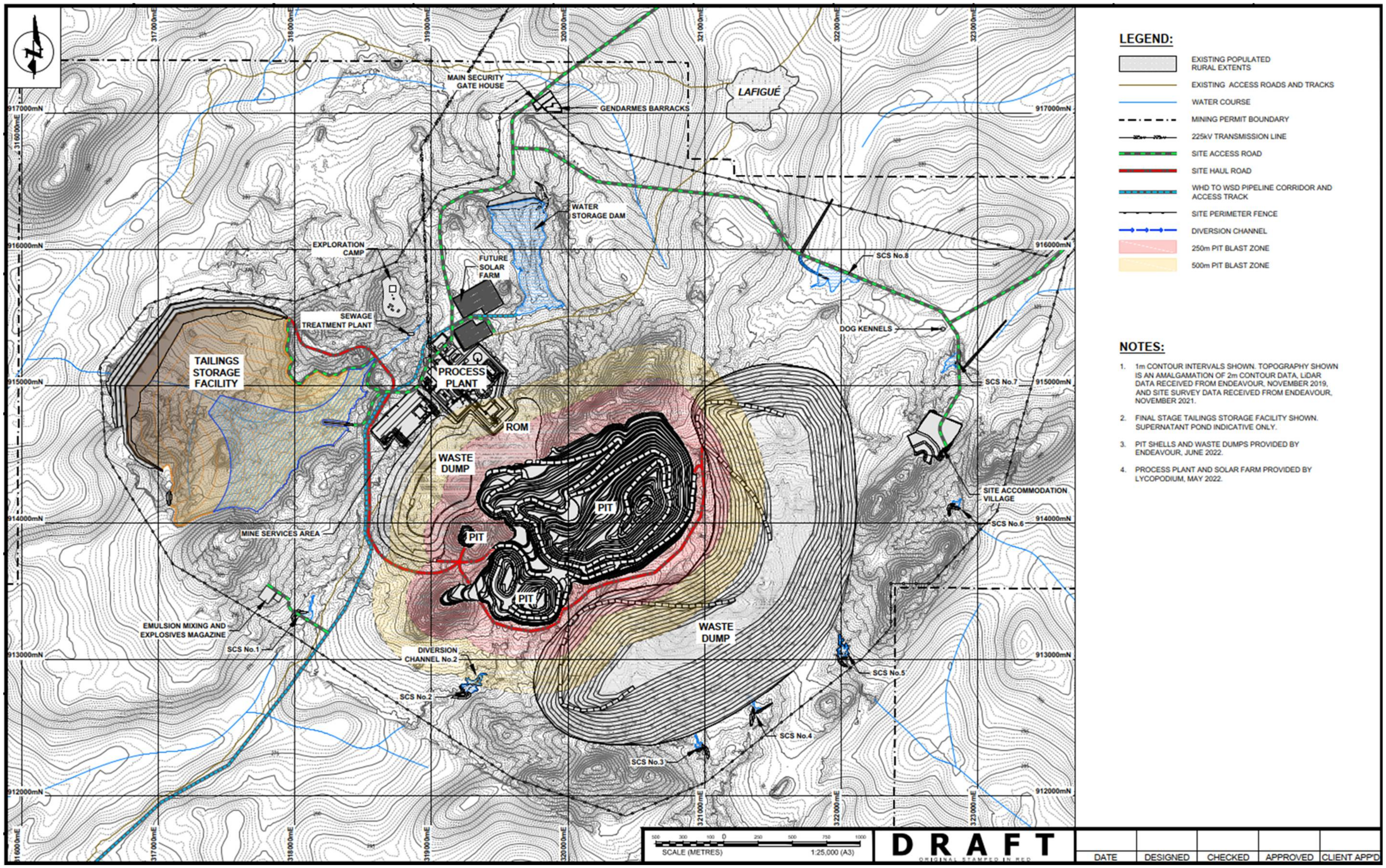


Figure 18-2: Project Site General Arrangement (KP, 2022b)

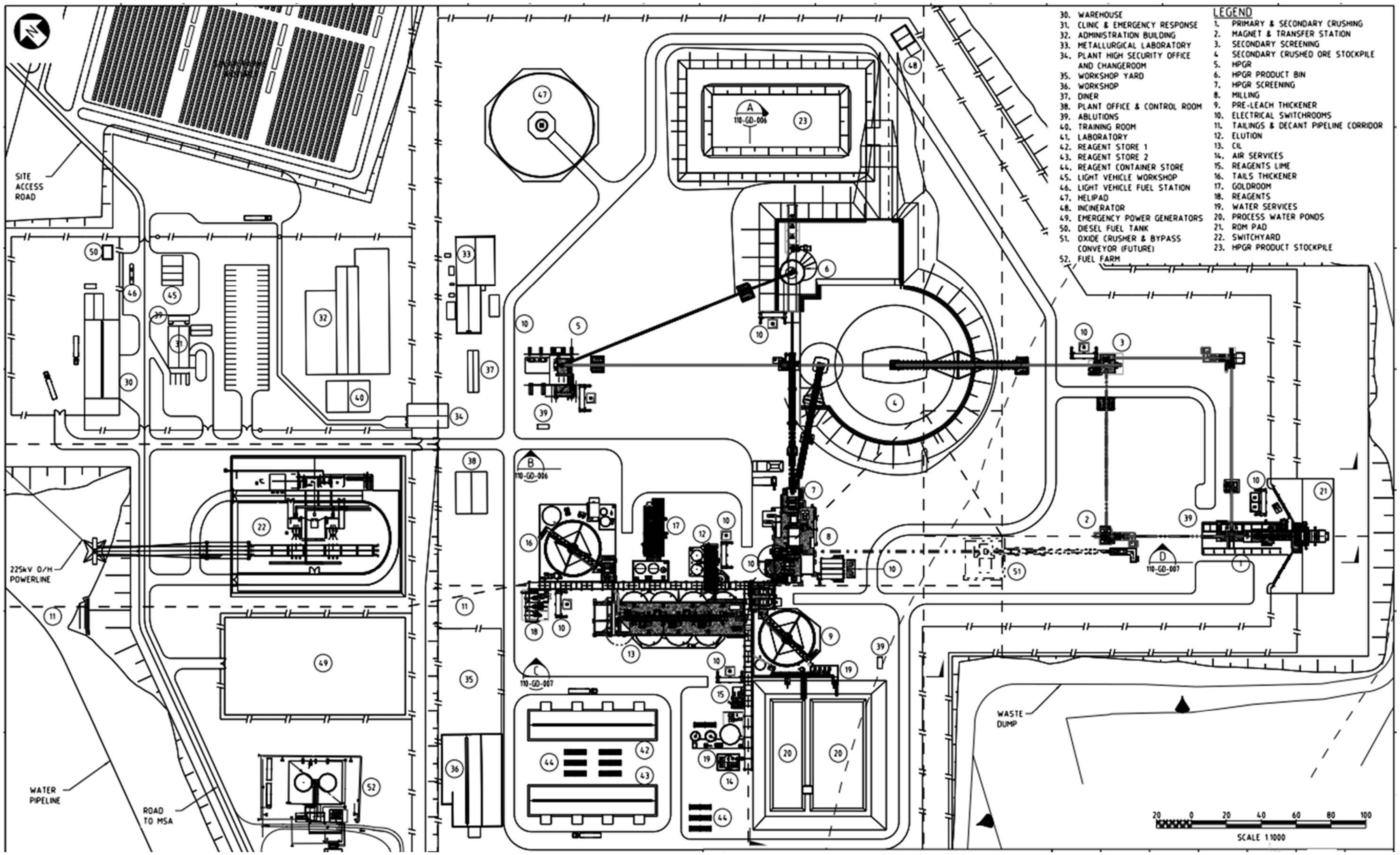


Figure 18-3: Project Layout of Plant and Central Infrastructure Buildings (Lycopodium, 2022a)

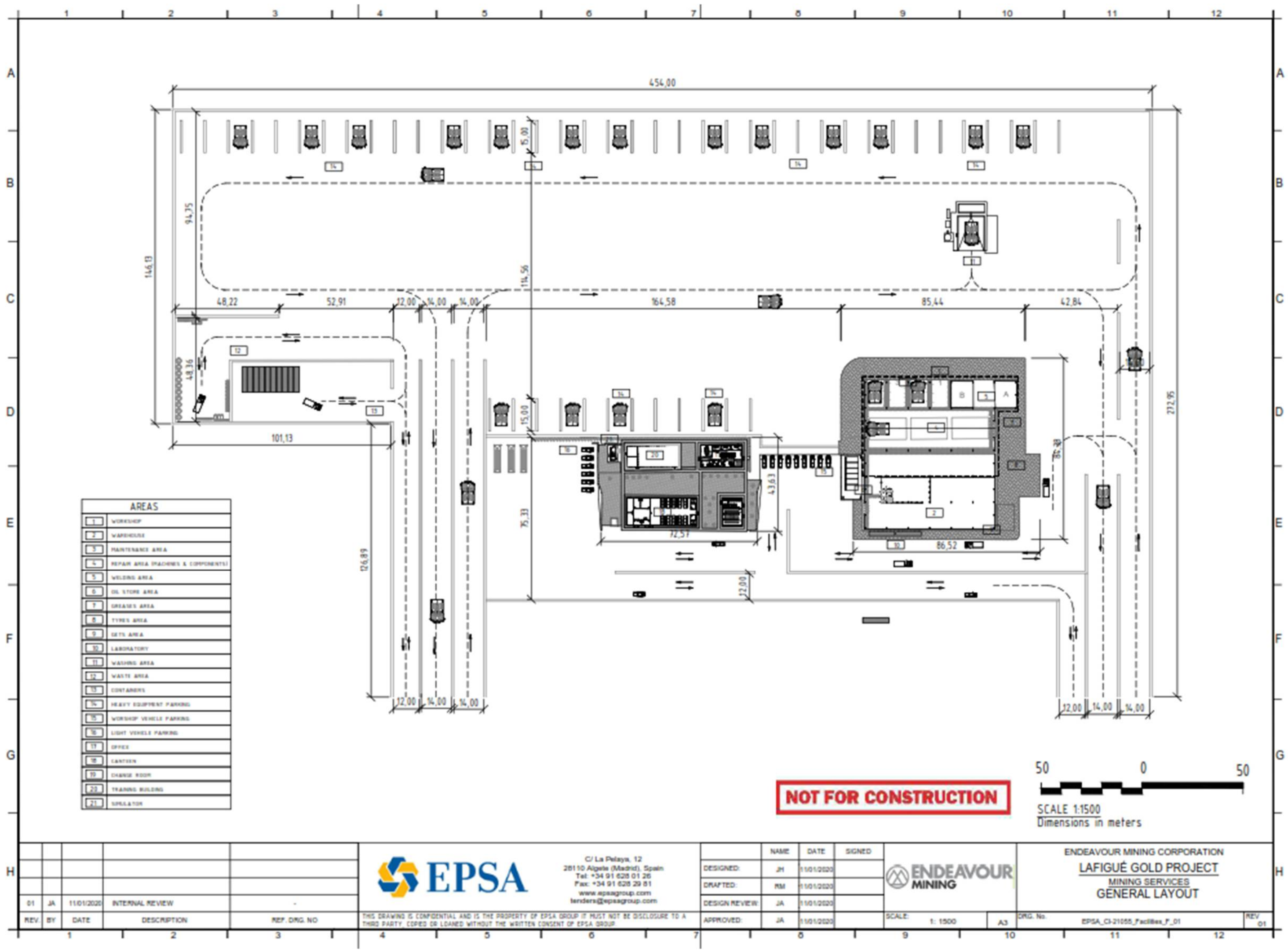


Figure 18-4: Indicative MSA Layout (EPSA), (Lycopodium, 2022a)

18.2 Earthworks and Site Preparation

18.2.1 Topography and Vegetation

Background topography and vegetation information as it pertains to terracing and earthworks is discussed in Sections 5 and 20 of this Report.

18.2.2 Geotechnical Considerations

18.2.2.1 Geotechnical Investigations

To assess the suitability of in situ materials for earthworks construction, and to provide design parameters for foundation and earthworks design, geotechnical investigation were carried out over four months in 2021 in accordance with the locations outlined in Table 18-1 (KP, 2022a). The scope of work comprised:

- drilling of 25 boreholes using diamond coring techniques, with in situ standard penetration tests (SPT);
- installation of 10 groundwater monitoring standpipes;
- undertaking falling head permeability testing in selected boreholes;
- machine excavation of 118 test pits; and
- sampling of soil and rock for laboratory testing.

Table 18-1: Geotechnical Investigation Summary (KP, 2022a).

Infrastructure	Boreholes	Test Pits
TSF	4	39
Process plant	11*1	19
WSD	2	7
WHD	1	10
Waste dumps*2	2	10
Airstrip	N/A	14
Construction materials	3	19

Table 18-1 notes:

- *1 Including 2 No. boreholes that collapsed,
- *2 Waste dump locations moved subsequent to investigation.

18.2.2.2 Geotechnical Results

Overview

The ground conditions encountered at the site comprise fine and coarse grained alluvial and colluvial material to a depth of approximately 1 m below ground level (bgl), which is low strength in places, with fines often having high plasticity. This is underlain by high plasticity residual clay, which is typically stiff to very stiff. The upper layer can be lower strength and firm, transitioning to extremely weathered material. Igneous bedrock is present at depths of approximately 10 mbgl. Rock outcrops are present in places.

The upper approximately 3 m of ground tends to be more variable and can be low strength and more compressible.

Groundwater was typically observed in boreholes at greater than 15 mbgl on higher ground, and close to surface in the base of the valleys.

Earthworks structures (stability design) will need to consider the undrained stability of the embankments and particularly, the lower strength near surface material and the extent to which this needs to be removed. Areas of poor ground can be expected in the valley floor and some material will need to be removed during embankment construction.

The laboratory test results indicate that the site materials are potentially dispersive. Earth dams constructed from dispersive soils are at greater risk of internal erosion and piping, and the design of the tailings storage facility will need to take this into account.

The materials are considered suitable for the construction of embankments, providing the design incorporates measures to mitigate against the dispersive nature of the soils.

Interpretations and conclusions, and recommendations for the various geographical areas following, are presented in Section 25 and 26, respectively.

Tailings Storage Facility

Ground conditions and artisanal mining activity at the TSF area are summarised in Table 18-2. Artisanal mining activities were observed within the TSF basin, but not within the area of the proposed embankment. Said activities have resulted in disturbed ground, perched water, and excavations.

Table 18-2: TSF Typical Soil Profile

Depth (mbgl)	Description
~ 0.25	Topsoil
~ 0.5 to 4	Variably loose and medium dense or firm and stiff alluvium and colluvium to an average depth of approximately 1 mbgl, with a greater thickness present in places (up to 4 mbgl) localised soft areas (e.g. TP-TSF-02 close to a stream course and soft to 0.5 mbgl). The fine-grained alluvium and colluvium typically comprised high plasticity clay, trace sand and gravel.
~4	Typically stiff and very stiff residual soil extending to approximately 4 mbgl (firm in places) becoming extremely weathered and very stiff to hard. The residual and extremely weathered material typically comprised high plasticity clay, trace sand and gravel.
+9.4	Igneous rock which tended to be moderately or highly fractured. The surface of rock was encountered at between 9.4 to 13.8 mbgl, with low strength rock extending to a depth of up to 21 mbgl.

Water Storage Dam

Ground conditions at the water storage dam (WSD) area are summarised in Table 18-3. Both the transported and residual soils tended to be sandier, and have a lower fines content and plasticity than at the TSF. The soils tended to be on the boundary between fine and coarse grained.

Table 18-3: WSD Typical Soil Profile

Depth (mbgl)	Description
~ 0.20	Topsoil
~ 1.3 to 4	Loose to medium dense granular alluvium/colluvium. Both boreholes indicated very loose to loose, and soft to firm material at the base of the valley. Residual high plasticity clay or extremely weathered rock of a maximum of a few metres in thickness was present below the transported soil.
+4	Rockhead was at a comparatively shallow depth. The test pits encountered rock at a depth of between (0.7 and 3.2) mbgl, and the boreholes at a depth of approximately 4 mbgl. The rock comprised low and moderately fractured distinctly to slightly weathered medium and high strength igneous rock.

Water Harvest Dam

Ground conditions and artisanal mining activity at the water harvest dam (WHD) area are summarised in Table 18-4. Little or no mining activity was observed within footprint of WHD.

Table 18-4: WHD Typical Soil Profile

Depth (mbgl)	Description
~ 0.25	Topsoil
< 2.5	Alluvium in the valley floor of variable thickness, soft in places.
< 6.5	Stiff and very stiff high plasticity residual clay, trace sand and gravel.
+6.5	Rockhead was identified in the borehole at 6.5 mbgl at the base of the valley, and shallower in some of the test pit located at higher elevations. The rock comprised moderately to highly fractured gneiss which was initially low to medium strength becoming medium to high strength from a depth of approximately 10 mbgl.

Plant

Ground conditions and artisanal activities at the proposed Plant area are summarised in Table 18-5, with additional findings below:

Table 18-5: Plant Site Typical Soil Profile

Depth (mbgl)	Description
~ 0.25	Topsoil
< 1.0	Loose to medium dense coarse grained granular alluvium or colluvium and occasionally, firm to stiff high plasticity clay trace gravel and sand to typically to a depth of 1 mbgl. Soft to firm material was identified in one test pit.
< 4.0	Firm and stiff high plasticity residual clay trace gravel and sand, where it transitioned to very stiff extremely weathered material.
~ 4.8 to 12.0	Igneous rock was encountered at variable depths.

- No artisanal mining was observed within the proposed plant site; however, artisanal mining activity was noted close by.
- Groundwater monitoring indicated groundwater to be at a depth of approximately 15 mbgl. The process plant is located on the crest of a rounded hill and no notable stream course run through the area.
- Chemical testing indicates that the soils are non-aggressive towards concrete.

A settlement analysis has been undertaken to calculate settlements of the major process plant foundations under a number of ground improvement scenarios (KP, 2022a). Key outcomes of the settlement analyses are summarised as follows:

- The ground conditions comprise an upper layer of transported and residual soil that extends to a maximum depth of 3 mbgl, and can comprise more compressible material. Underlying this is more competent residual soil that becomes more competent and less weathered with depth. Rockhead is variable and modelled at a depth of 11 mbgl.
- The Plant platform will be predominantly located in cut, with much of the more compressible surface soil being removed.
- The calculated values of settlement indicated are generally lower than the allowable settlement values, with the exception of total settlement for the main stockpile and differential settlement for the High-Pressure Grinding Rolls (HGPR). The settlement of a number of other structures are borderline. These structures require more detailed consideration in the Front-End Engineering Design (FEED) Phase.

Airstrip

The ground conditions at the airstrip were notably different to other parts of the site, with ferricrete and lateritic gravel commonly encountered. Where this material was encountered, test pits were generally terminated due to refusal in the material as it became more cemented. The ground was frequently a gravel, clayey with sand with the fines being intermediate and high plasticity. The gravel was generally to predominantly rounded, and fine and medium grained.

Construction Materials

The presence/suitability of borrow material on or around the Site, are summarised below:

- The TSF is to be provided with an HDPE low permeability liner which requires underneath, a smooth subgrade or lower permeability material underneath. The WSD and WHD embankments require low permeability material for their construction. The residual soil typically comprises intermediate or high plasticity clay, and selected material is considered to be suitable and a key source of borrow material. Much of the alluvium and colluvium also possesses a sufficient level of fines and plasticity, and much of this material is also expected to be suitable. As such, there are considered to be suitable deposits of low permeability fill (Zone A) around the site.
- It is expected that Zone C will be sourced from mine waste during the operation. Prior to mining, excavated in situ material from within the dam basin areas is considered suitable for use as Zone C fill.
- It is expected that Zone E and Zone G rockfill, will be sourced from mine waste or from an onsite quarry. A limited amount may be obtained from rock outcrops within the dam basins.
- Laboratory test results indicate that two local sand samples could be considered for use as drainage sand on a case-by-case basis.
- Selected ferricrete, laterite and gravel colluvium are expected to be suitable for sub-base and basecourse for unsealed roads, and structural fill.

Aggregate sources and their suitability are discussed below:

- Existing Quarries

  Laboratory testing of processed rock from two quarries (Sogecar and Caderac) close to the town of Bouake in CI was completed to assess the suitability for use as concrete aggregate for the Project. The two quarries are located approximately 11 km apart, and both comprise micro-granite. Test results and petrographic examinations indicate the materials to be very similar.

  The materials met the specification requirements (Australian Standards) for coarse aggregate with the exception of the Los Angeles (L.A.) abrasion test, which was borderline. The borderline test results do not discount the use of the materials, and it is recommended that strength testing of concrete mixes is undertaken to confirm that concrete mixes using the coarse aggregates meet the required strength specifications.

  The use of either source for the supply of fine aggregates is not recommended without further specialist advice due to the high mica content (approximately 6%). An alternative would be to mix this with another source, to reduce the proportion of mica to less than 2%.

- Fine Aggregate from Alluvial Sand

  Three locations of alluvial sand located close to the site were sampled. Koundougou and Delisso are located approximately 16 km from the site and Boboso, 10 km from the site. Preliminary fine aggregate testing was undertaken on the samples comprising a limited suite of laboratory tests.

  All of the samples comprise predominantly fine and medium sand which meets the requirements of AS2758.1, which permits 100% of material passing 0.6 mm.

  All samples exceeded the specification requirement of ≤ 5% fines for naturally occurring sand. The Koundougou sand has the lowest level of fines with 7%. The Delisso sample exceeded the specification requirement of ≤ 1% clay particles for naturally occurring sand; however, based on photographs of the site it is expected that the sand may possess moderately elevated organic content.

  Though none of the material meet the required level of fines and may possess elevated organic content, the Koundougou sand was closest to meeting requirements. It may be possible to wash the sand to lower the level of the fines and organic content. More detailed testing of the Koundougou sand should be undertaken to confirm suitability.

18.2.3 Seismicity

A seismic hazard assessment was completed for the Site (KP, 2021a). CI is located on the African tectonic plate, approximately 800 km northeast of the Mid-Atlantic Ridge, the closest plate boundary.

The Site is located within the West African Craton, which is one of five large masses, or cratons, of Precambrian basement rock that make up the African Plate. These land masses came together in the late Precambrian and early Palaeozoic eras to form the African continent.

For the Project it is recommended that:

- The 1000-year ARI earthquake is adopted for the operating basis earthquake (OBE), for a High B consequence category TSF. The estimated peak ground acceleration (PGA) is for a 1000-year ARI earthquake of 0.026 g (0.255 m/s^2) for site Class D. A design earthquake of magnitude M6.7 located at a distance of approximately 250 km has been selected for the 1000-year ARI OBE.

- The 5000-year ARI peak ground acceleration is adopted for the Safety Evaluation Earthquake (SEE) for a High B consequence category. The estimated PGA for the 5000-year earthquake was calculated as 0.034 g (0.33 m/s^2).
- West Africa is a region of relatively low seismic activity, making estimation of a realistic MCE difficult. As such, the 85% fractile PGA of 0.085 g (0.83 m/s^2) was selected as the MCE based on site Class D, corresponding to a M6.9 shallow crustal earthquake at a distance of 190 km from the site.

Parameters have also been provided for the seismic design of structures at the site in accordance with recommendations provided in the International Building Code (International Code Council, 2015). Based on the site investigation, the site conditions conform to the Site Class D defined by ASCE/SEI 7-10 as stiff soil with standard penetration test (SPT) values in the range between 15 to 50 in the top 30 m. The seismic parameters for structural design are as follows:

- Seismic coefficient, SS = 0.041 g or (0.04 m/s^2).
- Seismic coefficient, S1 = 0.110 g or (1.08 m/s^2).
- Peak ground acceleration (PGA) = 0.051 g or (0.50 m/s^2).

18.2.4 General Site Earthworks and Terracing

General Project site infrastructure earthworks and terracing development activities include:

- clearing and grubbing;
- top-soil removal, stockpiling and management (Section 18.2.8);
- earthworks cutting and filling to design levels; and
- importing suitable fill materials from borrow locations and exporting waste and surplus materials to stockpiles.

The estimated quantities for general site infrastructure bulk earthworks required for the Plant, MSA and camps is summarised in Table 18-6, following.

Table 18-6: Summary of Bulk Earthworks Quantities

Area	Cut (m^3)	Fill (m^3)	Balance (m^3)	Import/Export
Process Plant	623 167	204 207	418 960	export
MSA	126 764	126 512	252	export
Gendarmes Barracks & Main Gatehouse	25 262	1429	23 834	export
Permanent Accommodation Camp	89 590	107 224	-17 634	import

No specialist earthworks, foundations or ground improvement works (such as piling, ground anchors, grout injection, etc.) are proposed for the development of the Project infrastructure.

18.2.5 Plant and MSA Earthworks

The Plant, as illustrated in Figure 18-2 and Figure 18-3, is located northwest of the mine pit where the topography consists of a central ridge line along the spine of the Plant, which runs from the ROM pad through to the administration and warehousing area. The preliminary bulk earthworks design for the DFS consists of a single terrace pad that is predominantly in-cut below the natural ground level to allow for process plant foundations to be constructed on more competent ground material, thereby minimising the requirement for ground improvement works (Figure 18-5, following). The intent is for cut material to be used locally in fill where suitable for earthworks construction. Additionally, some allowances have been made for localised ground improvement works below concrete foundations involving the replacement of near surface ground material with select structural fill where required. The finished design levels of the earthworks pad/terrace will be graded for drainage purposes.

The MSA facilities are located adjacent to and southwest of the process plant (Figure 18-2). A balanced cut to fill has been allowed for constructing the MSA earthworks terrace as shown Figure 18-6. The finished design levels of the single terrace earthworks pad will be graded for drainage purposes.

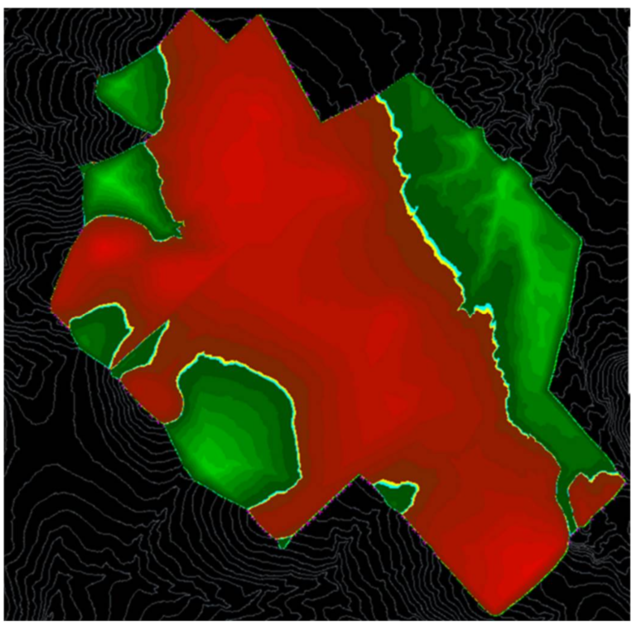


Figure 18-5: Plan of Plant Prelim. Bulk Earthworks Design

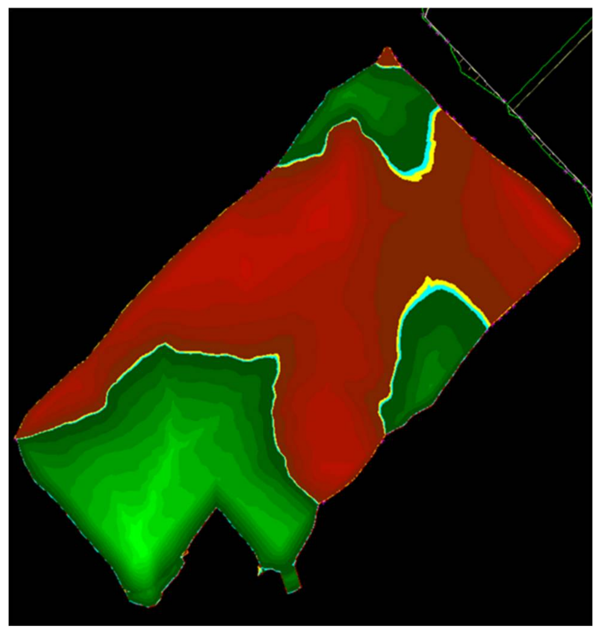


Figure 18-6: Plan of MSA Prelim. Bulk Earthworks Design

Figure 18-5 and Figure 18-6 legend: red = cut, green = fill. Illustrative purposes only, no scale or orientation provided

18.2.6 Permanent Camp Earthworks

The Permanent Accommodation Camp, as indicated in Figure 18-2, is located east of the mine and positioned on the side of a hill, with terraced bulk earthworks. Terracing will be constructed using earthworks embankment battered slopes (no retaining walls) to transition between terrace levels (Figure 18-7).

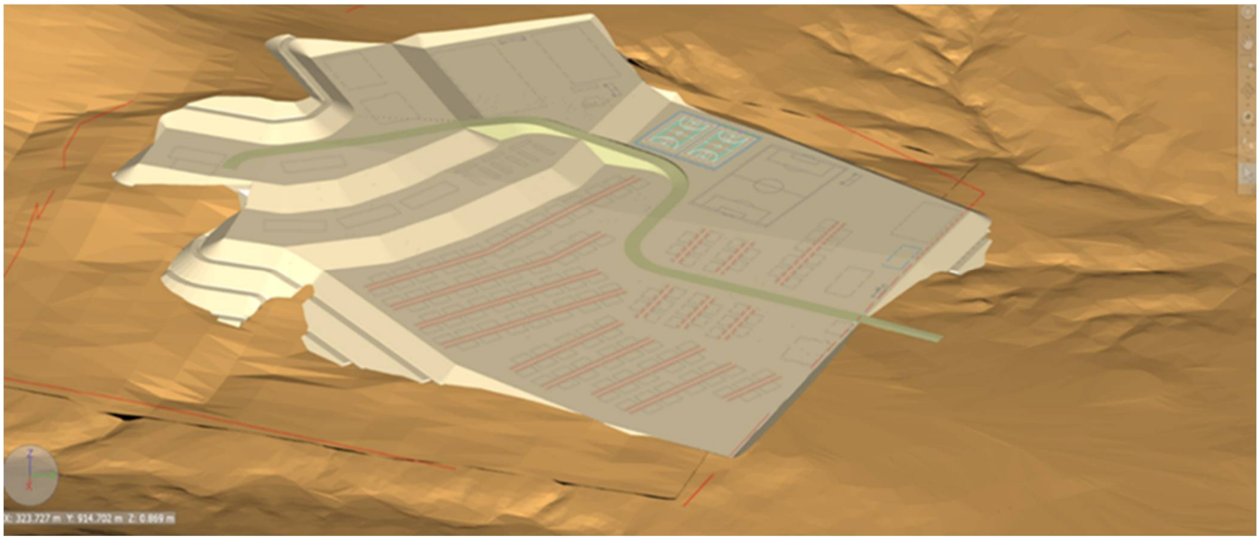


Figure 18-7: 33D Isometric of Permanent Camp with Bulk Earthworks Terraces

18.2.7 Gendarmes Barracks and Main Gatehouse Earthworks

The Gendarmes Barracks and Main Gatehouse area is located at the northern end of PE 58 intersecting the public access road entrance (Figure 18-2). The DFS preliminary bulk earthworks design has allowed for a terraced earthworks pad cut into a hill on the southern side, as shown in Figure 18-8, following.

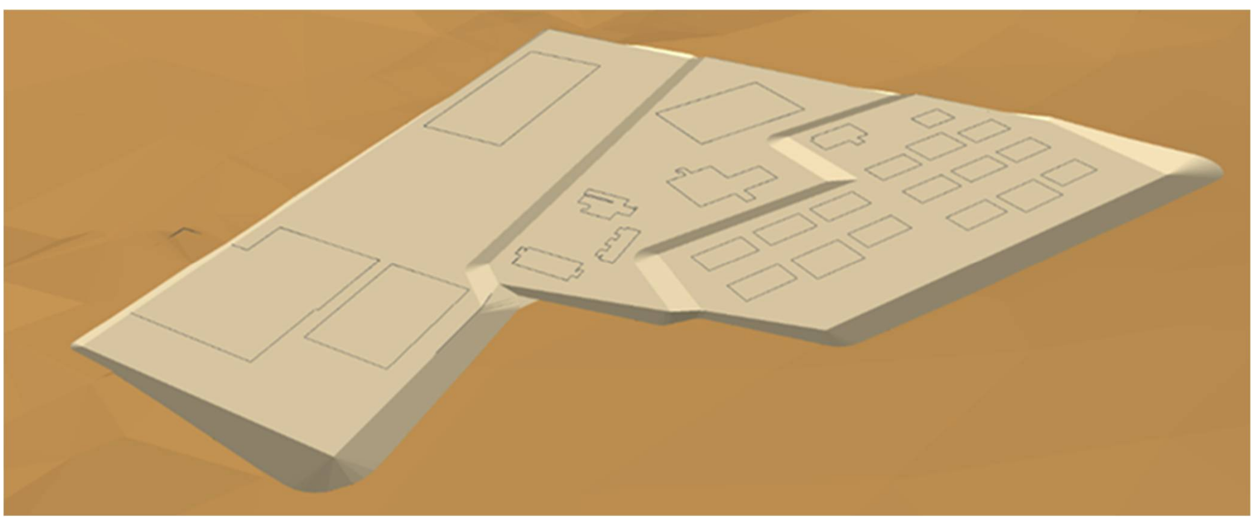


Figure 18-8: 3D Isometric of Gendarmes Barracks Bulk Earthworks Terraces (Lycopoidum)

18.2.8 Topsoil Management

Topsoil will be removed and stored in accordance with Endeavour/Lafigué Mine Procedures (Endeavour, 2022b) (Endeavour, 2022c). Approximate volumes of topsoil to be managed are summarised by area in Table 18-7.

Topsoil stockpiles shall not exceed 2 m in height and shall be flat topped to encourage establishment of vegetation. Topsoil will be placed in the stockpiles in such a way as to minimise compaction and loss of structure. The upper surfaces of the stockpiles shall be ripped on completion to facilitate seeding.

The side slopes of stockpiles shall not be steeper than 4H:1V (1 vertical in 4 horizontal). Surface drainage of the stockpiles shall be managed to reduce loss of material through erosion. Stockpiles shall not impede the drainage from upstream catchment areas.

Table 18-7: Topsoil Management Volumes by Affected Area (Endeavour, 2022d)[85]

Area	Topsoil Removal Area (ha)	Approximate Volume (m^3)
Roads	36.0	72 000
Airstrip	11.0	21 900
Camp	14.2	28 409
Water Storage Dam	15.1	30 200
Water Harvest Dam	66.6	133 100
Process Plant	38.0	76 000
Pit	**159.4**	**318 765**
• Main Pit - Pushback 4	126	252 918
• Pit B -Pushback 3	31	62 690
• Pit C - Pushback 1	2	3157
Waste Rock Dumps & RoM Pad	**411.5**	**822 904**
• RoM Pad and Western Dump	72.7	145 395
• South and Eastern Dump	338.8	677 508
Sediment control Structures	3.9	9750
Tailing Storage Facility	253.3	633 300
Totals	**1008.9**	**2 146 328**
Area Required ≤2 m high Stockpile	**107.3**	

18.2.9 Surface Water Management and Sediment Control

Refer to Section 18.5 for details on surface water management and sediment control infrastructure/earthwork requirements.

18.2.10 Demolition

The Project is a greenfields development free of existing facilities and no demolition work will be required.

18.2.11 Landscaping/Seeding

Landscaping and seeding is to be implemented as part of operational phase activities. No costs have been allowed for this activity in the CAPEX estimate.

[85] Average assumed topsoil thickness 0.21 m

18.3 Transport and Logistics

18.3.1 Off Site Transport Infrastructure

Off-site logistics infrastructure that is required to support the construction and operation of the mine, including: Ports; Airports; fuel pipelines, rail networks, and public roads is described fully in Section 5, and not repeated herein.

18.3.2 Site Transport Infrastructure

18.3.2.1 Roads/Access

For the transport of goods, materials and people, the mine will use existing all season paved public roads through to the town of Koundoudougo. An upgrade of the existing public road/track from Koundoudougou off the B412 to Site, is currently being executed as early works during the DFS phase. This upgraded all-weather unsealed road will extend southeast from Koundoudougo for approximately 11 km, before turning due south for a further 4 km towards the village of Lafigué. The Lafigué village will then be bypassed with the construction of a new 2 km all-weather unsealed access road to the main access gate at the Site. Outside of the main access gate, an area for parking and truck staging will be provided. The proposed road routing is illustrated in Section 5.

18.3.2.2 Site Access Roads

Site access roads will be provided as noted below:

- Main site entrance gatehouse to the Plant (2.4 km).
- Turn-off from the Plant access road to the Permanent Accommodation Camp (4.3 km).
- Turn-off from the Permanent Accommodation Camp access road to Airstrip (2.4 km).
- Plant to TSF, following the decant pipeline to the decant towers and tails pipeline along the eastern boundary of the TSF to facilitate maintenance and monitoring (4.3 km length TSF Stage 1). From here, access will be via the embankment wall.

The design basis for the 15.6 km of Site based roads is presented in Table 18-8, following.

Table 18-8: Site Access Road Design Parameters (KP, 2022b)

Road Cross Section	Formation Width: 9 m Lane Width: 3.5 m, with 1 m shoulder each side of road Safety Bunds: 0.5 m high, where fill height > 2 m Crossfall: 2%
Minimum Horizontal Curve Radius	30 m
Minimum Vertical Curve Length	25 m
Maximum Vertical Grade	8%
Minimum Culvert Diameter	600 mm
Culvert Design Criteria	20-year ARI
Pavement	150 mm laterite gravel wearing course

With respect to Table 18-8, the following may be noted (KP, 2022b):

- The road vertical alignments were designed to balance cut to fill as far as practicable, over the entire road length. The vertical alignment design includes an allowance for fill build-up around stream crossings to ensure correct operation of crossings.
- Drainage ditches, turnouts and level spreaders will be constructed along the site access road alignments as required.
- Culvert crossings were designed at significant stream crossing locations along the site access road alignments to convey all runoff resulting from a 20-year average recurrence interval (ARI) storm event (duration equal to time of concentration). The culverts will comprise corrugated steel pipe (CSP) culverts.

18.3.2.3 Minor Access Roads and Tracks

Minor roads and tracks have been provided for the Project/Site, to facilitate operations, maintenance and security management. These include:

- An access track from the Dabakala substation to the Site substation (28 km) has been provided to facilitate maintenance and monitoring of the transmission power lines.
- An access/maintenance track (9.5 km) from the process plant to the WHD. The WHD access track will run alongside the WHD-WSD abstraction pipeline. The WHD access track comprises a 6 m wide running surface. The road crossfall will vary along the road alignment to suit drainage requirements. A 150 mm laterite wearing course will be placed over the subgrade/general fill. An allowance has been made along the WHD access track for culvert crossings at stream locations.
- An access track off the WHD access track to the explosives magazine/emulsions plant (length of spur 0.6 km).
- A security access track on both sides of the site perimeter fence (total length: 24 km on either side).

18.3.2.4 Mine Haul Roads

Mine haul roads (6.6 km) connect the open pits, waste dumps, TSF embankment (for construction) and mine services area. Certain haul roads will be constructed progressively in line with the staged pit development as described in Section 16.

Site haul roads were designed to facilitate the following haulage movements during operation:

- Pits to waste dump(s).
- Pits to ROM pad.
- Pits to MSA.
- Pits to TSF embankment for construction.

Haul road design parameters are illustrated in Table 18-9, following.

Table 18-9: Haul Road Design Parameters (KP, 2022b)

Road Cross Section	Running Width: 30 m Lane Width: 12 m Safety Bunds: 1.5 m height both sides of road Crossfall: 2%
Minimum Horizontal Curve Radius	100 m
Minimum Vertical Curve Length	25 m
Maximum Vertical Grade	8%
Minimum Culvert Diameter	600 mm
Culvert Design Criteria	20-year ARI
Pavement	200 mm laterite gravel wearing course (to be upgraded by the mining fleet when competent rock available)

With respect to Table 18-9 the following may be noted (KP, 2022b):

- Road vertical alignments have been designed to balance cut to fill as far as practicable over the entire road length. The vertical alignment design includes an allowance for fill build-up around stream crossings to ensure correct drainage at crossings.
- The laterite wearing course, will be upgraded during operation when competent rock is available.
- Drainage ditches, turnouts and level spreaders will be constructed along the haul road alignments as required.
- Culvert crossings have been designed at significant stream crossing locations along the haul road alignments to convey all runoff resulting from a 20-year ARI storm event (duration equal to time of concentration). The culverts will comprise of CSP structures.

18.3.2.5 Airstrip

As illustrated in Figure 18-2 (Point 2) and Figure 18-9, following, a non-instrument/visual flight rules (VFR) airstrip is to be provided on Site for construction and operations. In normal operations it will be used twice weekly and monthly, for the transport of people and gold respectively. Given the distance from Abidjan to Site, no refuelling facilities will be provided at Site.[86]

The airstrip design parameters and the flight path Obstacle Limitation Surface (OLS)[87] assessment are presented in Table 18-10 and Figure 18-9, respectively. No issues were identified in the OLS assessment.

[86] Endeavour's Ity Mine has hangers and refuelling facilities for its CI aircraft.

[87] To comply with RACI 6001 regulations**Invalid source specified.**, an OLS must be determined for the aerodrome.

Table 18-10: Airstrip Design Parameters (KP, 2022b)

Design Criteria	Design Parameter
Design Aircraft	Pilatus PC-12
Aircraft Classification	2B
Runway Length	1 060 m
Runway Width	23 m
Runway Longitudinal Slope	≤ 2.0%
Longitudinal Slope Change Between Segments	≤ 2.0%
Longitudinal Slope Change	≤ 0.4% per 30 m
Longitudinal Slope Radius of Curvature	≥7500 m
Runway Traverse Slope	2.0%
Runway Strip Length	60 m
Runway Strip Width	80 m
Runway Strip Traverse Slope	1.0%
Taxiway Width	10.5 m
Taxiway Strip Width	40 m
Taxiway Longitudinal Slope	1.0%
Taxiway Transverse Slope	0.5%
Wheel Distance to Runway Edge (taxiing)	2.25 m
Taxiway Turning Radius (≤ 20 km/h)	24.0 m

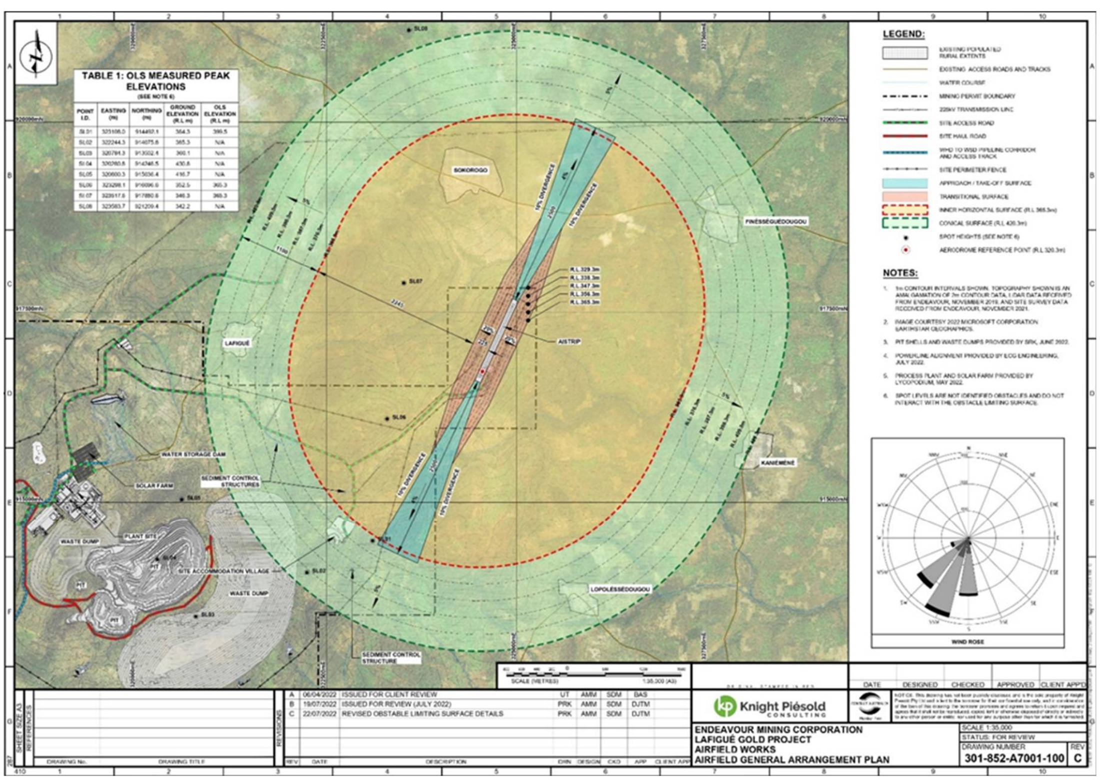


Figure 18-9: Obstacle Limiting Surface (KP, 2022b)

With respect to Table 18-10 and Figure 18-9, the following may be noted.

- The prevailing wind direction is south-southwest (SSW) to north-northeast (NNE).[88] The proposed airstrip alignment runs parallel to the prevailing wind direction. Site-specific data for wind velocities and prevailing wind directions is not currently available (see also Section 5). The airstrip alignment corresponds with other airstrips in proximity to Site (including Bouake).
- The cross-fall on the runway is 2.0% from the crown; however, it is decreased to 1.0% on the runway strip (external to the main runway) with the intent to reduce the required overall earthworks volumes.
- The longitudinal profile of the current natural topography exceeds the minimum longitudinal slope of 2% in multiple locations along the runway alignment.
- The maximum longitudinal runway slope along the runway is 2.0%, thus complying with (ICAO, 2006) and RACI (ANAC, 2020) guidelines. The average longitudinal runway slope is 0.7%.
- The apron is 100 m by 50 m and will comfortably park 2 design aircraft.
- A terminal building will be provided, comprising a prefabricated steel building 12 m x 6 m with toilet facility.
- The airstrip perimeter will be fenced.
- The aircraft trafficked pavement design (runway, taxiway and apron) comprises two pavement layers constructed over a prepared subgrade. In order of foundation to surface, these can be summarised as follows:
  - Subgrade: depth of 200 mm, compacted to achieve a CBR of at least 15%.
  - Sub-base course and wearing course: 150 mm thickness, compacted to achieve a CBR of at least 35%.
- The pavement depths and specified minimum CBR values for each pavement layer listed above have been based on the design aircraft (2 No. flights per day) for a nominal operating life of 20 years.
- The compatibility of this pavement design with the in-situ foundation conditions and available construction materials is discussed in Section 18.2.2.
- It is noted that the in-situ gravel material may achieve a CBR value of 40%. Typically, a CBR value >80% would be recommended for airstrip pavements but is subject to material availability and the acceptance of additional maintenance requirements. In this instance, the insitu materials were designated suitable for use as a pavement accepting that the pavement may degrade and require increased and more frequent maintenance.
- External to the trafficked areas (i.e. the runway, taxiways and apron), the in situ material will be graded to line and level, and compacted to achieve a CBR value of >15%.
- The drainage comprises surface water drains either side of the airstrip.
- The following items were designed to replicate existing, ANAC-permitted airstrips on existing Endeavour operations: wind direction indicators; runway markers and taxiway and apron markers.

Formal routine maintenance of the runway should be performed at least quarterly, more frequently if the pavement shows signs of degradation, deformation, rutting or erosion. Routine maintenance will comprise grading, watering and compaction. Occasional re-sheeting of the wearing course layer will be required during the airstrip operational life. The frequency of maintenance shall be adjusted according to in-situ conditions and performance of the runway and airstrip pavement.

[88] Data collected from the Bouake Weather Station, located approximately 80 km southwest from the Site was used as a design input.

18.3.3 Construction and Operation Logistics

18.3.3.1 Construction Logistics

Construction materials and equipment will be typically transported to site by truck using the public road network. Inbound construction materials and equipment sourced from overseas will be generally shipped in country via the Autonomous Port of Abidjan (APA)[89].

Based on a transport route survey completed during the DFS (MOVIS CI, 2021), the transport of large loads from the APA to PE 58 will be typically limited to a maximum weight of 320 t, width of 6 m and height of 4 m; however alternative transport routes exist for larger oversized loads if required. A comprehensive transport route survey will be undertaken during the early stages of the Project execution phase, as part of the transport and logistics contract.

The estimated quantity of sea containers and break-bulk cargo to be transported to site for the construction of the Plant is summarised in Table 18-11, following.

Table 18-11: Summary of Transport Cargo for Plant Construction

Transport Category	Unit	Estimated Quantity
20 ft containers	No.	85
40 ft containers	No.	582
Break Bulk Cargo	m^3	4615

The heaviest and largest break bulk cargo to be transported to site for the construction of the Plant is summarised in Table 18-12, following.

Table 18-12: Summary of Heaviest/Largest Break Bulk Cargo for Plant Construction

Break Bulk Item Description	Unit Weight (t)	Length (m)	Width (m)	Height (m)	Unit Volume (m^3)
Mill Heads	**48.0**	6.80	6.80	1.80	83
Mill Shells	43.3	6.80	4.57	**6.80**	211
Crusher Frame	42.1	4.80	2.40	3.26	38
HPGR	35.5	4.20	3.50	3.20	47
HPGR Base Frame	31.7	6.50	2.55	2.78	46
Apron Feeder	30.7	14.00	2.65	2.40	89
Pre-Leach HRT Radial beam Sector	9.3	**14.90**	2.10	2.60	81
Platework - Bulk (Item 19)	8.3	11.58	4.78	4.50	**249**

Local construction labour will be transported from nearby villages to and from Site, using buses on the public road network. Non-local construction personnel can currently travel to site from Abidjan by vehicle on the public road network. When the Site airstrip is operational during the early stages of construction, expatriate and senior personnel resident in or transiting through Abidjan, will be flown to and from Site.

[89] Discussed more fully in Section 5.

18.3.3.2 Operational Logistics

Maintenance and operational supplies will be typically transported to site by truck using the public road network similar to the transport logistics basis for construction. Inbound goods and equipment from overseas suppliers will typically arrive in CI via the APA (Table 18-13, following). From this table, it can be seen that transport volumes are low, and no logistics/materials handling constraints are foreseen.

Local operations and maintenance personnel will be transported from nearby villages to and from Site, using the public road network. Expat and non-local personnel will be flown to and from Site via Abidjan (two scheduled flights per week).

Gold produced at the Mine will be transported off Site by plane twice monthly.

Mine haulage will be limited to transport on Site within the main perimeter fence line and will not involve any interface with public access roads.

Table 18-13: Operational Logistics Requirements (Endeavour, 2022a)

Commodity	Weighted Average (t/a)	Maximum (t/a)	Trucks/mo. (max)
Diesel	20 561	28 775	56
Explosives (ANNP and Emulsion)	10 293	14 918	52
Steel Balls	2451	2582	9
Lime $Ca(OH)_2$	1005	1058	3.7
Cyanide (NaCN)	840	881	3.1
Hydrochloric acid (HCL)	657	677	2.4
Flocculant	238	251	0.8
Caustic (NaOH)	231	238	0.8
Carbon	156	164	0.6
Total			**128**

Table 18-13 notes:

- With the exception of diesel, it was assumed that the net payload per truck is 24 t.
- It has been assumed that the diesel truck payload will be 42.5 t (50 m^3 tanker) (to be confirmed in FEED phase), alternatively it could be 30 t.
- Approximately six trucks are expected per day at the Mine, assuming delivery five days a week (most are likely to arrive during day shift).

18.3.3.3 Mobile Equipment for Plant & Administration

The mobile equipment provided for Plant and general Site non-mining functions are summarised in Table 18-14 following.

Table 18-14: Summary of Mobile Equipment for Plant and Administration Functions

Mobile Equipment Description	Number of Units
4WD Light Vehicle	25
8 t 4WD Flatbed Truck with Hiab	2
Forklifts	2
Bobcat	1
Integrated Tool Carrier/Forklift	1

Table 18-14: Summary of Mobile Equipment for Plant and Administration Functions

Mobile Equipment Description	Number of Units
4 t Telescopic Handler	2
80 t All Terrain Crane	1
200 t Hydraulic Crane	1
Backhoe/FEL (Cat 988)	1
Manlift/Scissor Lift	1
30 Seater Minibus	3
Fire Truck	1
Ambulance	1

18.4 Power

18.4.1 Country Overview

Power supply in CI is discussed in Section 5 and not repeated herein.

18.4.2 Off-Site Infrastructure and Power Required.

The Project will tie into a new 225 kV switchyard in Dabakala, and a new 33 km, 225 kV transmission line will be installed from Dabakala to PE 58. The new Dabakala switchyard is on a new 225 kV ring main that forms part of the main link from Yamoussoukro to Ferkessédougou (ECG Engineering, 2020).

Power quality on the CI 225 kV transmission network is good, and power availability should be in excess of 98%. Whilst low rainfall/dam levels and other factors led to 'load shedding/'power rationing' in-country in 2021, heavy industry are likely the last to be load shed (ECG Engineering, 2020).

The Lafigué Mine is estimated to have an installed load of approximately 26 MW_e with a maximum demand of 19 MW_e, and an expected grid energy consumption of 148 GWh_e/a (Endeavour, 2022a).

The Project scope of work for the 225 kV power supply to the Site includes the modifications necessary in the electricity grid network and the infrastructure required at the Site, with the following battery limits:

- CI-ENERGIES supply system 225 kV bus at Dabakala Substation.
- Outgoing 11 kV feeder in the CIE Lafigué Substation.
- Water connection point at the Lafigué Substation.
- Low voltage power connection point at the Lafigué Substation.

In summary, the Project involves the extension of the Dabakala Substation by extending the existing 225 kV bus, adding a 225 kV transmission line feeder bay, construction of 33 km of 225 kV single circuit lattice tower transmission line, and constructing a substation at Site. The Site substation will be owned and operated by CIE and the Project will take a 225 kV tariff metered feeder, install a 225/11 kV transformer in their substation, and take an 11 kV feeder to the plant main 11 kV switchboard.

Currently, the full Project cost for the transmission line is borne by the mine developer, with no recovery of costs in the form of a reduced tariff over a defined period. The commercial terms for the supply of infrastructure/power are still to be negotiated.

The point of supply and point of change of ownership will be the primary 225 kV terminals off the 225/11 kV step down transformer at the Lafigué substation within the Site. This will be the point of the tariff metering.

18.4.3 Site Power Demand, Generation and Power Management

Site operational power requirements by area are summarised in Table 18-15.

Table 18-15: Summary of Primary Power Demand by Area

Area	Power Source	Connected Load (kWe)	Max Demand (kWe)	Average Power Consumed (MWh/a)
Plant & Admin	Distribution	23 483	17 699	137 285
MSA	Distribution	511	409	3582
TSF Decant	Distribution	292	122	972
WSD	Distribution	236	94	8
Exploration Camp	Distribution	100	80	701
Gendarmes Barracks & Main Gatehouse	Distribution	190	135	906
Dog Kennels	Distribution	50	40	350
Permanent Accommodation Camp	Distribution	660	457	4000
Airstrip	Genset	15	11	5
WHD	Genset	810	389	973
Total Power		**26 347**	**19 436**	**148 782**
Total Grid Based Power		**25 522**	**19 036**	**147 804**

18.4.4 Power Generation

18.4.4.1 Overview

Generation capacity on Site is limited to electrical loads not connected to the Site power distribution network (Section 18.4.4.2) and Emergency Power Generation (Section 18.4.4.4). Further, whilst solar generated power has been considered (Section 18.4.4.3), it has not been incorporated in the operational energy mix for the Site. In future, a Group wide solar strategy may be considered for CI.

18.4.4.2 Remote Power Generation

For techno-economic reasons, the following electrical loads will be met with local diesel power generation capacity, rather than from the Site's power distribution network:

- WHD pumping station.
- Remote Borefields.
- Mine pit dewatering.
- Explosives Storage and Emulsion Plant.
- Airstrip.
- Remote security control guardhouses.

18.4.4.3 Solar Power Generation

Whilst not forming part of the DFS, space has been allocated to the north of the process plant for any future solar hybrid photovoltaic facility (16 to 20 MWe capacity). A financial analysis was carried out for solar power generation as part of the DFS and this indicated that on-site solar power generation to offset some or all of the Site's primary power requirements is not economic for mine supply only, when compared to grid sourced power. However, on-site solar power generation may be reconsidered in the future, particularly a hybrid approach where solar power can supply both the mine and the grid.

Other opportunities exist for solar and battery power solutions to potentially be adopted for small power demand requirements at remote site facilities, supplemented with diesel power generation where required. These solar power opportunities will be considered further during the Project's detailed design phase.

18.4.4.4 Emergency Power Supply

To supply critical loads during planned and unplanned grid outages, an emergency backup power station using diesel generators will be provided for the Site.

Given the relatively low emergency power requirements for the Plant (1345 kW), high voltage generators units (>1000 V) would typically not be utilised, as they are usually more economical at ratings of over 1500 kVA (ECG Engineering, 2020). Having more than one diesel generator supply the total required power allows for more flexibility in operating when the demand is low, specifically saving on fuel costs as well as maintenance requirements.

Therefore, the plan for the Site is to utilise 850 kVA, 400 V diesel generators for emergency power. These units will provide approximately 544 kW of continuous power (603.6 kW and 680 kW of prime standby power, respectively). Consideration will be given to utilising the same generator units for the plant and camp, so as to standardise and minimise generator spares.

The emergency power station setup will be as follows:

- 3 x 850 kVA diesel generators at the Plant, providing 1632 kW of continuous emergency backup power; and 1 x 850 kVA spare generator that can provide 544 kW of continuous emergency backup power.
- Diesel day tank.
- Low voltage switchboard for the generators feeding a single step-up transformer.
- Single high voltage (11 kV) feed to the process plant.

Back up emergency power diesel generators and associated infrastructure will also be provided at the following facilities:

- Permanent Accommodation Camp (1 x 850 kVA diesel generator).
- Gendarmes Barracks and Main Gatehouse area (1 x 250 kVA diesel generator).

18.4.5 Power Distribution

An 11 kV overhead power line fed from the Plant main 11 kV switchboard, will distribute power to the following remote infrastructure loads: Gendarmes Barracks and Main Gatehouse area; Dog Kennels and Canine Caretakers' Accommodation; Permanent Accommodation Camp; Exploration Camp; Sewage Treatment Plant; MSA; Tailings Storage Facility.

Overhead transformers will be provided for each facility listed above. The power distribution at these locations also includes:

- Cabling from the power pole transformers to a main distribution board.
- Sub-feed power distribution from the main distribution board to a localised distribution board outside the buildings.

18.5 Water Management

18.5.1 Water Sources and Requirements

Site water requirements by source and by use were developed by Knight Piésold (KP) and are as noted in Table 18-16, following. Both ground and surface water are expected to be of a quality suitable for the intended use, with only minor treatment required. The closest perennial water source is 23 km from Site (the N'zi river) and thus, water will primarily be sourced from wet season surface water run-off, and to a lesser degree from ground water. This makes the sizing and the balancing of water between the WHD and WSD critical.

The basis for the sizing of these dams and the management of stormwater inflows to the TSF, is described in Sections 18.5.2 to 18.5.3, following.

Table 18-16: Site Water Sources and Users

Water Source/Use	Users	Minimum m^3/h	Nominal m^3/h	Maximum m^3/h	Nominal m^3/a
Ground Water					
• Potable Water	Camps, Plant, MSA		8.2		71 832
WHD/WSD		128.9	247.2	467.8	1 978 000
• Plant Make-up		106.9	210.3	430.8	1 682 000
• MSA		22.0	22.0	22.0	176 000
• Dust suppression		0	15.0	15.0	120 000
Pit Water					
• Dust suppression			TBC[90]		TBC
TSF Decant					
• Plant	Users	0	351.4	395.7	2 811 000

18.5.2 Rainfall and Evaporation Basis of Design

A summary of the climatic conditions for the area and the country is presented in Sections 5 and 20.

For Site infrastructure design, a baseline climate assessment for the Project was completed to determine design rainfall sequence and storm events (KP, 2021b).

[90] Source will be either the WHD or Pit.

For the Project, climatic data were sourced from the Dabakala weather station (25 km from Site, with 79 years of data)[91] via the Système d'Informations Environnementales sur les Resources en Eau et leur Modélisation (SIEREM)[92] database.

From this dataset (1922 to 2000), the following design information was derived:

- Depth/Duration/Frequency (DDF) curves for short-duration extreme rainfall events for a range of durations, with an Average Recurrence Interval (ARI) of two years up to the Probable Maximum Precipitation (PMP).
- Typical variability of monthly precipitation.
- Typical variability of annual precipitation.
- Extreme monthly precipitation (one year sequence) for 5 to 100 year ARI wet/dry precipitation).
- Extreme monthly precipitation (seven month wet season sequence) for 5 to 100 year ARI wet precipitation.

KP performed frequency analysis on 1-day and 3-day annual maxima from 79 years of data for the Dabakala daily dataset to estimate the statistical likelihood of experiencing extreme storms at Lafigué, shown in Table 18-17.

Table 18-17: Extreme 24-Hour and 72-Hour Design Precipitation (KP, 2021b)

Annual Recurrence Interval (ARI)	24-h Precipitation Depth (mm)	72-h Precipitation Depth (mm)
5	114	136
10	131	156
20	148	176
50	171	201
100	189	220
Largest Recorded	155	200

Table 18-17 note: largest recorded unfactored daily values for Dabakala:

- 24-h maximum occurred in 1937; and,
- 72-h maximum occurred in 1930.

KP utilised the 24-hour design precipitation information (Table 18-17) to derive DDF curves for short duration storms. The PMP storm event was estimated using the Dabakala daily precipitation dataset.

Table 18-18: Lafigué Gold Project Depth/Duration/Frequency Data (KP, 2021b)

Storm Duration	Precipitation Depth (mm) for given ARI (year) Storm							
	5	10	20	50	100	1000	10 000	PMP
5 min	10	11	13	15	17	23	29	47
10 min	18	21	23	27	30	41	52	85
15 min	25	28	32	37	41	56	71	116
30 min	39	45	51	58	65	88	112	184
1 h	56	64	72	83	92	126	160	262

[91] 89% of the data for this period was considered suitable for use.

[92] www.hydrosciences.fr

Table 18-18: Lafigué Gold Project Depth/Duration/Frequency Data (KP, 2021b)

Storm Duration	Precipitation Depth (mm) for given ARI (year) Storm							
	5	10	20	50	100	1000	10 000	PMP
2 h	72	83	93	108	119	163	207	339
3 h	80	92	104	120	133	182	232	379
6 h	93	107	121	140	154	211	268	439
12 h	104	120	135	156	172	236	300	491
18 h	110	126	143	165	182	250	317	519
24 h	114	131	148	171	189	2591	3291	538
72 h	136	156	176	201	220	3022	3832	6272

Table 18-18 notes:

- The 1000-yr and 10 000 year ARI depths are based on a logarithmic interpolation between the 24 hour 100-year ARI and the 24-hour PMP, assuming the PMP is equivalent to a 10 million-year ARI event.
- The 1000 year ARI, 10 000 year ARI and PMP 72 hour depths have been preliminarily estimated using the relationship of the 24 hour and 72 hour 100 year ARI depths.

Daily precipitation records (June 1922 to December 2000) from the Dabakala dataset were summed to produce annual totals for the 77 years of available record (two years were discarded due to missing data). KP performed frequency analysis on annual precipitation values. The rainfall pattern observed in one of the median years (1967), wettest year (1957) and the second-driest year (1997) were used for the average, wet and dry series respectively. These monthly ratios were then multiplied by the computed statistical annual values to form the required synthetic scenarios noted in Table 18-19.

Table 18-19: One-Year Duration Synthetic Precipitation Scenarios (KP, 2021b)

Month	Wet Scenarios (mm)			Average (mm)	Dry Scenarios (mm)		
	100-y ARI	50-y ARI	10-y ARI		10-y ARI	50-y ARI	100-y ARI
Jan	0	0	0	0	43	34	30
Feb	4	4	3	56	0	0	0
Mar	130	123	103	64	27	21	19
Apr	258	243	205	83	82	64	57
May	113	106	90	120	112	87	77
Jun	166	157	132	135	192	149	132
Jul	138	130	109	111	59	46	41
Aug	387	364	307	270	123	96	85
Sep	331	312	262	181	122	95	84
Oct	209	197	166	67	78	61	54
Nov	9	9	7	30	6	4	4
Dec	27	26	22	0	0	0	0
1-year Totals	**1772**	**1669**	**1406**	**1119**	**845**	**657**	**582**

Frequency analysis was also performed on the ANCOLD defined 'wet season'[93] (ANCOLD, 2019). For Dabakala, it was found that on average, 70% of the rainfall falls over approximately 210 days from April to October and as such, this was assumed to be the 'Wet Season'. Frequency analysis was performed on the maximum 210-day rainfall depths of this dataset, to derive different ARI events as noted in Table 18-20.

Table 18-20: Seven Monthly Wet Season Rainfall Sequences[94] (KP, 2021b)

Month	Wet Season Scenarios (mm)				
	100-yr ARI	50-yr ARI	20-yr ARI	10-yr ARI	5-yr ARI
Apr	278	261	239	222	204
May	110	104	95	88	81
Jun	161	152	139	129	119
Jul	160	150	137	128	117
Aug	290	272	249	232	213
Sep	393	369	338	314	289
Oct	177	167	153	142	131

After an extensive search to locate suitable data, KP was unable to find a suitable source of pan evaporation data near to the Site. Average monthly pan evaporation values were found for Zuenoula and Ferké, approximately 175 km to the southwest at an elevation of 202 mamsl and 165 km to the northwest of the site at an elevation of 340 mamsl, respectively. The values were used to estimate the monthly and annual Pan and Lake Evaporation for the Project, using a distance weighted average and published methods (KP, 2021b), as shown in Table 18-21.

Table 18-21: Estimated Pan Evaporation Rates for Site (230 to 400 mamsl)[95] (KP, 2021b)

Month	Pan Evaporation (mm)	Lake Evaporation (mm)	Pan Factor
Jan	201	144	0.72
Feb	208	146	0.71
Mar	218	155	0.71
Apr	182	131	0.72
May	167	121	0.73
Jun	136	100	0.74
Jul	117	87	0.74
Aug	116	86	0.74
Sep	121	89	0.74
Oct	137	101	0.74
Nov	136	100	0.74
Dec	153	112	0.73
Annual Total	**1891**	**1373**	**0.73**

[93] The period in which 70% of the annual rainfall occurs on average.

[94] Synthetic Precipitation Scenarios.

[95] Section 5.

18.5.3 Site Water Balance and Management

18.5.3.1 Overview and Basis

Given that runoff collected by the proposed WHD is the primary source of water for operations, seasonal water management plays a critical role in the design of the water infrastructure required to support the operation of the mine. Thus, to understand the seasonal and LoM water requirements, a site wide water balance model was developed (daily and then monthly time steps), based on the Water Block Flow Diagram (BFD) illustrated in Figure 18-10.

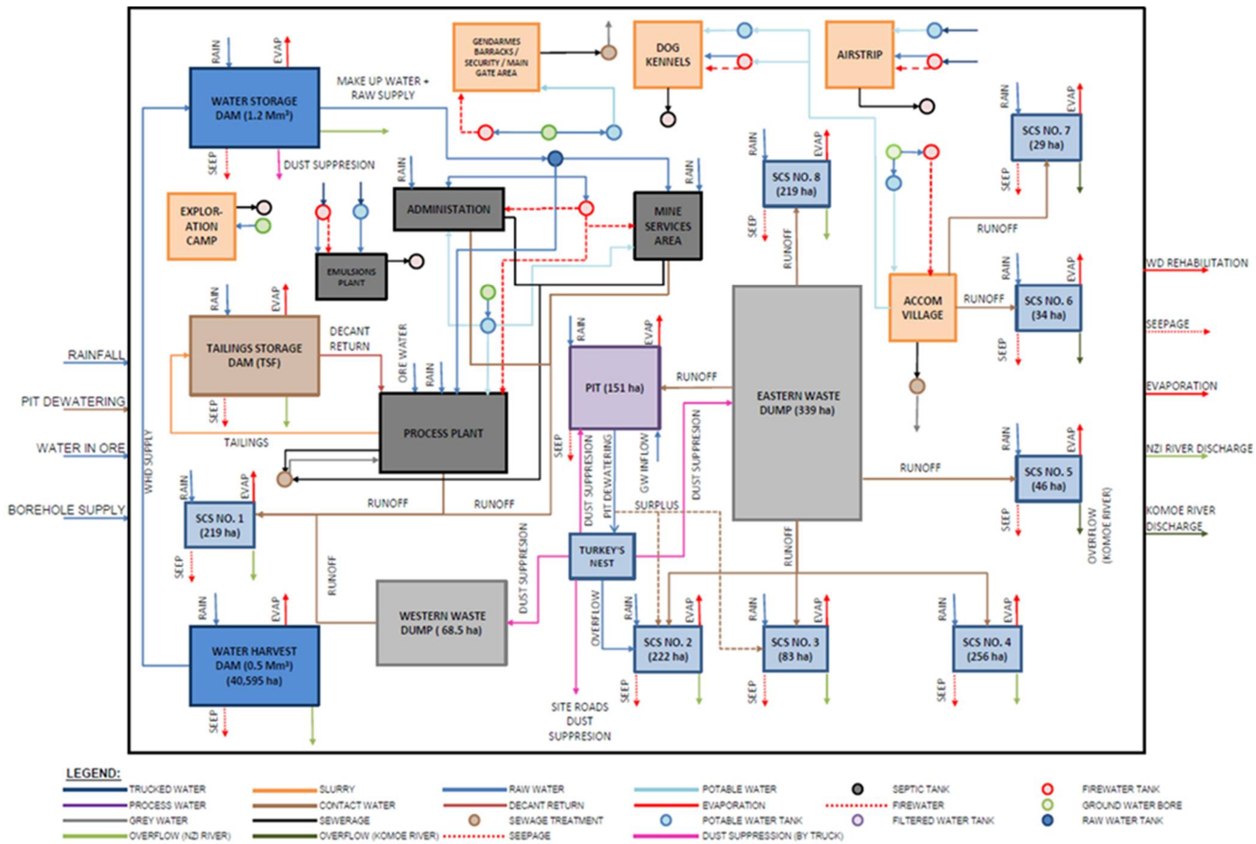


Figure 18-10: Site Water Block Flow Diagram (KP, 2022b)

The water balance model includes the WHD and WSD as a source of raw water for the Site and as a source of process make-up water. Water harvested from the WHD catchment will be pumped directly into the WSD for use as required in the Plant.

The primary objectives of the water balance modelling exercise are summarised below:

- establish the filling rate for tailings solids within the TSF and estimate the in situ tailings density within the TSF, taking into consideration tailings properties from laboratory testing (Section 18.8.1.3) and the TSF basin storage parameters;
- determine supernatant pond volumes within the TSF under average climatic conditions throughout operation;
- determine supernatant pond volumes within the TSF for design wet rainfall sequences and storm events, check TSF storm water storage capacity and confirm the suitability of the current TSF design philosophy;

- determine staged embankment crest elevations, to ensure containment of tailings and design supernatant pond volumes;
- determine the likelihood of recycle water shortfalls during average conditions and design dry rainfall sequences;
- for the various users of water on Site, ensure that their water requirements can be met (volumetric nominal and design);
- determine the required WSD capacity to store make-up water for these shortfalls;
- determine the required WHD capacity and abstraction rate (from WHD to WSD) to provide supplemental make-up water for shortfalls;
- determine the required groundwater supply rate (from pit dewatering and/or groundwater bores) to provide additional make-up water for shortfalls; and
- assess risk factors for water balance modelling.

The water balance modelling included the TSF, WHD, WSD and Plant, with a view of determining site water storage requirements. Design wet conditions were modelled to ensure that the TSF is designed with sufficient storage capacities to comply with design criteria.

For water management modelling, various design rainfall conditions were included for selected operational years. The following rainfall sequences were modelled:

- average conditions;
- 1 in 100 year recurrence interval, 1-year dry rainfall sequence;
- 1 in 100 year recurrence interval, 1-year wet rainfall sequence;
- 1 in 100 year recurrence interval, 72-hour duration storm event superimposed over an average rainfall sequence; and,
- a 1 in 10 year recurrence interval wet season (210 days duration), with 100% runoff and no evaporation.

Catchment and runoff characteristics used in the water balance model are presented in Table 18-22. The runoff coefficients presented, were assumed based on similar regional project experience.

Table 18-22: Catchment Characteristics (KP, 2021f)

Parameter	Value
TSF Catchment Area	274 Ha
WSD Catchment Area	219 Ha
WHD Catchment Area	40 593 Ha
Plant Site Catchment Area	1.9 Ha
Runoff Coefficients (Long-term)	
• Undisturbed areas	4.3% (average rainfall)
• Cleared areas	30%
• Compacted Soil Liner	40%
• HDPE Liner	90%
• Tailings beach	80%
• Ponds	100%
• Plant Site	80%

Table 18-22: Catchment Characteristics (KP, 2021f)

Parameter	Value
• Pit Area	90%
Runoff Coefficients (Short-term event*)	
• Undisturbed (external to basins)	15%
• Cleared areas	50%
• Compacted Soil Liner	80%
• HDPE Liner	100%
• Tailings beach	85%

Table 18-22 notes: *1 in 100-year average recurrence interval, 72-hour duration storm event

Based on the hydrogeology studies completed by Endeavour (EMSA, 2021), the predicted annual ground water inflows to the pit, range from 0 m^3/h in the first year of mining to 66 m^3/h (18 L/s) in the final year of mining, with an overall average of 31 m^3/h (9 L/s). The predicted yearly inflow rates indicate the annual minimum requirement for dewatering of the open pit(s). It is intended that the pit dewatering could potentially be pumped to the WSD for use in the process circuit, and as such will supplement the WHD abstraction volumes into the WSD. Water will also be pumped to a turkey's nest, with this water typically used for dust suppression in the mine and on Site/Off-Site roads.

Process parameters used in the water balance modelling and additional water requirements (both provided by Lycopodium) are summarised in Table 18-23 and Table 18-24, respectively. Seasonal variation for dust suppression was considered.

Table 18-23: Process Parameters (KP, 2021f)

Parameter	Average Flow (% of water in process stream)	
	Oxide	Fresh
Water in Ore	6.8%	3.3%
Minimum Raw Water Required (e.g. reagents, gland)	31.8%	31.9%
Maximum Allowable TSF Recycle	61.3%	64.7%

Table 18-24: Process Parameters (KP, 2021f)

Parameter	Average Flow (m^3/h)	
	Oxide	Fresh
Additional Fresh Water	20	20
Mine Services and Mine Dust Suppression	50*	50*

Table 18-24 notes *Peak rate, factors applied to this value to allow for night shift (75%) and wet season (50%).

A range of basin permeability values were considered to assess the impact of permeability on required storage capacity of both the WSD and WHD. Based on the outcomes of the geotechnical investigation (Section 18.2), the WHD and WSD basin permeability for the water balance modelling was assumed to be 1 x 10^{-7} m/s.

18.5.3.2 Storm Water Management on the TSF

Based on the assumptions in Sections 18.5.2 and 18.5.3.1, and in accordance with Mine Schedule 12i,[96] it was noted that the supernatant pond volume peaks in September of each year (at the end of the wet season), before returning to the minimum operating pond volume during the subsequent dry season.

A plot of pond volume against time is illustrated in Figure 18-11, following, whilst pond volume as a function of different rainfall event scenarios, is presented in Table 18-25.

The TSF is designed to hold the tailings plus the design rainfall volume, and thus has sufficient storm water storage capacity for all design storm events and rainfall sequences, namely the TSF has a minimum storm water storage capacity of 3.6 Mm^3 (Stage 1) to 5.5 Mm^3 (final).

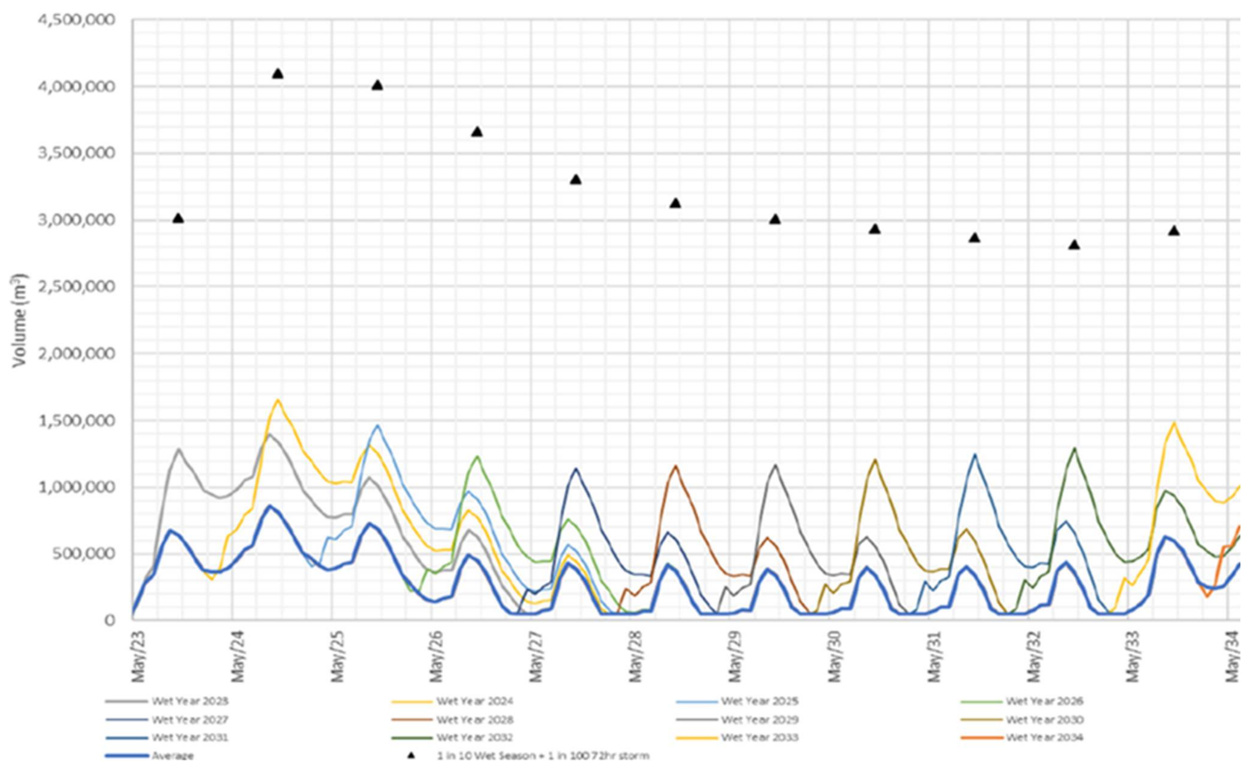


Figure 18-11: TSF Supernatant Pond Volumes (KP, 2021f)

[96] The TSF is designed to accommodate a total of 41Mt (db) of tailings based on the Sc12I mining schedule. It is noted that the current mining schedule (Sc13B) requires an additional 6.7 Mt of tailings; however, the additional tonnage occurs during the final years of operation Therefore, the supernatant pond volumes estimate in the water balance modelling are not anticipated to change significantly with the updated mining schedule.

Table 18-25: Wet Conditions - Peak Supernatant Pond Volumes (KP, 2021f)

Year (in which event occurs)	Peak Supernatant Pond Volume (m^3)		
	Average Conditions	1 in 100 Year, 12 Month Wet Sequence	1 in 100 Year, 72 hour Storm Event + 1 in 10 Year, Wet Season Runoff *3
2023*1, 2	674 000	1 287 000	3 018 000
2024	861 000	1 660 000	4 103 000
2025	731 000	1 470 000	4 016 000
2026	493 000	1 235 000	3 661 000
2027	430 000	1 145 000	3 310 000
2028	417 000	1 163 000	3 133 000
2029	388 000	1 171 000	3 014 000
2030	403 000	1 208 000	2 937 000
2031	405 000	1 250 000	2 872 000
2032	438 000	1 293 000	2 823 000
2033	628 000	1 488 000	2 923 000
2034*2	429 000	1 050 000	2 311 000

Table 18-25 notes:

- *1 Pond volumes for 2023 are pre-commissioning (July 2023)
- *2 2023 operation comprises July - December only; 2034 operation comprises January - June only
- *3 No evaporation and 100% runoff

With respect to stormwater management on the TSF, the following points may be noted:

- The critical design rainfall event in terms of pond elevation is the design storm event occurring in the last month of each stage of operation, when the TSF storm water capacity is at a minimum. There is sufficient stormwater storage capacity in the TSF for this situation.
- It is noted that the critical wet season pond volumes occur in October each year, when there is significant stormwater capacity over and above the critical wet season volume.

18.5.3.3 Water Recovery from the Tailings Storage Facility

Estimated decant return rates for average and design dry conditions are provided in Table 18-26 and Table 18-27.

Table 18-26: Average Conditions - TSF Recycle Rates (KP, 2021f)

Year	Total Annual Water in Slurry Volume (m^3/year)	Total Recycle Volume (m^3/year)	Average Monthly Recycle Rate (%)	Maximum Monthly Recycle Rate (%)	Minimum Monthly Recycle Rate (%)
2023	1 834 000	1 187 000	65%	65%	65%
2024	4 278 000	2 770 000	65%	65%	65%
2025	4 889 000	3 166 000	65%	65%	65%
2026	4 889 000	3 166 000	65%	65%	65%
2027	4 889 000	3 052 000	62%	65%	50%
2028	4 889 000	2 981 000	61%	65%	47%
2029	4 889 000	2 939 000	60%	65%	47%

Table 18-26: Average Conditions - TSF Recycle Rates (KP, 2021f)

Year	Total Annual Water in Slurry Volume (m³/year)	Total Recycle Volume (m³/year)	Average Monthly Recycle Rate (%)	Maximum Monthly Recycle Rate (%)	Minimum Monthly Recycle Rate (%)
2030	4 889 000	2 911 000	60%	65%	44%
2031	4 889 000	2 904 000	59%	65%	41%
2032	4 889 000	2 909 000	60%	65%	40%
2033	3 682 000	2 139 000	58%	65%	40%
2034	1 238 000	802 000	65%	65%	65%

Table 18-26 notes: 2023 operation comprises July to December only; 2034 operation comprises January to June only.

Table 18-27: Dry Conditions – TSF Recycle Rates (KP, 2021f)

Year	Total Annual Water in Slurry Volume (m³/year)	Total Recycle Volume (m³/year)	Average Monthly Recycle Rate (%)	Maximum Monthly Recycle Rate(%)	Minimum Monthly Recycle Rate (%)
2023	1 834 000	1 152 000	63%	65%	61%
2024	4 278 000	2 714 000	63%	65%	46%
2025	4 889 000	3 130 000	64%	65%	56%
2026	4 889 000	2 883 000	59%	65%	41%
2027	4 889 000	2 616 000	54%	65%	37%
2028	4 889 000	2 473 000	51%	65%	34%
2029	4 889 000	2 372 000	49%	65%	32%
2030	4 889 000	2 307 000	47%	65%	31%
2031	4 889 000	2 283 000	47%	65%	30%
2032	4 889 000	2 306 000	47%	65%	29%
2033	3 682 000	1 885 000	51%	65%	27%
2034	1 238 000	719 000	58%	65%	25%

Table 18-27: 2023 operation comprises July to December only; 2034 operation comprises January to June only.

18.5.3.4 Site Water Make-up Requirements

Decant return/process water shortfall is expected to occur under average and design dry climatic conditions. Process water shortfall volumes are provided in Table 18-28. The shortfalls provided in Table 18-28 are for TSF recycle water only, and prior to the WSD supply to the process plant.

Table 18-28: Dry Conditions - Plant Water Shortfall (Prior to WSD Supply) (KP, 2021f)

Year	Average Conditions				Design Dry Conditions			
	Recycle Water Shortfall Volume	Recycle Water Shortfall (% of water in tailings slurry)	Raw Water Required	Total Water Demand	Recycle Water Shortfall Volume	Recycle Water Shortfall (% of water in tailings slurry)	Raw Water Required	Total Water Demand
	m^3/a	%	m^3/a	m^3/a	m^3/a	%	m^3/a	m^3/a
2023*		0	751 000	751 000	36 000	2%	757 000	793 000
2024		0	1 732 000	1 732 000	57 000	1%	1 731 000	1 788 000
2025		0	1 926 000	1 926 000	36 000	1%	1 934 000	1 970 000
2026		0	1 926 000	1 926 000	283 000	6%	1 934 000	2 217 000
2027	114 000	2	1 926 000	2 040 000	550 000	11%	1 934 000	2 484 000
2028	186 000	4	1 926 000	2 113 000	693 000	14%	1 935 000	2 628 000
2029	227 000	5	1 926 000	2 152 000	795 000	16%	1 933 000	2 728 000
2030	255 000	5	1 926 000	2 181 000	859 000	18%	1 934 000	2 793 000
2031	262 000	5	1 926 000	2 188 000	883 000	18%	1 934 000	2 817 000
2032	258 000	5	1 926 000	2 185 000	860 000	18%	1 935 000	2 795 000
2033	246 000	7	1 540 000	1 786 000	500 000	14%	1 547 000	2 047 000
2034*		0	595 000	595 000	83 000	7%	594 000	677 000

Table 18-28 *2023 operation comprises July to December only; 2034 operation comprises January to June only.

All Site make-up water requirements can be provided by the WSD reservoir, supplemented with water from the WHD for design dry conditions. It is necessary that the WHD and WSD are completed early to allow a full wet season for filling prior to commissioning.

A WSD storage capacity of 1.2 Mm^3[97] is required to provide sufficient make-up water, supplemented by an abstraction rate of 536 m^3/h from the WHD. The WSD stored reservoir volume over time for average and design dry conditions is plotted on Figure 18-12.

[97] Late stage DFS change to 1.8 Mm^3, see Section 18.5.4.2 (costs included in DFS).

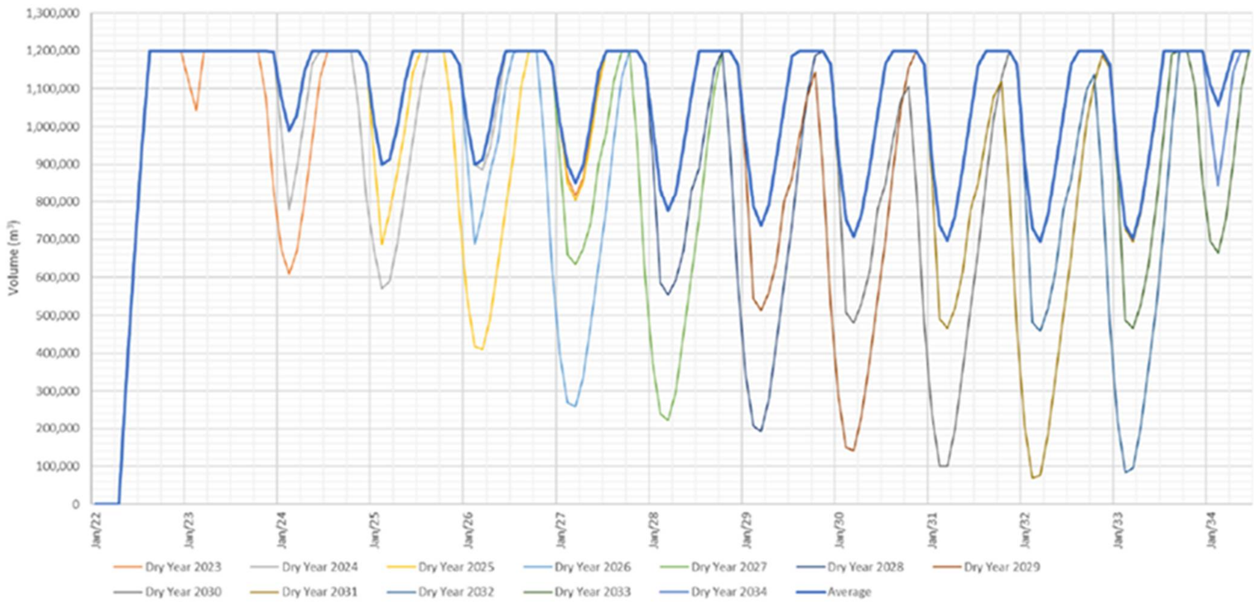


Figure 18-12: WSD Reservoir Volume Over Time (KP, 2021f)

A WHD capacity of 540 000 m³ is required to reduce the risk of shortfalls under design dry conditions.

18.5.4 Water Harvest Dam

18.5.4.1 Overview

Given that there are no perennial sources of water close to the Site, a WHD will be built to capture surface water run-off occurring in the wet season to meet the Mine's LoM water requirements. From the WHD, water will be pumped to the WSD before use. As per the DFS documentation prepared by KP, the combined storage capacity of the two dams is 2.14 Mm³.

Layouts for the WHD and WSD are illustrated in Figure 18-13 and Figure 18-14 and described more fully in Sections 18.5.4.2 and 18.5.4.3, respectively.

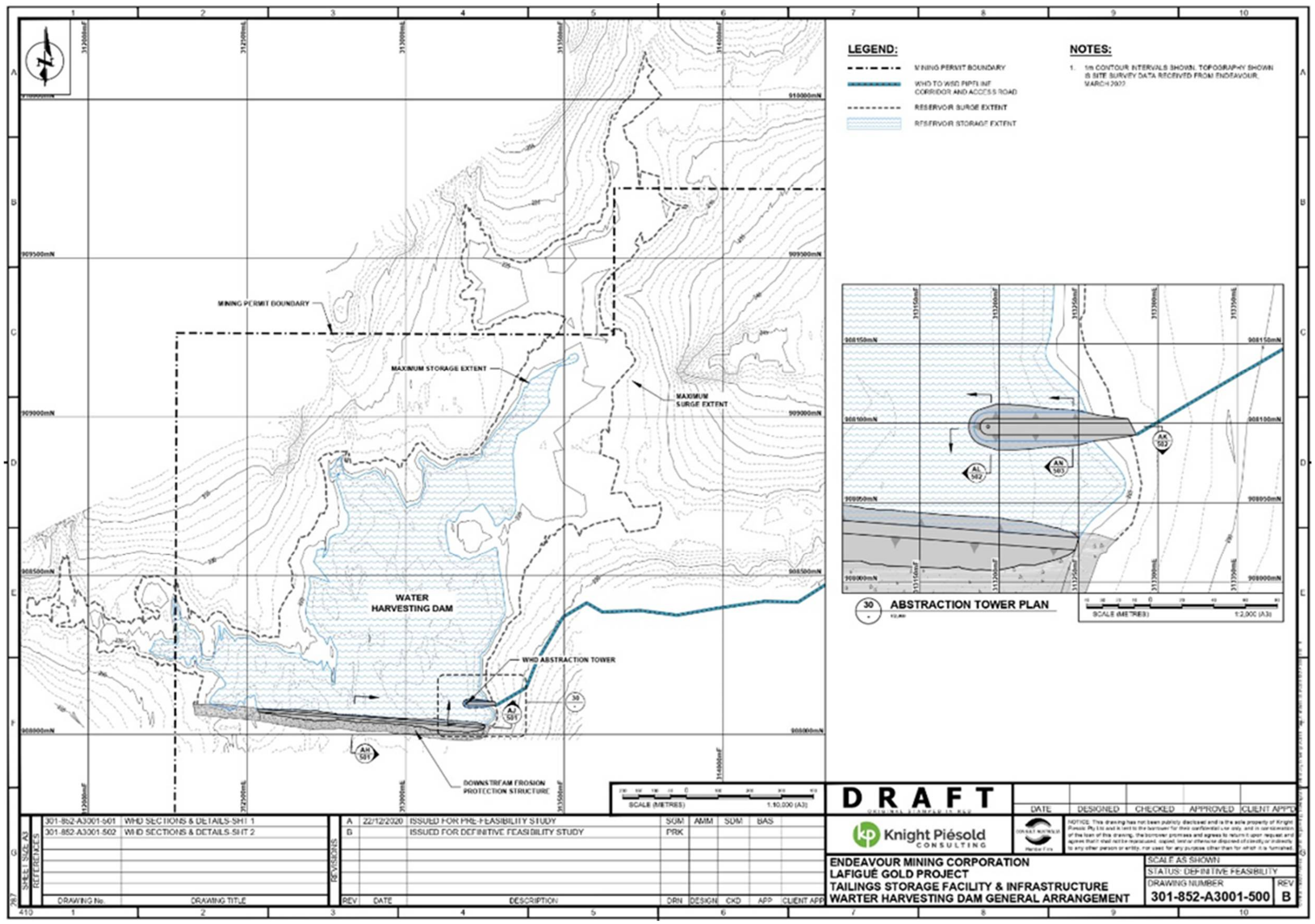


Figure 18-13: WHD Layout (KP, 2022b)

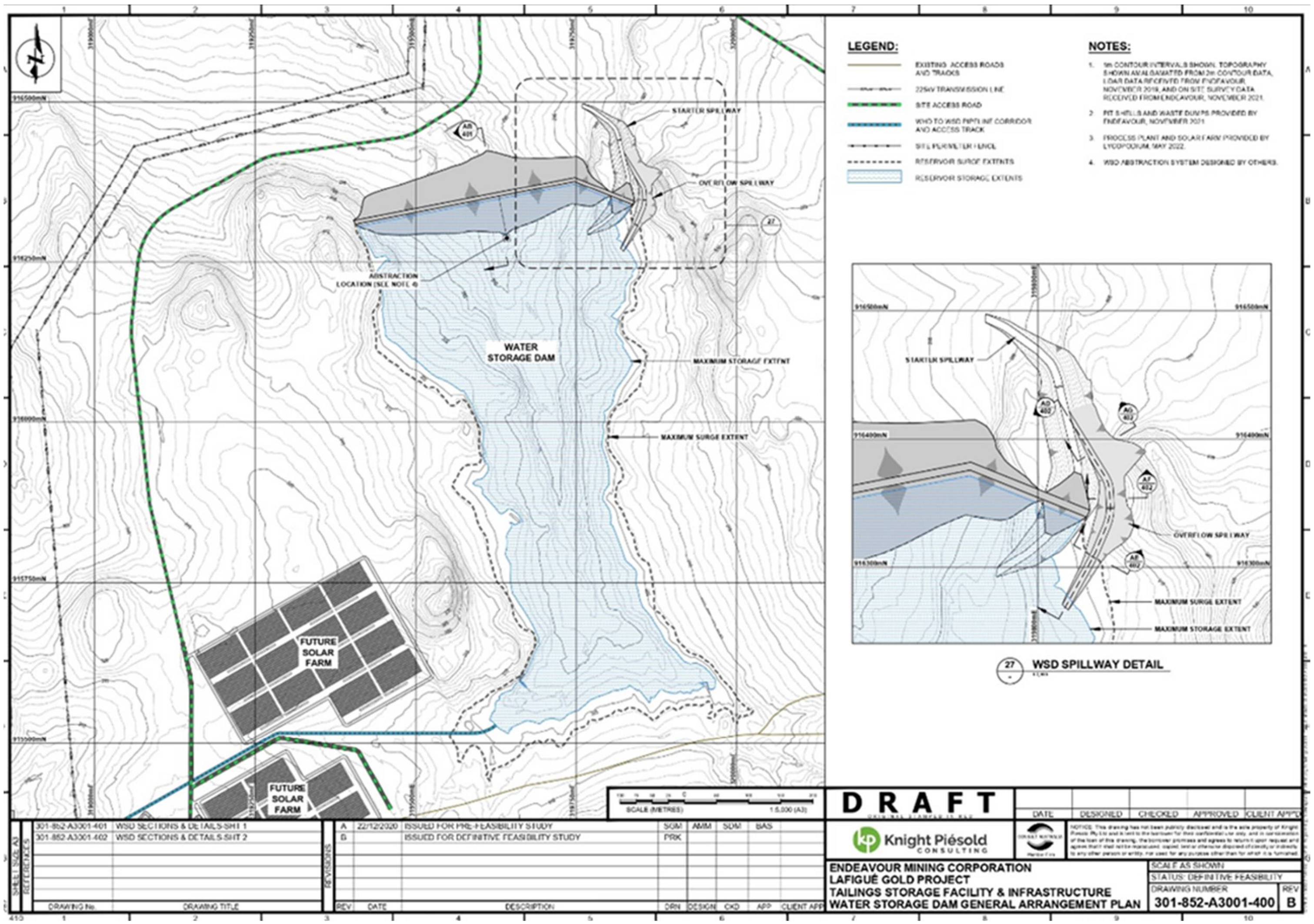


Figure 18-14: WSD Layout (KP, 2022b)

18.5.4.2 Water Harvest Dam

The WHD design was completed to a DFS level of technical and cost development by KP (KP, 2022b). The WHD will be the Site's primary water collection structure, and will be able to store up to 0.34 Mm^3 of water at the maximum operating level. The design intent of the WHD is that the reservoir is frequently pumped to the WSD during each wet season, with a view to filling the WSD reservoir to its maximum storage level prior to each dry season. The WHD will operate as a barrage-type structure, designed so that any overflow will pass over the full width of the embankment crest. The WHD has a catchment area of 40 593 ha, and when the pond volume is at maximum level the reservoir surface area will be 51 ha.

During late study optimisation work of the DFS, a costing review of the WHD and WSD design was undertaken to assess potential capital cost reductions. As an outcome of this optimisation work, an alternate cross section for the WHD was adopted as the basis of the DFS capital costing and for future detailed design development for the Project. The alternate cross section includes a low permeability fill (Zone A) central core surrounded by coarse rockfill (Zone G), replacing the concrete lined barrage included in the original DFS design. This option reduces construction costs by using a maintainable cross section and adoption of an observational approach during the operation (i.e. repair and remediation as needed). A similar approach has been adopted on previous projects, with repairs to the spillway, embankment crest/downstream face being completed during dry conditions. In addition, the pumping rate from the WHD to WSD was increased to 605 m^3/h (KP, 2022c).[98]

Design parameters for the WHD are provided in Table 18-29.

Table 18-29: WHD Design Parameters (KP, 2022c)

Storage Capacity	0.34 Mm^3
Outlet Structure Capacity	10 000-year ARI for embankment surge elevation, 100-year ARI for downstream outlet structure
Catchment Area	40 593 Ha

The WHD embankment comprises a central low permeability core (Zone A), with upstream and downstream structural zones (Zone C). A cut off trench will be located beneath the low permeability core (Zone A) of the embankment, constructed continuously along the embankment and backfilled with low permeability fill (Zone A). The upstream embankment face embankment crest and downstream embankment face will be lined with reinforced concrete (nominal 200 mm thickness). Above the embankment crest, the existing ground on the abutments will be scarified, moisture conditioned and compacted before being lined with reinforced concrete, to the design surge elevation.

Typical specifications for material types are summarised as follows:

- Zone A material shall be won from borrow to form the low permeability core of the embankment.
- Zone C material shall be won from borrow to form the outer structural zones.

[98] Value engineering WHD changes were not updated in the DFS documentation outlined herein, due to time constraints.

An outlet structure will be installed at the downstream toe of the embankment to dissipate flows over the WHD embankment prior to discharge. The outlet structure will comprise a reinforced concrete-lined 'plunge pool'. Discharge from the WHD will occur in a controlled manner via the outlet structure. As the WHD is expected to fill during each year of operation, it is anticipated that overflow will occur frequently each wet season (April to October) and discharged water will report to the existing stream bed downstream of the WHD.

Water will be removed from the WHD by submersible pump(s) situated at the base of an abstraction tower.

Vibrating wire piezometers will be installed at the base of the WHD embankment to allow monitoring of the phreatic surface within the embankment. Survey pins will be installed at regular intervals along the WHD embankment crests to monitor embankment movements and assess effects of any such movement on the embankment.

Source control measures, examples listed in 18.5.6, should be installed upstream of the WHD reservoir to reduce the amount of sediment reporting to the WHD from artisanal mining operations.

At closure, the WHD embankment will be breached to allow flow-through, silt will be removed from the reservoir area, topsoil replaced and revegetation in areas outside of the stream bed. Alternatively, the structure may be left to the local communities.

18.5.4.3 Water Storage Dam

The WSD design was completed to a DFS level of engineering and cost development (KP, 2022b). The WSD is the primary storage pond for clean process water on Site and will be able to store up to 1.2 Mm^3 of water at its maximum operating level.

During late study optimisation work of the DFS, a costing review of the WSD design was undertaken to assess potential capital cost reductions. As an outcome of this optimisation work, the pumping rate from the WHD to WSD was increased to 605 m^3/h, which required an increase in the WSD capacity to 1.8 Mm^3. This revised WSD capacity was adopted as the basis of the DFS capital costing and for future detailed design development for the Project. However, the details of the DFS documentation outlined herein was not updated for this change, due to them occurring at the end of the DFS. Refer to KP DFS WHD/WSD optimisation memorandum (KP, 2022c) for further information. Design parameters for the WSD are provided in Table 18-30, following.

Table 18-30: WSD Design Parameters (KP, 2022c)

Storage Capacity	1.8 Mm^3
Spillway Capacity	100-year ARI/critical duration
Catchment	219 ha

The WSD has a catchment area of 219 ha, and when the pond volume is at maximum level, the reservoir surface area will be 22 ha. The WSD is intended to be recharged by water abstracted from the WHD and rainfall runoff from its upstream catchment. Pit dewatering will be pumped to a turkey's nest, and it was assumed that dust suppression and wash down water will be sourced from both the turkey's nest and/or the WSD.

The water collected in the WSD will be pumped to the Plant to supply the Plant's raw water requirements and process make-up water requirements. Water will be recovered from the WSD by a floating pump.

The WSD embankment comprises a central low permeability core (Zone A), with outer structural zones (Zone C). A chimney drain (Zone F1) and embankment finger drains are included within the embankment cross section. The upstream embankment face will be lined with textured HDPE geomembrane liner for erosion protection. Topsoil will be spread over the downstream embankment face, which will be vegetated with local grasses. Typical specifications for material types are summarised as follows:

- Zone A material shall be won from borrow to form the low permeability core of the embankment.
- Zone C material shall be won from borrow to form the outer structural zones.
- Zone F1 shall be clean sand/gravel drainage material supplied to a stockpile adjacent to the works.

The WSD basin area will be cleared, grubbed and topsoil stripped to ensure that the process water supply remains free of organic material.

To protect the integrity of the embankments from overtopping failure, discharge from the WSD will occur in a controlled manner via an engineered spillway. The emergency spillway will be lined with riprap erosion protection. As the WSD is expected to fill during each year of operation, it is anticipated that the spillway will overflow each wet season, and discharged water will report to the existing stream bed downstream of the WSD.

Standpipe piezometers will be installed in the WSD embankment to monitor pore water pressures within the embankments. The base of each piezometer will be located within the embankment fill to ensure that the phreatic surface within the embankment, as opposed to natural groundwater level, is being measured. Survey pins will be installed at regular intervals along the WSD embankment crests to monitor embankment movements and assess effects of any such movement on the embankment.

Source control measures, examples listed in Section 8.2.5, should be installed upstream of the WHD reservoir to reduce the amount of sediment reporting to the WHD from artisanal mining operations in the stream bed upstream of the WHD.

At closure, the WSD embankment will be breached to allow flow-through, silt will be removed from the reservoir area, topsoil replaced and revegetation in areas outside of the stream bed. Alternatively, the structure may be left to the local communities.

18.5.5 Dam Breach Assessment

A dam breach assessment (KP, 2021h) based on ANCOLD guidelines was carried out for the WSD and WHD to estimate the Population at Risk (PAR), business risk and environmental impact in the event of a dam failure.

In the event of dam failure, a significant volume of water would be released. The WSD and WHD reservoir stored volumes are 1.20 Mm^3 and 0.54 Mm^3 respectively, when filled to the spillway invert and it was assumed that all of this water would mobilise if failure occurred where the dam is at its maximum height. In the event of failure, it is expected the water will be conveyed downstream by topographical features within existing water courses.[99]

It is also noted that in the event of a WSD failure the outflow will report downstream to the WHD where it has been conservatively assumed that 100% of the storage capacity of the WHD will be released. It is anticipated that the additional water mobilised from the failure of the WHD will lead to a larger runout distance downstream and the likelihood of the flow reaching the N'zi River would increase.

[99] Dam break modelling was based on the 1.2 Mm3 not the revised 1.8 Mm^3 storage volume. This is not expected to significantly change outcomes of the modelling.

The failure of the WSD and WHD embankment at the highest point, is considered to have the following impacts:

- The WHD downstream of the WSD will be inundated. The stored water will be displaced and report to the N'zi River. The WSD and WHD water is assumed to be clean and not contain environmentally significant elements.
- Water will enter the N'zi River downstream of the site infrastructure. It is noted that the N'zi River is a tributary of the Bandama River, the largest River in CI.
- A number of site and local access roads are at risk of being cut-off for a short period (and may be washed out) in the event of failure. Security personnel are expected to pass downstream of the WSD during routine inspections of the site perimeter.
- A review of satellite imagery indicates that some local agriculture areas downstream, may be affected by the failure.
- A number of small settlements were identified downstream near the N'zi and Bandama Rivers; however, they are generally located in areas of higher ground relative to the flow path and as such present low risk in the event of failure.
- Based on the above analysis/observations, a PAR of '≥1 to <10' for Dam Failure is recommended.

The key conclusions from the Dam Breach Assessment for the Lafigué WSD and WHD are summarised as follows:

- A PAR of '≥1 to <10' is expected for the assumed failure case for both the WSD and WHD.
- For both the WSD and WHD, the Severity Level would be 'Medium' On the basis of significant damage to the business (primarily temporary loss of project water supply).
- The ANCOLD Dam Failure Consequence Category would be 'Significant'.
- The ANCOLD Environmental Spill Consequence Category would be 'Very Low'.
- The Dam Breach Assessments for the WSD and WHD will be updated to reflect the current design capacities during the detailed design phase.

18.5.6 Sediment Management

All contact water from site infrastructure (primarily the waste dumps, Plant and MSA) will report to the sediment control system (SCS) prior to discharge to the receiving environment. The positioning of these structures relative to other site infrastructure is shown in the Site water BFD (Figure 18-10), and in the SCS Location Plan (Figure 18-22). It is estimated that 87% of SCS overflow will ultimately report to the N'zi River, with the remaining 13% reporting to the Komoe River.

Design parameters for the SCSs are provided in Table 18-31 and Table 18-32.

Table 18-31: SCS Design Parameters (KP, 2022b)

Design Maximum Water Depth	2.0 m
Spillway Capacity	100-year ARI storm event, occurring when pond is at spillway inlet level.

The SCSs are labelled SCS No. 1 through to SCS No. 8 (Table 18-32). SCS Nos. 7 and 8 will form part of the site access road between the Plant and accommodation village/airstrip. The SCSs were designed to limit maximum water depth to 2 m for safety reasons (drowning risk). As such, the maximum embankment height for the SCSs is approximately 3 to 4 m. The SCS embankment will be a homogeneous earth fill embankment comprising low permeability fill (Zone A), won from local borrow areas within the SCS basin if possible. The upstream face of the SCS embankments will be lined with riprap (Zone E) for erosion protection, and the downstream face will be revegetated.

Table 18-32: SCS Design Summary (KP, 2022b)

SCS	Volume (m³)	Catchment Area (ha)	Surface Area (m2)	Earth Works Volume (m³)	Outflow to:
SCS 1	3300	26.1	9100	2000	N'zi River
SCS 2	1500	415.4	6300	1400	N'zi River
SCS 3	1500	146.2	6300	900	N'zi River
SCS 4	1100	286.3	2000	2100	N'zi River
SCS 5	700	145.5	1900	900	Komoe River
SCS 6	1500	33.8	1900	1400	Komoe River
SCS 7	6100	28.6	7600		Komoe River
SCS 8	22 600	506.0	37 100		N'zi River

Discharge from the SCS Nos. 1 to 6 will be to the environment downstream of the Site via an engineered spillway. The spillway will be lined with erosion protection material (Zone E). Discharge from the SCS Nos. 7 and 8 will be to the environment downstream of the project site via culverts installed within an abutment of the SCS embankment, to protect the embankment from erosion/piping damage. The spillways and culvert inlet/outlets will be lined with erosion protection material (Zone E). As the SCSs are expected to fill frequently, it is anticipated that the spillway will flow frequently during each wet season.

At closure, the SCS embankments will be breached to allow flow-through, silt will be removed from the reservoir area, topsoil replaced and revegetation in areas outside of the stream bed.

18.6 Site Services

18.6.1 Security and Fencing

18.6.1.1 Overview

Security for the Site will be implemented using Endeavour security standards, which include the following features:

- Perimeter monitoring of Plant.
- Targeted monitoring of high-risk areas.
- Access control to high security areas for personnel and vehicles.
- Remote monitoring of operations (via CCTV).
- Security fencing of facilities.

The following categories of security fencing will be provided for the Site:

- Level 2, High Security Fencing: 2.4 m high wire mesh 'cyclone' fence with coiled razor wire above the top strand and along the inner base.
- Level 3, General Security Fencing: 1.8 m high wire mesh 'cyclone' fence with 3-strand barb wire above the top strand.

18.6.1.2 Site Perimeter Security

The Site will have a Level 2 perimeter fence around the entire site, including the airstrip (approximately 28 km total length of fencing). A main entrance gatehouse will be provided at the northeast entrance to Site, joining the main access road. A secondary gatehouse will be provided to control access along the access track to the water harvesting dam. A gated system in the fence line along the access road to the airstrip will be provided to enable access across this road by the local community when the airstrip is not in use.

A track suitable for 4WD vehicle patrols will be provided inside the perimeter fence line plus a track on the outside to allow the community to circumnavigate the fenced perimeter.

18.6.1.3 Accommodation Camp Security

The accommodation camps will each have a local area Level 2 perimeter fence, including a security gatehouse with manual boom gate controlled by security guards on 24-hour duty. This includes approximately 1.5 km of perimeter fencing at the Permanent Accommodation Camp and 350 m of perimeter fencing at the Gendarmes Barracks. No lights or cameras will be provided around these perimeter fences.

18.6.1.4 Plant Security

The Plant area will have approximately 3.9 km of Level 2 high security double fence line surrounding the process plant with a 10 m separation, creating a central 'no man's land' between the fences. The no man's land will be cleared and grubbed. Lighting masts, located along the inner perimeter fence will be provided, to illuminate the high security fence and surrounds. For perimeter monitoring, motion sensing digital CCTV cameras will be provided at regular intervals along the fence line.

A separate gatehouse with turnstile and search facility will regulate personnel entry to and from the Plant. Vehicle access to the Plant area from the main gate will be controlled via a palisade gate. Personnel access to various security areas will be generally controlled via swipe card access.

18.6.1.5 ROM Pad

Normally locked double gates through both fences will be provided to allow restricted access for mobile equipment to enter the process plant via the ROM pad. A personnel access gate at the primary crusher top of ROM pad level will provide restricted access for plant operations and maintenance personnel to access the ROM pad for dump pocket and ROM grizzly inspections.

18.6.1.6 Gravity Circuit

Fixed wire mesh panels or fencing with locked personnel gates will be provided for the gravity concentrators and the intensive cyanidation reactor (ICR). Fixed digital CCTV cameras will be provided for these areas.

18.6.1.7 Goldroom Security

The goldroom will have restricted access limited to authorised personnel, with security guards providing 24-hour coverage. Goldroom security will include magnetic locks with a Biometric entry together with motion sensors, CCTV and proprietary security.

18.6.1.8 Local Area Security

Local area Level 3 fencing will be provided to secure and control access to other specific areas around Site, including but not limited to: warehouse facilities, laydown yards, electrical switchyard, remote water supply bore pump stations and the WHD pumping station.

Access to the Level 3 areas will generally be via a manual boom gate controlled by a security post with no lights or cameras provided around the perimeter fence. Remotely located Level 3 areas may be accessed via a locked gate where a security post is not provided.

The Emulsion Plant and Explosives Storage facility will have Level 2 high security fencing provided including a security gatehouse with manual boom gate controlled by security guards on duty 24-hours per day.

18.6.2 Water Systems

18.6.2.1 Raw Water Supply

Raw water will be primarily sourced from a WHD located approximately 9 km southwest of the Plant site, collecting rainwater runoff during the wet season. Water will be pumped via a pipeline to the WSD located near the Plant. For security reasons, the water pipeline will be buried underground for the section of pipeline located outside of the mine perimeter fenced area. From the WSD, water will be reticulated to the Plant and other Site facilities. The water supply dams and water management as a whole are described in Section 18.5.

Water supply for the potable water treatment plants will be sourced from local ground water bores, rather than the WSD.

18.6.2.2 Potable Water Treatment and Reticulation

Potable water will be generated by the treatment of raw water from dedicated borefields proximate to the points of use. Potable water treatment plants will be provided at the Plant, gendarmes barracks and permanent accommodation camp. The water treatment facilities will include micro filtration, ultra-violet sterilisation and chlorination. Potable water storage tanks will be provided at each facility, with the potable water being reticulated around the camps and at the plant to the site ablutions, safety showers and other plant potable water outlets, including the administration buildings, workshops, warehouses and MSA facilities.

Ultra-violet sterilisation units will be installed on the outgoing potable water distribution headers close to the points of use.

Potable water will also be transported to remote facilities on Site in large refillable water containers for use in water coolers.

18.6.2.3 Plant Water and Reticulation

Raw water for the process plant will be reticulated from the WSD to the Plant raw water tank. The Plant raw water tank will have sufficient capacity to minimise the impact of short-term supply interruptions. The Plant raw water pumps will distribute raw water to the Plant and MSA facilities. Recycling of water within the Plant will be maximised to minimise raw water usage.

Section 17 of this Report details the various process plant water services, including; raw water, firewater, mill water, process water, fluidising water, filtered water, gland water, potable water and cooling water.

18.6.2.4 Dust Suppression Water Systems

General dust suppression around the Site including dust generated from roads (Site and access), earthworks and mining activities will be managed by water spraying from water trucks/carts. Water trucks/carts will be filled from water standpipes at turkey's nests and/or water tanks located around the Site. For the DFS, no chemical additives have been allowed for managing dust.

18.6.3 Bulk Fuel Supply, Storage, Distribution and Dispensing

18.6.3.1 Country Fuel Supply Overview

Fuel supply and distribution in CI is discussed in Section 5 of this Report.

Until such time as fuel supply agreements are signed, the sourcing/depot for fuel supply is not yet defined. For operations, Endeavour's policy is to have 15 to 20 days of fuel storage capacity at the mine to cover for unplanned fuel supply interruptions.

Based on in-country capacity, no fuel supply issues are foreseen.

18.6.3.2 Mine Fuel Supply Basis

Diesel fuel will be transported by road to site from Yamoussoukro, using bulk fuel road tankers that will be unloaded at a bulk fuel storage facility. Fuel will be distributed on Site using mobile refuelling trucks to supply diesel for remote fuel requirements, including; diesel power generators, plant diesel tanks, waste incinerator, explosives emulsion plant and in-pit mine fleet refuelling.

A vendor supplied bulk fuel storage (1300 m^3 total storage capacity) and pumping system will be supplied as part of the diesel fuel supply contract. This facility will be located between the MSA and Emergency Backup Power Station, and will include:

- fuel unloading station;
- bunded bulk storage tanks (2 x 600 m^3 tanks) to provide 18 days onsite storage;
- transfer pumping system to heavy vehicle day tanks (2 x 40 m^3) for refuelling heavy vehicles in the mine services refuelling bay (2 x 25 m^3/h);
- fuel loading bowser (1 x 40 m^3/h) for filling mobile refuelling trucks;
- transfer pumping system to light vehicle day tanks (2 x 10 m^3) for light vehicle refuelling bowsers (2 x 8 m^3/h dispensers);
- filtration system (3 stages up to 6 µm);
- fuel management system;

- fire suppression system; and a
- lubricants storage facility.

A fenced serviced area (bulk earthworks preparation, power, water, communication, etc) will be provided to the fuel supplier for the construction of these facilities.

The diesel fuel supply and distribution system for the Plant is discussed in Section 17.

An additional light vehicle fuel storage and refuelling facility will be provided near the main administration and warehouse area, outside of the Plant high security area.

The estimated diesel fuel consumption during the operations phase is summarised in Figure 18-15. The maximum quantity of fuel delivered is~ 29 000 m^3/a (Endeavour, 2022a), which equates to up to three[100] fuel truck deliveries per day on average.

Three fuel samples are taken from each truck on delivery, one is retained as reference sample, whilst the other two samples are retained by the buyer and seller, respectively, for testing.

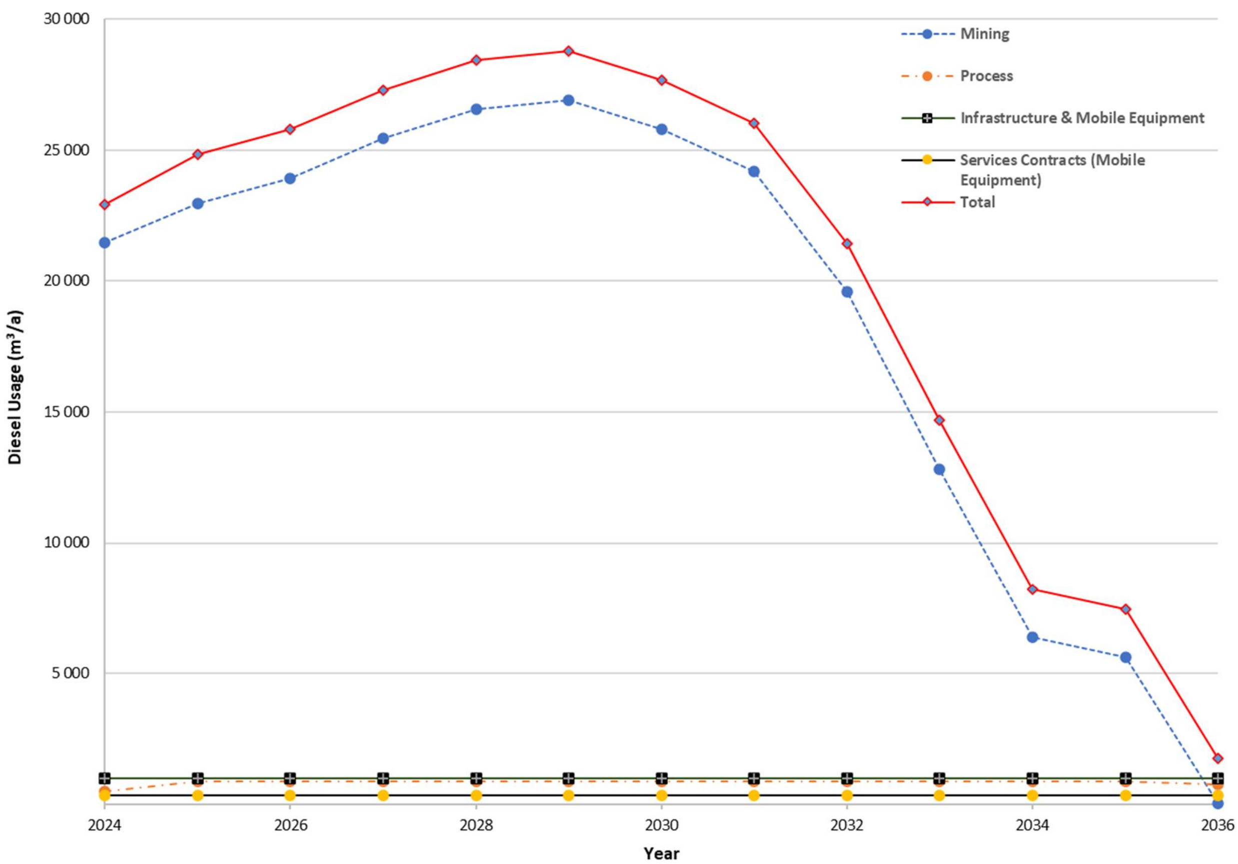


Figure 18-15: Estimated Diesel Fuel Consumption (m^3) During Operations (Endeavour, 2022a)

[100] Assumes that fuel will be delivered during week days only.

18.6.4 Fire Detection and Protection

Fire detection and protection systems will be provided for the Site are as summarised below.

18.6.4.1 Firewater System

Fire water for the Plant and adjacent MSA area will be drawn from the raw water tank fire reserve. Suctions for other water services fed from the raw water tank will be at a higher level to ensure a fire water reserve always remains in the raw water tank.

The fire water pumping system will contain:

- an electric jockey pump to maintain fire ring main pressure;
- an electric fire water delivery pump to supply fire water at the required pressure and flowrate; and
- a diesel driven fire water pump that will automatically start in the event that power is not available for the electric fire water pump or that the electric pump fails to maintain pressure in the fire water system.

Fire hydrants and hose reels will be placed throughout the Plant, Plant buildings, administration offices and MSA area at intervals that ensure appropriate coverage in areas where flammable materials are present. Fire suppression systems for the bulk fuel storage facility will be provided as part of the fuel supply contract.

Electrical switchrooms will incorporate Very Early Smoke Detection Apparatus (VESDA) fire detection systems. Smoke sensing input points will be located in each individual high voltage panel or MCC tier. Signals from the VESDA fire detection systems will be wired to the PLC located in each switchroom, which will annunciate on the control room operators screen. CO_2 fire extinguishers will also be provided inside all switchrooms.

18.6.5 Non-Production Waste Management

18.6.5.1 Sewage Treatment

Separate packaged sewage treatment plants (STPs) will be provided to process daily sewage waste from the following areas:

- Process Plant and MSA facilities - sewage piped to a common STP facility located between the process plant and exploration camp. Grey water effluent is proposed to be pumped back to the tails hopper in the process plant.
- Gendarmes Barracks and Main Gatehouse area STP, with grey water effluent discharged to a local spray field or used for local irrigation purposes.
- Permanent Accommodation Camp STP, with grey water effluent discharged to a local spray field or used for local irrigation purposes.

The STPs are likely to utilise Moving Bed Bio Reactors (MBBR) technology and consist of intake screens, equalisation tanks, biological reactors, clarifiers, and disinfection system.

Septic tank systems will be installed at smaller remotely located facilities including the dog kennels, airstrip, explosives storage area, WHD pumping station and remote security guardhouses. When required, a honey sucker will be used to empty tanks periodically, with the contents of the honey sucker being emptied into one of the aforementioned sewage treatment plants.

18.6.5.2 General Waste Management

A diesel-fired waste incinerator facility will be provided within the Plant high security fenced area. The incinerator is located within the Plant area to minimise the risk of gold theft associated with the movement of waste materials outside of the high security Plant area. The incinerator will be used to dispose of medical and general combustible wastes. Ash from the incinerator will be co-disposed with tailings.

A waste management facility and salvage/recyclable yard will be established on site for the storage and short-term management of waste materials.

Waste streams will be managed as outlined below:

- Non-Hazardous Solid Waste:

  Non-hazardous waste will be disposed of on site by either incinerating or storing within a permitted general waste disposal pit, close to/or adjacent to the waste dumps.

- Hazardous Waste:

  A temporary hazardous waste storage facility will be established on site with deposits controlled and logged. Hazardous waste will be taken off site to a licensed hazardous waste disposal facility.

- Radiation Sources:

  Radiation sources will be stored in a separate concrete building. The disposal of used sources will be the responsibility of the OEM supplier.

- Oil and Lubricants:

  Used oil will be temporary stored in aboveground steel tanks with secondary containment. The disposal of used oils and lubes will be the responsibility of the contracted fuel supplier.

- Chemicals:

  Chemical spills and containment materials will be collected in suitable sealable containers/drums and temporarily stored in the hazardous materials storage area until they can be removed from site by an approved disposal contractor.

- Recyclable/Reusable Materials:

  A salvage/recyclable yard will be developed for receiving and sorting of recyclable/reusable materials. This yard will consist of a fenced area, with separate areas for the sorting and storage of different types of materials (e.g. metal, wood, HDPE).

- Wood Pellets:

  Wooden pellets may potentially be provided to the local community for fuel or use.

- Food Wastes:

  Food wastes will be disposed of in the general waste pit over the short-term, and over the longer-term, these wastes may be used as part of ESG initiatives (i.e. pig farming).

18.6.6 Communication Systems

18.6.6.1 Country Overview

CI communication infrastructure available to the mine, both fixed and mobile, is discussed in Section 5 of this Report.

18.6.6.2 Site Infrastructure

Endeavour will engage with a local mobile service provider to connect the Site, with CIs fibre backbone via a Microwave link.

For the Plant and general offices, internal communications and IT services will be distributed via a site-wide high-capacity fibre optic network. The backbone of the system will be single mode fibre optic distributed throughout the site via a fibre optic cable (OPGW) run along the overhead power lines. This fibre network will be interconnected from the Plant's network panels to network switches installed at remote facilities, including; the Permanent Accommodation Camp, Gendarmes Barracks, and Main Security Gatehouse. The services that will use the common fibre optic backbone include:

- corporate local area network (LAN) including telephony (voice over internet protocol – VoIP);
- plant control system; and
- CCTV/security services.

A backup radio link will be deployed from the Main Administration Offices to the camp communications tower in case of a failure on the optic fibre link.

Network connection ports and Wi-Fi coverage will be provided for infrastructure buildings. Corporate servers, network switches and a firewall will be installed onsite to support the users locally with a VPN link to any remote central office as required. Staff workstations/laptops will be provided along with required software and office equipment such as docking stations, monitors, photo-copiers, and cabling.

The IT server infrastructure will consist of a main datacentre and storage at the Administration Building, plus a replication link to the MSA Offices IT Room for storage redundancy.

On-site general radio communication will be undertaken via hand-held radios, with a central radio control located within a communications control centre. Communication towers will be provided to support the site-wide radio communications network. Heavy and light vehicles will be equipped with radios for site communications.

18.6.7 Process Control System

The general control philosophy for the Plant and process related infrastructure will be one with a high level of automation and remote control. Process Instrumentation will be provided to measure and control key process parameters (the 'Process Control System' or 'PCS').

The main plant control room, located in the Plant office complex, will house two PC based operator interface terminals (OIT). Two servers will also be located here to act as the control system supervisory control and data acquisition (SCADA) servers in a redundant configuration. The control room is intended to provide a central area from where the plant is operated and monitored and from which the regulatory control loops can be monitored and adjusted. All key process and maintenance parameters will be available for trending and alarming on the PCS.

The PCS that will be used for the Plant and process infrastructure will be a programmable logic controller (PLC) and SCADA based system. The PCS will control the process interlocks and PID control loops for non-packaged equipment.

Vendor supplied packages will typically use vendor standard control systems throughout the project. Standardisation of hardware will be implemented as far as possible during the tendering and procurement stages. Negotiations will be undertaken with the equipment vendors to maximise the centralisation of control on the PCS with controls for vendor packages being either programmed into the plant PLCs or inputs to and outputs from the field controllers being duplicated on the OIT with minimum local set-point input and adjustment in the field. General equipment fault alarms from each vendor package will be monitored by the PCS system and displayed on the OIT. Fault diagnostics and troubleshooting of vendor packages will be performed locally.

18.7 Buildings, Stores, Workshop and Ancillary Facilities

18.7.1 General Infrastructure and Plant Facilities

General infrastructure/plant/mine buildings, stores, workshops and ancillary facilities required to support the mine are summarised in Table 18-33 and Table 18-34, and detailed more fully in the DFS report, Appendix 9.2 (Lycopodium, 2022a; Lycopodium, 2022b).

Table 18-33: General Site Infrastructure Buildings (Figure 18-3, Section 18.1)

Building Name	Description
Main Administration Offices	Prefabricated flatpack buildings catering for up to 150 personnel plus a large training/meeting room. Office facilities will include air-conditioned reception area, meeting rooms, small kitchen facilities, ablutions, stores, plus a mix of open plan and closed offices.
Clinic/First Aid and Emergency Response Buildings	Prefabricated flatpack buildings. The medical clinic will include air-conditioned reception, waiting area, pharmacy, examination rooms, isolation rooms, inpatient rooms, medical laboratory, stores, ablutions and ambulance bay. An adjacent emergency response building will include additional office space for the management and coordination of emergency response services.
Main Warehouse	Steel framed building with roof sheeting, wall cladding, motorised roller doors and concrete floor slab. Central warehouse facility will include open store area, warehouse racking, air-conditioned dispatch/receivals office, warehouse offices, meeting room and ablutions. A separate warehouse annexe constructed using shipping containers and dome shelter will also be provided for storing heavy lift insurance spares. A fenced laydown yard will be provided at the main warehouse facility.
Light Vehicle Workshop	Steel framed prefabricated building including; covered vehicle maintenance area, enclosed workshop space, air-conditioned office, breakroom, ablutions and maintenance stores.
Airstrip Arrival/Departure Building –	Prefabricated flatpack building providing an air conditioned waiting space for travellers on arrival and departure, plus a separate ablution block facilities.
•Social Performance Offices	Prefabricated flatpack building located near the main gatehouse within the perimeter fence and catering for 30 personnel. Office facilities will include air-conditioned reception area, meeting rooms, small kitchen facilities, ablutions, stores, plus a mix of open plan and closed offices.
Main Entrance Security Gatehouse	Blockwork building construction including boom gates, concrete apron, turnstyles, entry counter, air-conditioned search rooms, offices and a separate ablutions block.
•Security Command Posts/Guardhouses	Prefabricated flatpack buildings located at various security access control points around site. Guardhouse facilities include air-conditioned small office space for up to three guards, ID check/issue room, video induction room, ablutions and store room.
Security Control Centre	Prefabricated flatpack building located at the Permanent Accommodation Camp. Facilities will include air-conditioned open plan and closed offices, control room, meeting room, server room, workshop, storage and ablutions.

Table 18-34: Plant Infrastructure Buildings (Figure 18-3, Section 18.1)

Building Name	Description
Plant Security Gatehouse and Change Room	Blockwork building construction located at the entrance to the high security process plant area. Facility will include guard counter, turnstyles, search rooms, office, storage, ablutions, showers, change rooms and laundry.
Plant Offices and Control Room	Prefabricated flatpack building catering for up to 30 personnel. Building facilities will include airconditioned reception area, meeting rooms, small kitchen facilities, ablutions, stores, plant control room plus a mix of open plan and closed offices.
Plant Diner	Prefabricated flatpack building for on-site messing of plant operations personnel. Food preparation to be carried out at the permanent accommodation camp
Plant Ablutions	Prefabricated flatpack buildings at various locations around the process plant area.
Plant Laboratory	Blockwork building construction with adjoining steel framed sample shed. Lab facilities will include air-conditioned environmental lab, wet lab, bullion room, assay room, sample prep room, metallurgical lab, exhaust system, stores, offices, break room and separate ablutions block.
Plant Workshop	Steel framed building with roof sheeting, wall cladding and concrete floor slab. Workshop facilities will include main workshop with 10 t overhead travelling crane spanning 14.8 m, welding area, oil depot, tool crib, electrical workshop, storage, air-conditioned offices, meeting room and ablutions. A fenced laydown yard will also be provided at the plant workshop.
Reagent Stores	Two steel framed buildings with roof sheeting, wall cladding and concrete floor slab with floor sumps for sump pumping and internal concrete bund walls for separating reagents stores.

18.7.2 Mine Services Area Facilities

The Mine Services Area (MSA) will be developed by the appointed Mining Contractor, in consultation with Endeavour/SML. Facilities provided are summarised in Table 18-35.

Table 18-35: MSA facilities (Figure 18-4, Section 18.1)

Building Name	Description
Mining Offices	20 personnel including meeting rooms, kitchen and ablutions.
Training facility	Mining training building and simulator.
Canteen	120-person seating capacity
Changes rooms	Lockers, showers and ablutions
Mine Warehouse and Stores	
Mine Workshops	Including: • Heavy vehicle mine workshop including maintenance area (4 bays), welding area (2 bays), machines repair area (3 bays), components repair area, oil storage area and greases area. • Oil & lube area • Tyre change area. • Wash bay
Heavy vehicle fuel bays	Fuel pumped from the nearby bulk fuel storage facility (Section 18.6.3)
Vehicle laydown area	Including: • Heavy vehicles • Light vehicles
Laydown yards	Including: • Waste • containers

18.7.3 Emulsion Plant and Explosives Storage

An emulsion plant and explosives storage facility will be located within a locally fenced and secured compound at the southeast end of the Site, providing direct road access to the mine and MSA, whilst ensuring suitable blast separation distances to protected works facilities. The explosives facility will include:

- Emulsion plant for the on-site production of an emulsion for mine blasting activities, including chemical storage, fuel storage, water storage, gassing agent, boiler, ammonium nitrate storage and emulsion storage.
- Explosives magazines for storing detonators and explosives surrounded by earth mounding for secondary blast containment. Magazines will be separated from the emulsion plant to avoid potential knock-on blast effects.
- Small office, crib room, guardhouse and ablution facilities for operators and security personnel working at the facility.

Preliminary drawings for the emulsion and explosive storage facility from an equivalent Endeavour operation are presented in Figure 18-16 and Figure 18-17, respectively.

DFS capital and operating costing for the facilities and operations is based on an outsourced business model approach (facilities and operation), with costs used based on an equivalent Endeavour operation in CI.

Services/utility requirements whilst minor, have not been defined for the DFS.

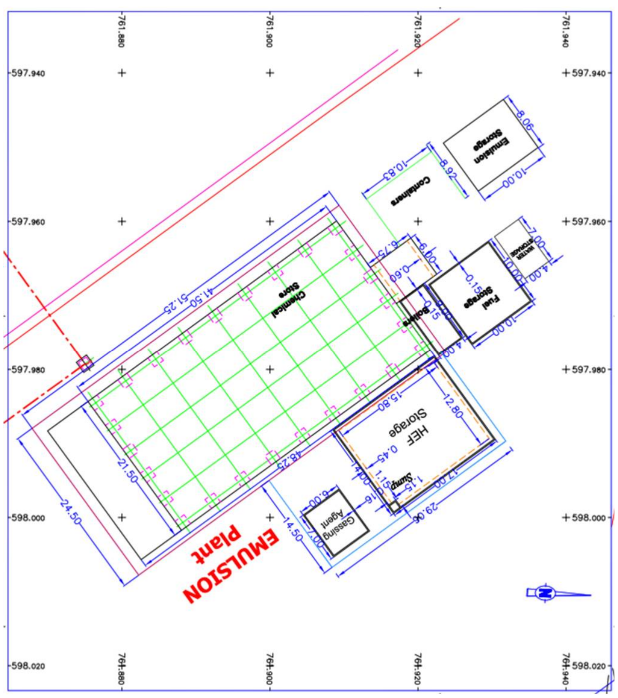


Figure 18-16: Indicative Emulsion Facility

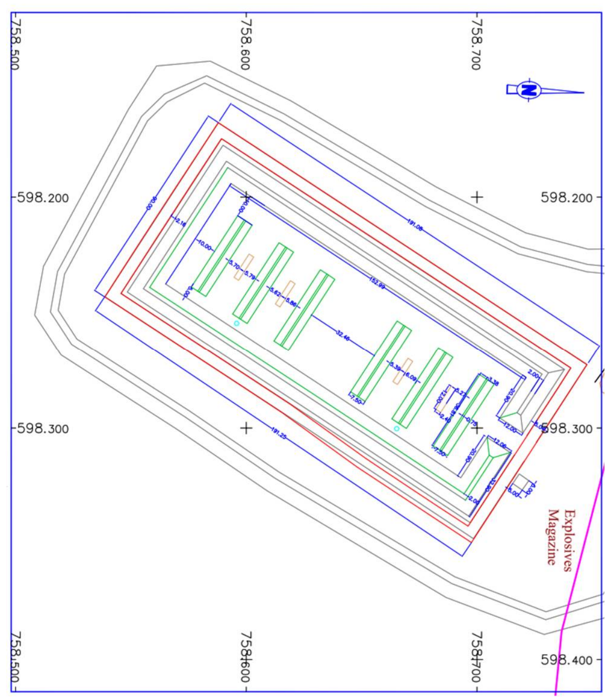


Figure 18-17: Indicative Explosive Facility

18.7.4 Site Accommodation

18.7.5 Construction Camp

A construction camp catering for 324 personnel will be installed using prefabricated flatpack buildings, which will be repurposed post construction to form part of the permanent accommodation camp. The construction camp will be located at the permanent accommodation camp compound at the eastern end of the mine site within the main perimeter fence line. The camp compound will be fenced with a security gatehouse controlling access into the area.

A large proportion of the construction workforce, including unskilled labour and trades, will be sourced from the local surrounding areas and bussed in on a daily basis.

A starter construction camp will initially be installed to accommodate 38 construction personnel prior to the construction camp completion. The starter camp will predominantly consist of ensuite accommodation rooms (28 of 38 rooms), plus laundry, shower block and ablutions, kitchen/diner, camp management offices, first aid clinic, genset power supply, lighting, potable water treatment plant and sewerage treatment plant. The majority of the starter camp facilities are planned to be relocated to the Gendarmes Barracks once the construction camp is operational.

The construction camp will include ensuite accommodation rooms, kitchen and dining, recreational gymnasium, laundry facilities, camp management offices and first aid clinic. Supporting camp services will include genset power supply, lighting, potable water treatment, water-based fire suppression system and sewerage treatment.

18.7.6 Permanent Accommodation Camp

The Permanent Accommodation Camp will have capacity to accommodate up to 340 operations staff on a single basis, utilising the repurposed construction camp facilities. Accommodation at the Permanent Camp will be prioritised to management and operations personnel who are not sourced from the local towns.

The selected location for the Permanent Accommodation Camp is approximately 4 km east of the plant site and approximately 8 km by road. In addition to the accommodation facilities provided from the repurposed construction camp, additional blockwork buildings will be constructed for the General Manager's residence and VIP accommodation. All buildings will be single storey. The accommodation camp facilities are summarised below:

- Accommodation units (single bedroom) with ensuites for up to 324 staff – prefabricated flatpack buildings installed as part of the construction camp.
- General Manager's residence and VIP accommodation units with ensuites for 16 staff – blockwork building construction.
- Restaurant (dry mess) with dining hall, kitchen, food storage and preparation area, dish washing area and catering staff facilities.
- Recreational facilities including gymnasium, bar, TV area, sports courts and soccer pitch.
- Laundry facilities.
- Camp management office facilities.
- First aid clinic.
- Gatehouse for controlling access into the fenced camp area.

18.7.7 Dog Kennels

Dog kennels for 12 dogs and accommodation for a canine caretaker (two ensuite rooms) will be provided for security management purposes. Containerised facilities will also be provided for storage purposes and an office/duty room. These facilities will be located near the airstrip road turnoff approximately 1 km north of the Permanent Accommodation Camp. This fenced compound will include power, potable water, and sewage management services.

18.7.8 Gendarmes Barracks

A Gendarmes Barracks will be fenced separately and located just outside the main gatehouse entrance to site. It will provide housing for up to 48 gendarmes as required and will include; basic kitchen, dining, laundry, and recreational facilities. The barracks will be constructed using prefabricated flatpack buildings, with the majority of the facilities relocated from the starter construction camp. Supporting services such as power, water and sewage handling will be provided at the barracks.

18.7.9 Exploration Camp

An exploration camp was established early on site to support exploration and early works activities for the Project. It is proposed that this camp remains in the same duty throughout construction and operations. Permanent power and communications will be distributed to this camp via overhead power lines as part of the Project execution phase.

18.8 Mine and Production Wastes

The design basis and the layout requirements for the tailings storage facility and the mine waste rock dumps are discussed in Sections 18.8.1 and 18.8.2, respectively. Supporting environmental information is presented in Section 20.

18.8.1 Tailings Storage Facility

18.8.1.1 Design Objectives

The design objectives for the TSF are:

- permanent and secure containment of all solid waste materials (tailings) generated by the process plant over the LoM;
- maximisation of tailings densities using subaerial deposition;
- removal and reuse of free water as far as practicable;
- reduction and control of seepage;
- excess storage capacity to retain ANCOLD-prescribed design storms and annual rainfall sequence, including containment of runoff from upstream catchments;
- rapid and effective rehabilitation;
- ease of operation;
- an effective monitoring network for environmental and safety control, comprising; embankment piezometers, survey pins, and groundwater bores; and
- effective rehabilitation and closure.

18.8.1.2 Design Basis/Criteria

The TSF has been designed to ANCOLD guidelines (ANCOLD, 2019), whilst the design criteria for the TSF is summarised in Table 18-36. The detailed design of the TSF will incorporate a review of the GISTM (Global Tailings Review, 2020) guidelines, which is not expected to impact the current TSF design.

An ANCOLD consequence category of 'High B' was determined for the TSF on the basis of a potential Population at Risk (PAR) in the range of '≥10 to <100' and a Severity Level of 'MAJOR', on the basis of significant damage to the business and public health. An ANCOLD environmental spill consequence category of 'Significant' was determined on the basis of a potential PAR in the range of '<1' and a Severity Level of 'MAJOR'.

Table 18-36: Tailings Storage Facility Design Criteria and Specifications (KP, 2022b)

Description	**Value**
Design	
TSF Consequence Category" Dam Spill Consequence Category: TSF Stormwater Storage Capacity (ANCOLD, 2019) TSF Emergency Spillway: • Spillway capacity • Erosion protection TSF Closure Spillway: • Spillway capacity • Erosion protection Contingency Freeboard (ANCOLD, 2019) • Wave run-up • Additional freeboard Earthquake Loading (KP, 2021a) • Operating • Final	High B Significant Average supernatant pond superimposed with greater of: • 100-year ARI (Average Recurrence Interval) wet year peak pond volume • 100-year ARI, 72 hr flood • 10-year ARI wet season runoff (assuming 100% runoff and no evaporation) PMF (Probable Maximum Flood)/critical duration 100-year ARI/critical duration PMF/critical duration 100-year ARI/critical duration 10-year ARI Wind (allowance of 0.5 m) 0.3 m Operating Basis Earthquake (OBE) - 1000-year ARI earthquake (0.026 g) Safety Evaluation Earthquake (SEE) - 5000-year ARI earthquake (0.054 g)
Operations	
Capacity • Final • Starter	 41.0 Mt (db) of dry tails. 11.0 Mt (db) of dry tails (36 months initial capacity).

Table 18-36: Tailings Storage Facility Design Criteria and Specifications (KP, 2022b)

Description	Value
Design Production Basis	Mine Schedule (TSF design): 12I • 3.0 Mt/a (db) for first 12 months • 4.0 Mt/a (db) normal operation • 2.0 Mt/a (db) for final 12 months • Total Capacity required 41.0 Mt (db) Mine Schedule (Current): Sc13k (SRK, 2022) • 2.2 Mt/a (db) for first 12 months • 3.8 Mt/a (db) normal operation • 3.2 Mt/a (db) for final 12 months • Total Capacity required 47.7 Mt (db)
Slurry Characteristics (KP, 2021d) • Beach slope*1 • % Solids (w/w) • P_{100} • P_{80} • Tailings density – Stage 1 – Final	 150H:1V 45% 0.3 mm (Fresh)/0.3 mm (Oxide Blend) 111 µm (Fresh)/107 µm (Oxide Blend) 1.41 t/m^3 1.44 t/m^3
Stability Factors of Safety: • Long-term undrained • Short-term undrained: – potential loss of containment – no potential loss of containment • Post-seismic	 1.5 1.5 1.3 1.0 to 1.2

18.8.1.3 Test work

Tailings physical testing undertaken by Knight Piésold (KP, 2021d) is summarised below, whilst the geochemical testing is summarised in Section 20 (KP, 2021c).

A fresh ore composite sample 'CIP (Fresh)' described as fresh ore tailings, and an oxide blend composite 'CDO (Blend)' described as 35% oxide/65% fresh were tested. The tailings testing results were incorporated into the water balance modelling and the TSF design, by interpolating tailings properties over the LoM schedule.

The physical properties of the tailings tested, are as noted below:

- CIP (Fresh) consisted of 36% sand, 61% silt and 3% clay. The testing indicates the material is non plastic sandy silt with trace clay. The sample had a P_{80} of 111 µm.
- CDO (Blend) consisted of 33% sand, 63% silt and 4% clay. The testing indicates the material is non-plastic sandy silt with trace clay. The sample has a P_{80} of 107 µm.
- The grading curves indicate the samples fall inside the boundary for potentially liquefiable soils, and therefore liquefaction of the tailings mass was considered in the stability assessment for the TSF.

- CIP (Fresh) tailings sample exhibited very rapid settlement and achieved a moderately high dry density from settlement before air drying or consolidation. The sample achieved a maximum dry density from air drying of 1.46 t/m^3 (adb). The dry density is considered moderate for gold tailings.
- CDO (Blend) settled quickly and achieved a moderately high dry density from settlement before air drying or consolidation. The sample achieved a maximum dry density from air drying of 1.51 t/m^3 (adb). The dry density is considered moderate for gold tailings.

Based on the aforementioned testwork it was concluded that; the rate of supernatant release for all samples sample was quick and reached typical dry densities with a good increase due to drying and consolidation; and, assuming that the facility is efficiently operated, it is estimated that the average settled density for the tailings mass will be approximately 1.44 t/m^3 (based on density and consolidation modelling completed).

18.8.1.4 TSF Dam Breach Assessment

A dam breach assessment based on ANCOLD guidelines (ANCOLD, 2019) was carried out for the Lafigué TSF (final configuration, Figure 18-19) to estimate the Population at Risk (PAR), business risk and environmental impact in the event of a dam failure (KP, 2021e). The likelihood of failure is not considered in the ANCOLD Consequence Category Assessment.

A tailings inundation estimate was generated for the dam break scenario for the TSF. Published methods were used for the modelling.

In the event of dam failure, significant volumes of water are expected to be mobilised. Two distinct flow regimes are anticipated, a quicker outflow of supernatant and stored water, followed by a slower prolonged outflow of solids

The failure of the TSF embankment at the maximum embankment height is considered to have the following impacts (Figure 18-21):

- The WHD downstream of the TSF will be inundated. The stored water will be displaced, and report to the N'zi River.
- Based on existing available topographical data, it is anticipated that the majority of the tailings flow will be attenuated within 5.0 km downstream of the TSF, where flatter topography is encountered. It is noted that tailings mobilised during the release of the supernatant pond will progress over time to the N'zi River.
- It is likely that the tailings solids (carried by supernatant water outflow) will enter the N'zi River. It is noted that the N'zi River is a tributary of the Bandama River, the largest river in Côte d'Ivoire.
- It is considered probable that the tailings solids and supernatant water will have adverse impacts on the groundwater and soil quality within the inundated area.
- A number of access roads are at risk of being cut off in the event of failure.
- Review of satellite imagery indicate that some local agriculture downstream will be affected by a TSF failure.
- A number of local settlements have been identified downstream of the TSF near the N'zi and Bandama Rivers, which may be at risk in the event of failure. The major settlement identified is Gboly Carrefour, population 1 006 (KP, 2021e).
- Both tailings solids and contaminated water are likely to be released from the current mining lease.
- If failure was to occur during a TSF raise, construction workers will be at risk.

The key conclusions from the Dam Breach Assessment for the Site TSF are summarised below. In the unlikely event of a catastrophic embankment failure:

- A PAR of '≥10 to <100' is expected for the assumed failure case.
- The Severity Level would be 'Major', primarily due to the anticipated business and public health impacts if tailings were to impact local communities downstream.
- The ANCOLD 2019 Dam Failure Consequence Category would be 'High B'.
- The ANCOLD 2019 Environmental Spill Consequence Category would be 'Significant'.
- The Dam Breach Assessment for the TSF will be updated to reflect the current mining schedule during the detailed design phase.

18.8.1.5 Design Summary

The TSF design was completed to a DFS level of development by Knight Piésold (KP, 2022b) and is summarised in this section. The TSF, a cross-valley storage facility built using multi-zoned earth fill embankments (Figure 18-20) will be constructed in nine phases over the LoM (Table 18-37). A layout of the TSF in year one and at the end of the LoM (final) is illustrated Figure 18-18 and Figure 18-19, respectively.

Table 18-37: Staged Embankment Construction (KP, 2022b)

Stage	Tailings Storage (Cumulative) (Mt) (db)	TSF Embankment Elevation (m RL)	Maximum TSF Embankment Height (m)	Approximate Basin Area(ha)	Total Embankment Volume (m^3)
1	11.0	276.6	27.6	113	1 081 000
2	15.0	279.0	30.0	125	1 376 000
3	19.0	281.5	32.5	136	1 737 000
4	23.0	283.5	34.5	144	2 070 000
5	27.0	285.6	36.6	152	2 472 000
6	31.0	287.6	38.6	159	2 910 000
7	35.0	289.4	40.4	166	3 362 000
8	39.0	291.3	42.3	172	3 905 000
9	41.0	292.2	43.2	175	4 426 000

Table 18-37 notes: Stage 1 embankment designed for 36-month storage capacity

Stage 1 is designed for 36 months of storage capacity. The layout of the Stage 1 TSF is shown in Figure 18-18. Subsequently, the TSF will be constructed in annual downstream raises to suit storage requirements; however, this may be adjusted to biennial raises to suit mine scheduling during operations.

The TSF is designed to accommodate a total of 41 Mt (db) of tailings based on the Sc12I mining schedule. It is noted that the current mining schedule (Sc13k) requires an additional 6.7 Mt of tailings; however, the additional tonnage occurs during the final years of operation. Subject to embankment stability checks, it is estimated that the TSF can be expanded to approximately 80 Mt (db) before impacting other site infrastructure.

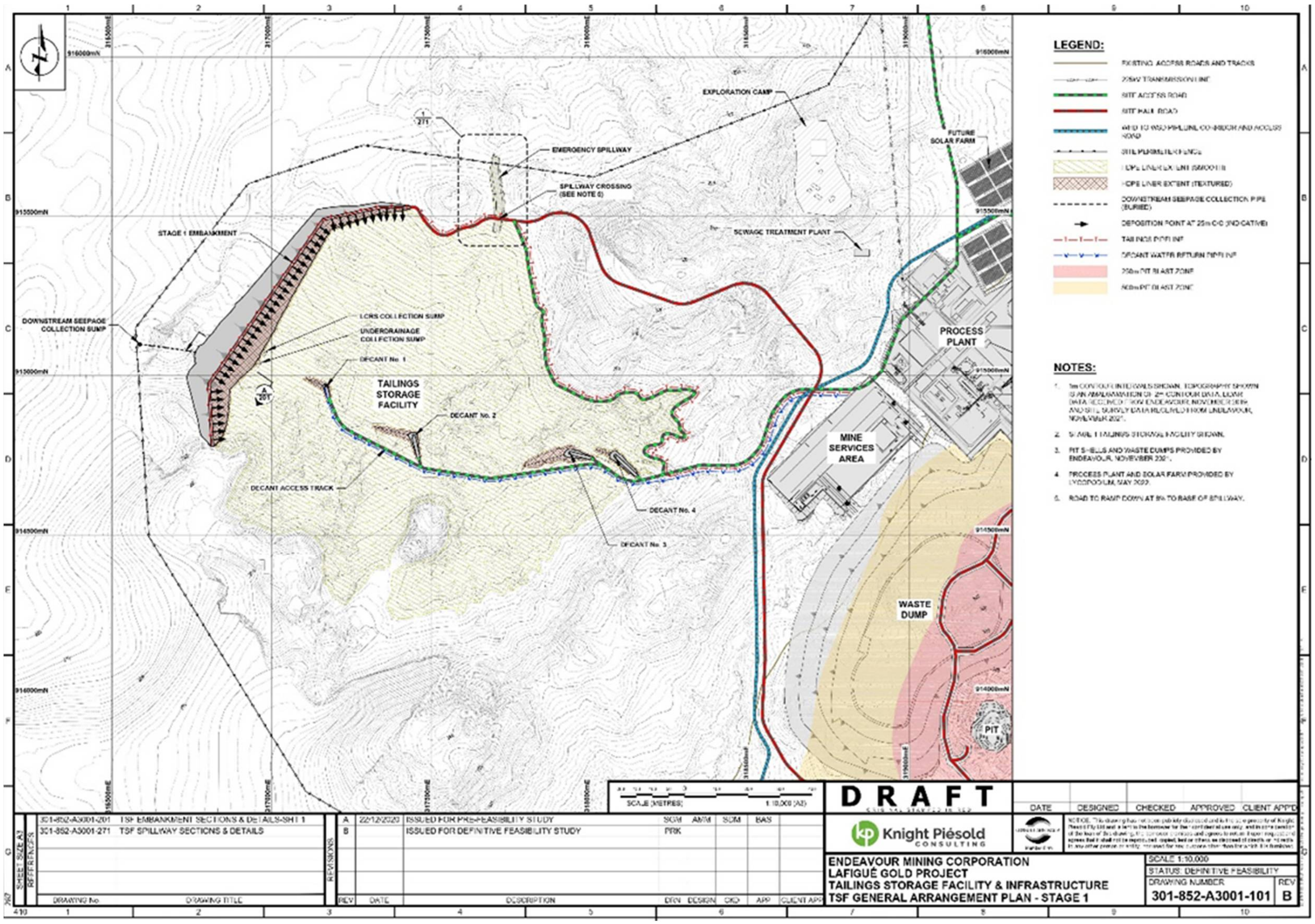


Figure 18-18: TSF Stage 1 Layout (KP, 2022b)

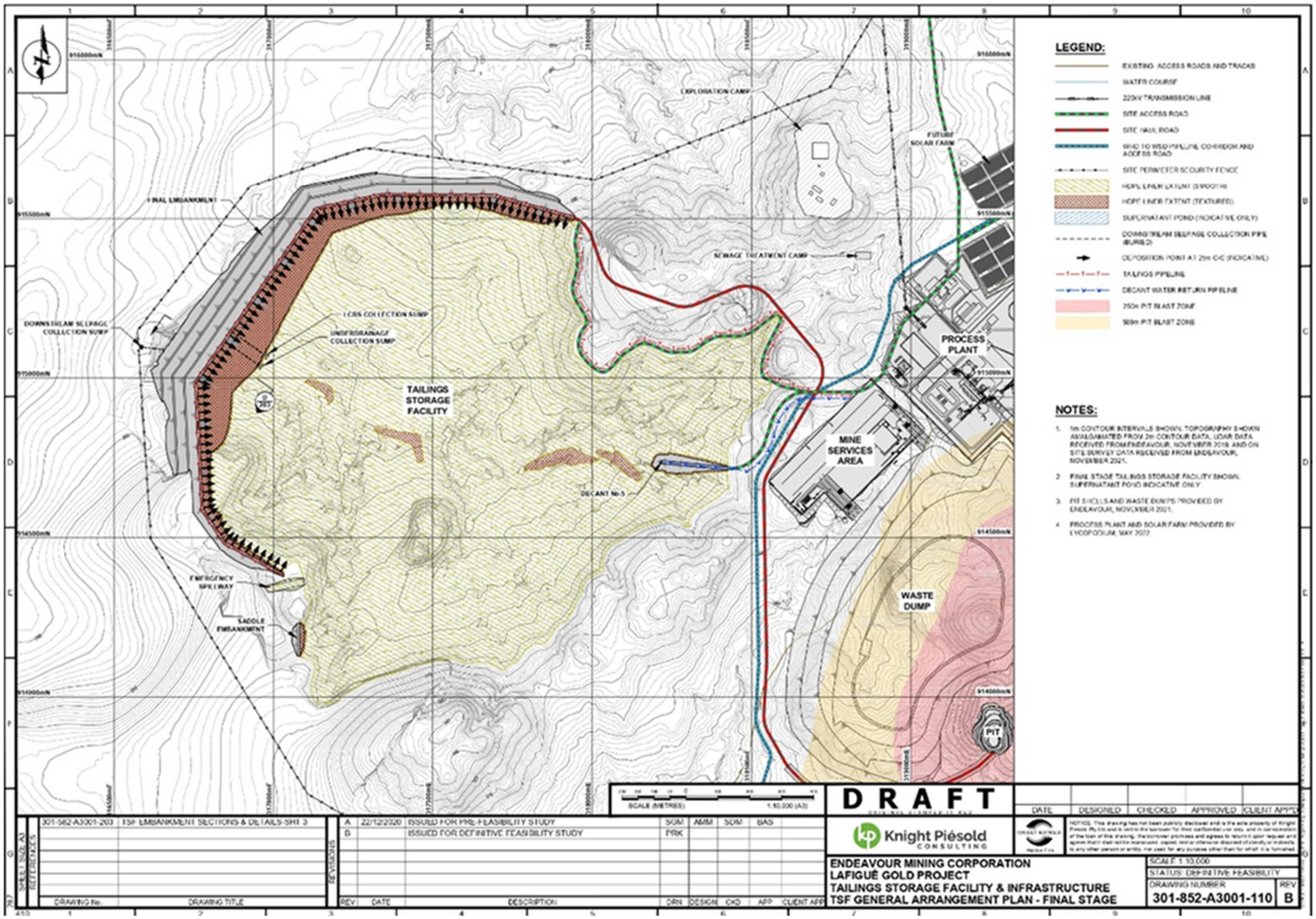


Figure 18-19: TSF Final Layout (KP, 2022b)

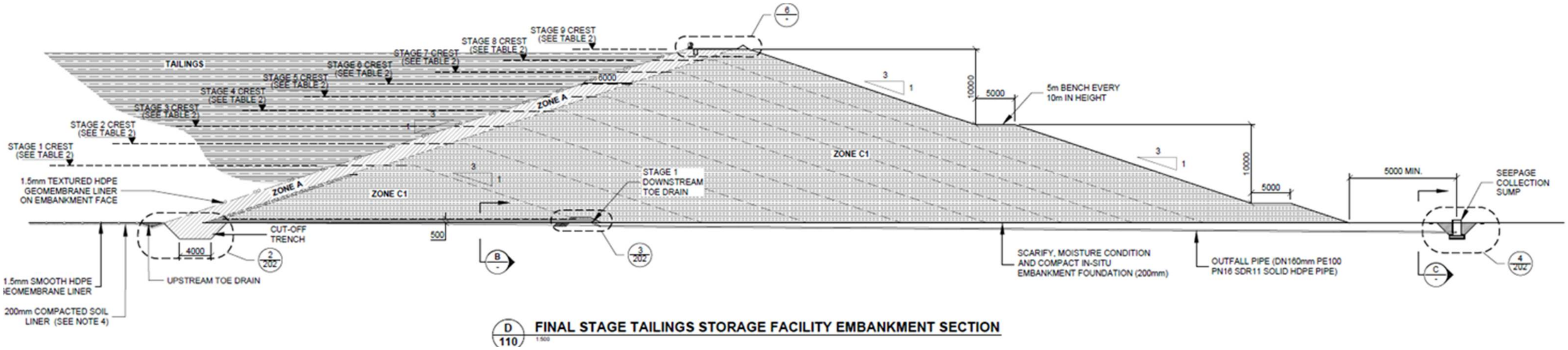
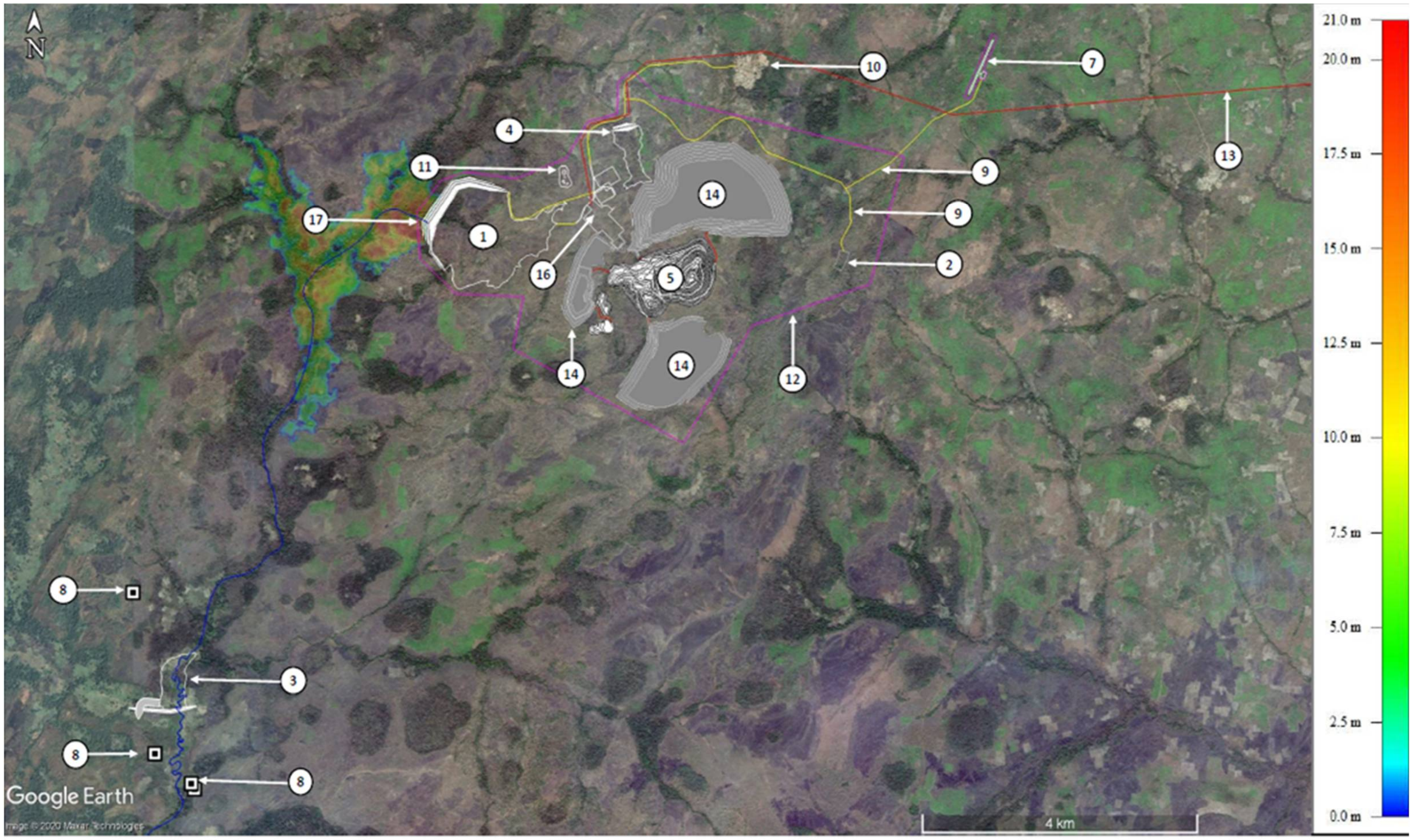


Figure 18-20: Typical Embankment Cross Section (KP, 2022b)



LEGEND									
1	TAILING STORAGE FACILITY	5	PIT (FINAL)	9	ACCESS ROADS	13	POWERLINE	17	ASSUMED FAILURE LOCATION
2	ACCOMODATION VILLAGE	6	HAUL ROAD	10	LAFIGUE VILLAGE	14	WASTE DUMP		
3	WATER HARVESTING DAM	7	AIRSTRIP	11	EXPLORATION CAMP	15	N'ZI RIVER / FLOW PATH		
4	WATER STORAGE DAM	8	LOCAL VILLAGE	12	FENCELINE	16	PROCESS AND MINE SERVICES AREA		

Figure 18-21: Dam Breach Inundation Extents (KP, 2021e)

18.8.1.6 TSF Design and Operating Parameters

The TSF embankment will have an upstream slope of 3H:1V to facilitate high density polyethylene (HDPE) geomembrane liner installation, an operating downstream slope of 3H:1V and a minimum crest width of 8 m. The final downstream embankment profile will consist of an overall slope of approximately 3.5H:1V, comprising a 3H:1V slope, with 5 m benches at 10 m vertical intervals. The typical embankment cross section is shown in Figure 18-20.

The embankment upstream face will be lined with textured HDPE geomembrane liner, to allow safe egress from the TSF for people and animals. A downstream seepage collection system will be installed within and downstream of the TSF embankment, to capture seepage from the TSF and return it onto the tailings beach (to report to the supernatant pond).

The TSF embankment comprises an upstream low permeability zone (Zone A), and downstream structural fill zone (Zone C1). A chimney drain (Zone F1) is included within the Stage 1 embankment cross section, as the embankment will be water retaining for the initial stages of operation. The requirement for a transition zone (Zone B) between the Zone A and Zone C1 will be determined during the operation as part of each raise design (and verified during each raise), based on the properties of each material.

It is envisaged that the mining fleet will place the embankment bulk fill material (Zone C1) on a continuous basis throughout operation, hauling directly from the Open Pit(s). A minimum structural fill profile (Zone C1) is required at the commencement of the construction period each year to facilitate the Zone A (and Zone B if required) material placement; however, additional fill can be placed earlier than required to suit the mining schedule. It is envisaged that a civil earthworks fleet will place Zone A (and Zone B if required) material during each raise construction, as space will be limited.

The stability of the TSF was assessed (KP, 2022b) to confirm the factors of safety against shear failure, considering; long-term drained conditions, short-term undrained conditions, and post seismic conditions in accordance with the requirements prescribed by ANCOLD guidelines (ANCOLD, 2019). The effect of pore water pressures on embankment stability was modelled during the stability assessment. Based on the slope stability assessment, the TSF will have satisfactory factors of safety based on the recommended minimum factors of safety by ANCOLD, and therefore should be stable as designed. The stability analyses indicate that during operation the embankment profile satisfies all requirements for operational minimum factors of safety. All stability models were completed with the minimum embankment profiling. During operation, embankments will be further buttressed by ongoing mine waste placement by mining.

Typical specifications for material types are summarised as follows:

- Zone A material shall be won from borrow to form the low permeability zone of the embankments (approximately 0.27 Mm^3).
- Zone B material (if required) shall be won from borrow to provide sufficient transition between the finer Zone A material and coarser Zone C1 material after conditioning and compaction.
- Zone C1 material shall be delivered to the embankment by the mining operation, levelled with a dozer and traffic compacted by loaded haul trucks on an ongoing basis during the operation. When material from the open pits is not available, or for areas not easily accessible by the mining fleet, Zone C material shall be won from borrow (approximately 4.1 Mm^3).
- Zone F1/F2 shall be clean sand/gravel drainage material supplied to a stockpile adjacent to the works (F1=26 000 m^3, F2 = 2600 m^3).

- Zone G shall be clean rock fill for decant tower surrounds, supplied to a stockpile adjacent to the works (~15 500 m^3).

A cut-off trench will be located beneath the upstream low permeability zone (Zone A) of the embankments. The cut-off trench will be excavated to extend through to competent low permeability foundation material and backfilled with low permeability fill (Zone A) to reduce seepage losses through the embankment foundations.

The TSF basin area will be cleared, grubbed and topsoil stripped, and a 200 mm thick compacted low permeability soil liner will be constructed over the entire basin, comprising either reworked in situ material or imported low permeability material. The compacted soil liner surface shall drain positively and be finished with a smooth drum vibratory roller to promote runoff. The area within the TSF basin will be lined with a 1.5 mm HDPE geomembrane liner, overlying the compacted soil liner. The decant tower areas will be lined with 1.5 mm textured HDPE liner to improve stability of the tower surround and causeway fill slopes. The TSF embankment upstream face will be lined with 1.5 mm textured HDPE liner, to improve safety during construction and operations.

The TSF design incorporates an underdrainage system to reduce pressure head acting on the basin liner system, reduce seepage, increase tailings densities, and improve the geotechnical stability of the embankments. The design of the underdrainage system takes advantage of the natural fall of the ground and thus minimal re-shaping of the basin will be required. The underdrainage system will consist of two interconnected drainage networks, namely the collector drains and the finger drains. Finger drains will be installed at approximately 100 m centres over the TSF basin area. The finger drains will consist of a draincoil pipe covered with drainage medium (Zone F1), wrapped in geotextile. The finger drains will connect into the collector drains. The collector drains will consist of a draincoil pipe contained within a trench, backfilled with drainage medium (Zone F1), double-wrapped in geotextile. The collector drains will be located either side of a shaped drainage trench, approximately 8 m wide. The collector drains will feed directly into the underdrainage collection sump. The underdrainage system drains by gravity to a collection sump located at the lowest point in the TSF basin.

The underdrainage sump will collect solution from the underdrainage network and collected solution will be pumped back onto the tailings beach by a submersible pump situated at the base of a riser pipe running up the embankment upstream face, with flows reporting to the supernatant pond for recycling back to the process plant. The sump will be backfilled with clean gravel material (Zone F2), overlain with drainage sand/gravel material (Zone F1) and wrapped in geotextile. Two HDPE riser pipes will run from the base of the sump on the embankment upstream face, to the crest elevation. The bottom of riser pipe (within the sump) will be slotted. The riser pipes will be located on the embankment face and ballasted with pipes filled with concrete and fixed with a steel brace. A submersible pump will be situated at the base of one of the riser pipes (slotted section), within the collection sump. The pump will operate with a level control. A pump sled will be installed to facilitate removal of the underdrainage pump. The second riser pipe will provide redundancy in the event of pipe failure.

In addition, a toe drain will be constructed along the upstream toe of the embankment. The toe drain comprises a draincoil pipe within a 'v-ditch' at the embankment toe, backfilled with drainage sand (Zone F1) and double-wrapped in geotextile. The toe drain will flow into the underdrainage collection sump.

A leakage collection and recovery system (LCRS) will be installed beneath the basin composite liner to reduce water pressure build-up on the HDPE liner. The LCRS is independent of all other seepage control components. The LCRS drains will consist of draincoil pipes contained within a trench beneath the collector drains, backfilled with drainage medium (Zone F2) (wrapped in geotextile), overlain by low permeability (Zone A) material cap beneath the basin compacted soil liner. The LCRS drains will feed directly into the LCRS collection sump. The sump will be backfilled with clean gravel material (Zone F2), wrapped in geotextile and covered with the basin soil liner and HDPE. Two HDPE riser pipes will run from the base of the sump on the embankment upstream face, to the crest elevation. The bottom of riser pipe (within the sump) will be slotted. The riser pipes will be located on the embankment face and covered with stabilised fill. A submersible pump will be situated at the base of one of the riser pipes (slotted section), within the collection sump. The pump will operate with a level control. A pump sled will be installed to facilitate removal of the LCRS pump. The second riser pipe will provide redundancy in the event of pipe failure. Solution recovered from the LCRS will be released to the top of the tailings mass via submersible pump, reporting to the supernatant pond.

A seepage collection system will be constructed within and downstream of the TSF Stage 1 embankment. The chimney drain (within the embankment cross section) will drain water to the embankment base, which will be transferred to 0.5 m wide by 0.5 m high sand drains (at 20 m spacing), which in turn will report to a toe drain constructed at the Stage 1 embankment downstream toe. The top elevation for the embankment chimney drain is governed by the expected maximum pond volume for a 100-year ARI annual rainfall sequence occurring in Year 1. The downstream embankment toe drain will comprise a draincoil pipe laid at the embankment toe, covered drainage sand (Zone F1) and overlain by a 200 mm thick layer of erosion protection material (Zone E). The downstream toe drain will capture and direct flow to the low point(s) of the downstream toe, at which point flow will be directed into an HDPE outfall pipe, finally reporting to a seepage collection sump outside of the final TSF embankment footprint. The seepage collection sump comprises a buried concrete tower structure.

Supernatant water will be removed from the TSF via submersible pumps located within a series of decant towers (4 No. towers in Stage 1 and 5 No. towers in total) located within the eastern valley of the TSF basin. The decant pump will be moved between towers as the supernatant pond migrates up the valley during the course of operation. Solution recovered from the decant system will be pumped back to the plant for re-use in the process circuit and to dilute tailings. Each decant tower will comprise a concrete base and slotted precast concrete sections. The decant tower will be surrounded by free-draining coarse rockfill (Zone G).

Under normal operating conditions with the TSF managed in accordance with standard operating procedures, the available stormwater storage capacity will be in excess of the design storm event volumes and no discharge from the TSF is expected. In the event that a storm event greater than the TSF design criteria occurs, that exceeds the available storage capacity during operation, rainfall and supernatant water which cannot be attenuated and stored within the supernatant pond will discharge from the TSF in a controlled manner via an engineered spillway into the N'zi river immediately downstream of the TSF. Discharge under these conditions is required to protect the integrity of the embankments from overtopping failure. An operational emergency spillway will be available during TSF operation to protect the integrity of the constructed embankments in the event of emergency overflow. The operational emergency spillway will be constructed as part of each embankment raise.

Tailings will be discharged into the TSF by sub-aerial deposition methods, using a combination of spigots at regularly spaced intervals from the TSF embankment. Deposition will occur from multiple spigots inserted along the tailings distribution line. The deposition location(s) will be moved progressively along the distribution line as required to control the location of the supernatant pond. During the final stages of operation, the deposition will be managed to push the supernatant pond to the closure spillway location.

To reduce the environmental impact of a tailings line (slurry and supernatant) failure, a pipeline containment trench will be constructed during Stage 1. The tailings pipeline will run along the TSF perimeter on the HDPE geomembrane liner (adjacent to the TSF perimeter access track) in between the Tailings and Decant Return Trench (TDRT) and the TSF embankment.

18.8.1.7 Monitoring

A total of two groundwater monitoring stations will be installed downstream of the TSF to facilitate early detection of changes in groundwater level and/or quality, both during the operating life and following decommissioning. The monitoring bore station consists of one shallow bore, extending to a depth of 10 m in the deep surface horizon, and one deep bore terminating in fresh rock (depth to be advised by Endeavour based on hydrogeology assessments completed at the site). The shallow bore is intended to detect any seepage from the TSF flowing within the surface sediment, whilst the deep bore will monitor the chemical composition of the groundwater. It is recommended that the boreholes are constructed before commissioning of the TSF to accumulate baseline data specific to the TSF location.

To inform stability assessments (KP, 2022b), standpipe piezometers will be installed in the TSF embankment to monitor pore water pressures at several locations within the embankments. The base of each piezometer will be located within the embankment fill to ensure that the phreatic surface within the embankment, as opposed to natural groundwater level, is being measured. Additional piezometers will be installed as the TSF embankments are raised, to monitor the development of the phreatic surface in the embankments.

Survey pins will be installed at regular intervals along the TSF embankment crest to monitor embankment movements and assess effects of any such movement on the embankment.

The TSF will undergo annual audits by the Engineer of Record, to ensure that the facilities are operating in a safe and efficient manner in accordance with the design intent.

18.8.1.8 Closure Summary

At the end of the TSF operation, the downstream faces of the embankment will have a slope of 3H:1V, with 5 m wide benches located at 10 m height intervals, for an overall slope profile of approximately 3.5H:1V. The profile will be inherently stable under both normal and seismic loading conditions and will provide a stable surface water drainage system and will allow for revegetation.

Rehabilitation of the tailings surface will commence upon termination of deposition into the TSF. The closure spillway will be constructed in such a manner as to allow rainfall runoff from the surface of the rehabilitated TSF to discharge via the closure spillway.

The TSF closure spillway will be excavated after the remaining supernatant water is proven to be suitable for release and during rehabilitation of the tailings surface subsequent to decommissioning. The closure spillway will be located through the eastern saddle, at the low point of the final tailings beach. The closure spillway will discharge into the existing drainage course downstream of the TSF. Upon closure, the TSF will be a fully water-shedding structure. The closure spillway will allow conveyance of probable maximum precipitation (PMP) storm events without any attenuation in the TSF.

The final soil cover for the tailings surface subsequent to decommissioning will be confirmed during operation based on ongoing operational tailings geochemistry testing results. The following cover design for the tailings beach has been adopted at this stage but is subject to ongoing testing and review:

- Mine waste capillary break (500 mm thick, 0.74 Mm^3).
- Low permeability fill layer (300 mm thick, 0.44 Mm^3).
- Topsoil growth medium layer (200 mm thick, 0.30 Mm^3, sourced from site stockpiles).
- The finished surface will be shallow ripped and seeded with shrubs and grasses (161 Ha).

18.8.2 Waste Rock Facilities

18.8.2.1 Waste Rock Dump (WRD) Design Criteria/Considerations

The basis of the Waste Rock Dump (WRD) design and positioning is based on: the Mine Plan (SRK, 2022); the waste rock geochemical and geotechnical parameters (Section 18.8.2.2 and 18.8.2.3 respectively); and

- material handling, haulage, and rehabilitation costs;
- environmental, community and visual impacts;
- topographical features;
- proposed mine infrastructure;
- resource sterilization; and
- rehabilitation requirements.

The fundamental principles of construction will be as per the Environmental Management Plan (EMP). The dump benches will be covered with a soil layer pad, docked and vegetated as soon as possible after dumping. The top surface of the dumps will be soil clad and graded back at approximately 1:200 to prevent ponding on the top cover.

The WRD management plan (the 'Plan') will be updated annually, to ensure that the construction of the WRD progresses smoothly. The Plan will address:

- all significant potential risks identified;
- progressive topsoil capture;
- changes to the original design;
- progressive rehabilitation as specified in the environmental management plan;
- results of environmental monitoring addressed where applicable; and
- compliance to all other commitments made in the environmental management plan.

18.8.2.2 Geotechnical Design Parameters

The WRD are anticipated to remain relatively stable over the long-term, with minor slope creep over an extended period. Potential environmental impacts resulting from individual batter failure of the dumps will be minimal due to wide catch berms. The parameters illustrated in Table 18-38 following, were used during the design process. It should be noted that a more detailed geotechnical assessment of waste characteristic study is required to ensure compliance with all relevant requirements.

Table 18-38: WRD Geotech Design Parameters

Description	Units	Value
Lift Hight	(m)	15
Berm Width	(m)	30
Batter Angle	(°)	35
Overall Slope Angle	(°)	20
Ramp Width	(m)	28
Ramp Gradient 1:10	(%)	10

18.8.2.3 Waste Rock Geochemistry

Based on test work undertaken (KP, 2021d), no issues are foreseen with respect to WRD contact water quality, and it's planned to discharge directly to the receiving environment, after the contact water has passed through the sediment control systems (Section 18.5.6). Waste rock geochemistry results are discussed in Section 20.

18.8.2.4 Waste Rock Dump Water Runoff

Sediment control systems (SCS) or dams will be constructed to lessen the sediment-laden runoff to the receiving environment (KP, 2022b). The positioning of the sediment control dams is illustrated in Figure 18-22, following.

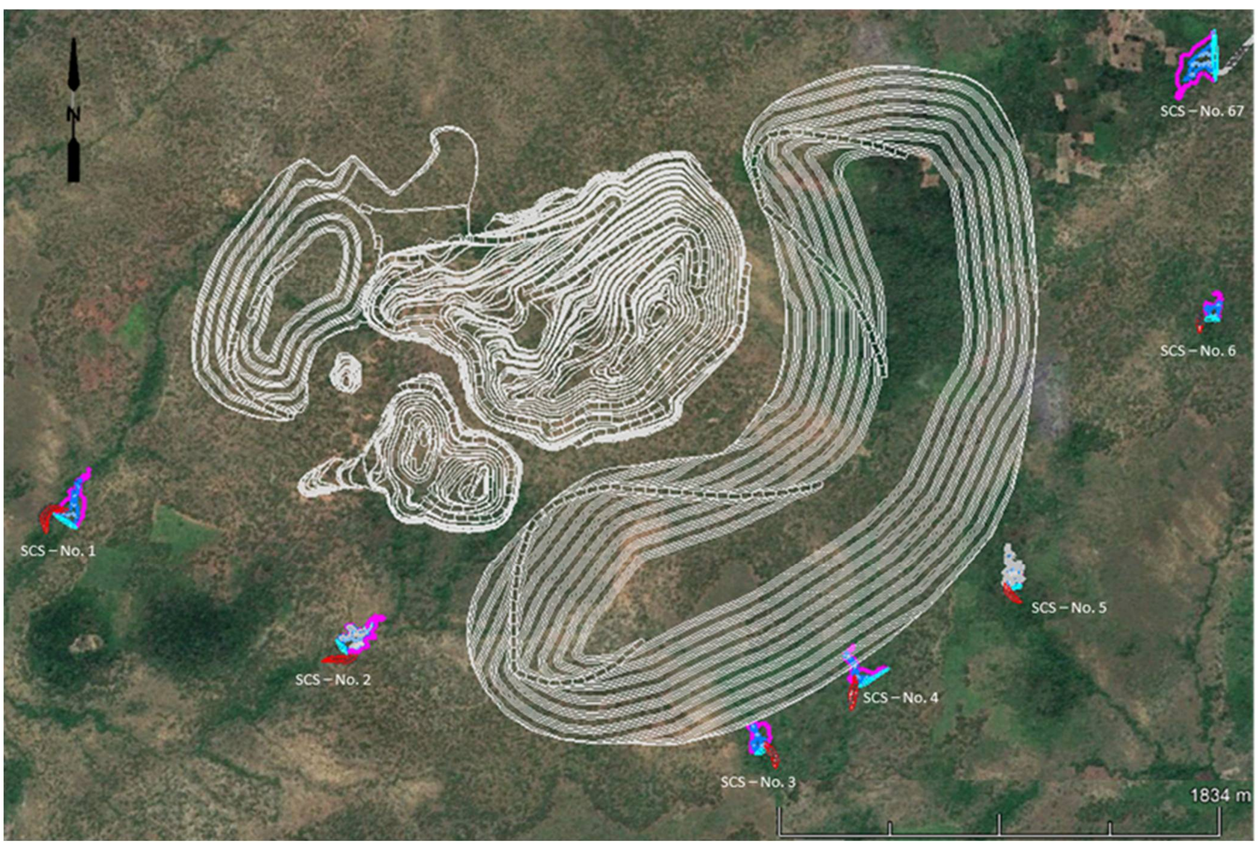


Figure 18-22: SCS Locations

18.8.2.5 Waste Rock Dump Design

Figure 18-23, following, illustrates the WRD locations over the LoM. The west dump is primarily for material from mining Stages 1 and 2, with the northern end currently planned for the RoM pad incorporating a skyway system for stockpile deposition and management. The central dump is the primary dump for the bulk of the waste. The WRD is split into two areas with access in the east and south. Waste is flagged for each location, focusing on optimising the haulage distances. Additional waste dump space is available in the north, but due to the undulating terrain and impact on haulage distances, most of the waste is mined from the south.

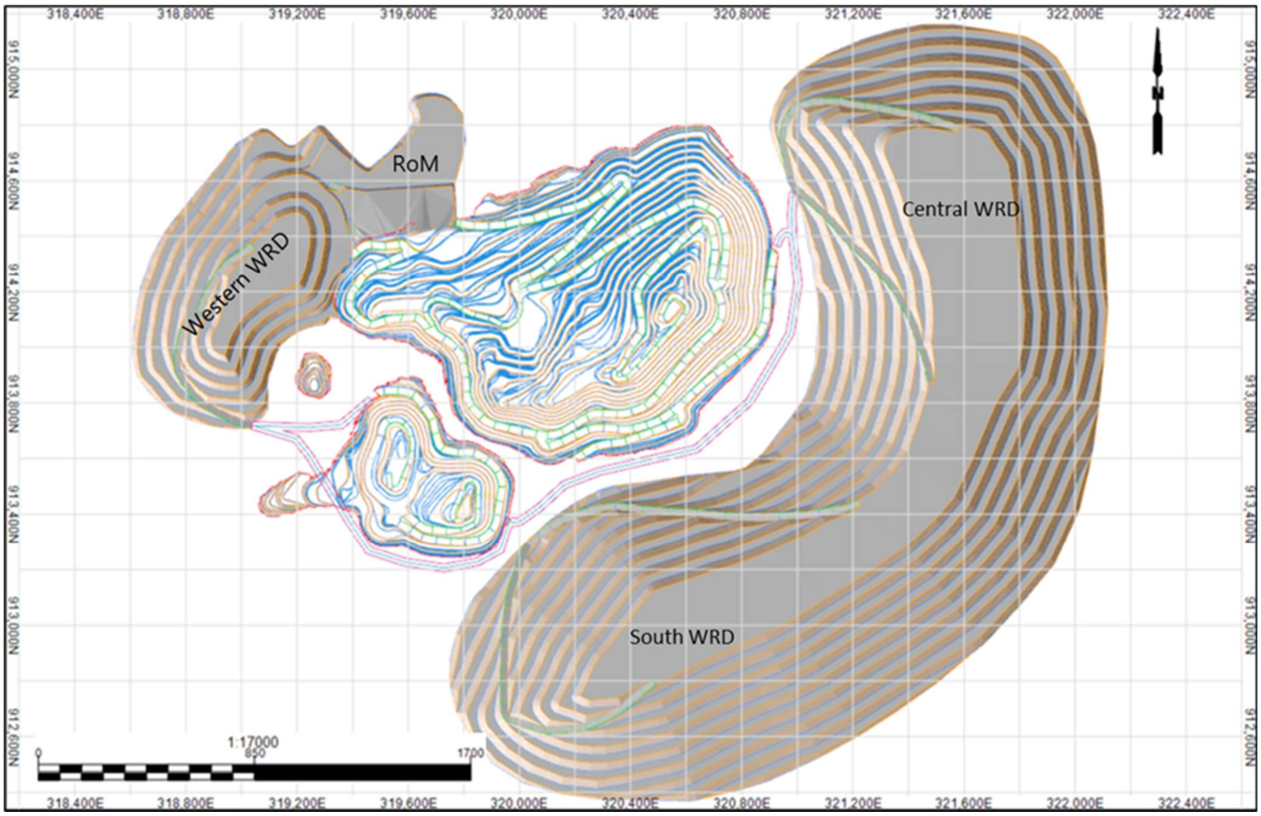


Figure 18-23: Waste Dump Locations

Table 18-39 illustrates the storage capacity requirements of the various waste dumps and the ROM pad. Over the LoM, waste comprises: 88% Fresh, 7% transitional, 4% saprolite and 1% laterite.

Table 18-39: Waste Dump Capacities

Location	Waste Tonnes Mined (Mt)	In situ Waste Volume (Mm³)	In situ Density (t/bcm)	Waste Volume Mined (40% Swell) (Mlcm)	Loose Density (t/lcm)	Waste Dump Capacity (Mm³)	Contingency (%)
Waste South and Central	408.9	148.5	2.75	207.9	1.97	216.0	3.8
• Waste	408.0	148.1	2.75	207.4	1.97		
• Inferred	0.9	0.3	2.81	0.5	2.01		
Western and RoM	32.9	13.9	2.37	19.4	1.69	19.42	0.1
• Waste	32.8	13.8	2.37	19.4	1.69		
• Inferred	0.03	0.01	2.79	0.0	1.99		
Total	**441.8**	**162.3**	**2.72**	**227.3**	**1.94**	**235.4**	**3.9**

Table 18-39 notes: all data is reported on a dry basis

The current design capacity is sufficient for the 223 Mlcm of waste (assuming 30% swell with re-compaction) and allows for variations in waste tonnes and swell, with additional capacity available on the south and central dump above 420 mamsl. In addition, more dump space is open in the north, but this area was not utilised due to the higher elevation and haulage distances.

18.9 Data Verification

The approach adopted by the various consultants who contributed to this section to verify technical and other data for appropriateness and validity before use is described in Section 12.

18.10 Comments on Section 18

Comments with respect to whether the level of technical and cost development for this section is in alignment with the requirements of a DFS are summarised in Sections 18.10.1 to 18.10.8, following. Additional supporting detail is presented in Section 25.

18.10.1 Geotechnical

Engineering geotechnical work undertaken is sufficiently representative of the site and site features for use. Further the geotechnical testwork results are appropriate for use in a DFS without reservation.

18.10.2 Roads and Airstrip

The level of technical and cost development for the Site Roads and Airstrips are in alignment with the requirements of a DFS/Class III estimate (AACE).

18.10.3 Mine Services Area (MSA)

The level of technical and cost development for the MSA area is in alignment with the requirements of a feasibility study. If an owner mining operation were reverted to, capital costs may differ. As described earlier in this section, the MSA facility is sized appropriately to properly support the mine operations over the LOM.

18.10.4 Emulsion Plant and Explosives Storage

The emulsion plant and explosives storage facility have not been developed to a DFS level of technical and cost development, albeit it is not material to the overall study results. The explosive supply services will be contracted out, thus the plant and associated equipment required to deliver the blasting services, will be the explosive contractor's responsibility to size and install.

18.10.5 Power Supply

ECG consider the level of technical and cost development for the grid connection power supply are in alignment with the requirements of a DFS.

18.10.6 Tailings and Water Management

The tailings are water management infrastructure has been developed and presented in accordance with the level of technical/engineering detail expected of a DFS.

18.10.7 Waste Rock Management

The level of technical development and layout of the waste rock dumps is in accordance with the requirements of a DFS.

18.10.8 Balance of Infrastructure

For the balance of infrastructure, the level of technical and cost development is in alignment with the requirements of a DFS, with no caveats.

18.11 Interpretations and Conclusions

Interpretations, conclusions and risks for Section 18, are presented in Section 25 of this Report.

18.12 Recommendations

Recommendations for Section 18 are presented in Section 26 of this Report.

18.13 References

References cited in the preparation of Section 18 are presented in Section 27 of this Report.

19. MARKETING STUDIES AND CONTRACTS

19.1 Marketing Studies

19.1.1 Overview

The marketing section outlines current and future scenarios for commodity (gold and silver) market pricing, as well as the macroeconomic drivers for changing key raw material input costs.

In the use of the data presented in Sections 19.1.2 to 19.1.4, consideration has been given to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) guidance on commodity pricing (CIM, 2020), specifically:

- long-term historical averages (five years or longer of data) - not adjusted for inflation;
- three year moving averages; and
- consensus prices.

Endeavour Mining PLC (Endeavour) leverages consensus commodity price data in determining budget and LoM gold prices. This, in combination with benchmarking against peers, ensures that gold prices used for resource, reserve and financial modelling calculations are reasonable and in alignment with industry peers. Said prices are reviewed annually and published internally (Endeavour, 2022a).

Key precious metal and fuel pricing applied in the study are presented in Table 19-1 and discussed more fully herein. Other pricing data used in this Report, is as per the estimate base date, as defined in Section 21.

Table 19-1: Pricing of Key Commodities (Endeavour, 2022a).

Commodity	Pricing Basis
Gold	
• Resource modelling	USD 1500/ozt
• Reserve modelling:	USD 1300/ozt
• Revenue modelling:	USD 1500/ozt
Silver (if applicable)	USD 15/ozt
Long-term Brent Crude Price	USD 73/bbl.

The data presented herein also provides guidance for the work carried out in Sections 14, 15, 16, 17, 18 and 22.

19.1.2 Gold Market

19.1.2.1 Historical Gold Trends

The gold market is highly liquid and benefits from terminal markets (London, New York, Tokyo, and Hong Kong) on almost a continuous basis. Gold prices were in a general downward trend from 1980 to 2000, where gold traded as low as approximately USD 250/ozt. From 2000, the price increased annually until 2011 and 2012, where the price peaked at just under USD 1800/ozt. There was a sharp correction to the gold price in 2013, with the end of Quantitative Easing monetary policy by the US Reserve Bank.

From 2013 to 2019, the gold price remained range bound between USD 1050/ozt and USD 1350/ozt.

Since 2019, gold price has increased steadily from the USD 1300/ozt level, to around USD 1800/ozt today, with a peak above USD 2000/ozt in August 2020 and again in March 2022.

19.1.2.2 Gold Pricing

Section 19.1.2 looks at historical gold pricing and analysts' projections, and secondly the resource and reserves gold pricing used by peer group companies.

Table 25-1 presents historical gold price statistics on an annual basis from 2000 through to 31 March 2022 and illustrates price volatility over this period.

Table 19-2: Annual Historical Gold Price Statistics (S&P Capital IQ, 2022a) (Endeavour, 2020)

Period	Min USD/ozt	Max USD/ozt	Average USD/ozt	3YDMAV USD/ozt	Nominal Close USD/ozt	Real Close[101] USD/ozt
2000	264	316	279	296	272	260
2001	255	293	271	281	279	267
2002	278	349	310	285	347	332
2003	323	416	364	306	415	397
2004	375	455	410	339	438	419
2005	412	528	445	382	517	494
2006	517	719	604	456	636	608
2007	607	839	697	539	833	796
2008	710	1,002	871	654	878	839
2009	810	1215	973	787	1096	1047
2010	1063	1423	1226	942	1419	1356
2011	1311	1899	1572	1160	1564	1495
2012	1538	1789	1669	1360	1674	1600
2013	1190	1692	1410	1469	1205	1151
2014	1141	1382	1266	1479	1184	1131
2015	1051	1301	1160	1376	1061	1014
2016	1060	1366	1248	1271	1151	1100
2017	1151	1349	1258	1233	1302	1244
2018	1174	1358	1269	1234	1283	1226
2019	1270	1552	1393	1292	1517	1450
2020	1475	2053	1771	1375	1898	1813
2021	1682	1947	1799	1675	1822	1741
2022 Q1	**1785**	**2056**	**1877**	**1735**	**1941**	**1855**

Endeavour has leveraged Consensus Market Forecasts (CMF) and historical prices for the purposes of determining gold price defined in Table 19-1.

[101] Real terms prices as of Q1 2022 money terms

Table 19-3, following, presents the analysis of gold CMF annually from 2022 through to 2026 and long-term price (LTP) assumptions in real terms[102] (assumed 1 January 2022).

The CMF LTP derive from a June 2022 analyst poll (nine analysists) indicates a median nominal gold price of USD 1746/ozt with a range of USD 1600 to 1900/ozt. For the 3-month period which ended March 31, 2022, the gold price ranged from a low of USD 1785/ozt to a high of USD 2056/ozt, with an average of USD 1877/ozt, and a three-year moving daily average (3YDMAV) of USD 1735/ozt.

Table 19-3: Gold Consensus Market Forecast Analysis (S&P Capital IQ, 2022a)

	Units	2022	2023	2024	2025	2026	LTP (Nominal)
High	(USD/ozt)	1951	2150	2017	2000	2000	1900
Median	(USD/ozt)	1875	1800	1767	1750	1750	1746
Average	(USD/ozt)	1869	1805	1770	1768	1730	1724
Low	(USD/ozt)	1779	1300	1303	1590	1475	1600
STDEV.S	(USD/ozt)	48.54	130.41	141.91	124.45	137.17	89.89
Analysts	(No.)	36	36	32	20	12	9

Table 19-4, following, illustrates peer group gold price assumptions (PriceWaterhouseCoopers, 2020). For 2020, findings are as noted below:

- For reporting of Mineral Resources, 2020 gold price assumptions ranged from USD 1250 to 1750/ozt, average USD 1459/ozt.
- For reporting of Mineral Reserves, 2020 gold price assumptions ranged from USD 1200 to 1750/ozt, average USD 1383/ozt.

Table 19-4: Mining Company Gold Price Assumptions (PriceWaterhouseCoopers, 2020)[103]

Aspect	Range	Gold Price (USD/ozt)			
		2021	2022	2023	LTP[104]
Impairment Testing	Low	1400	1385	1350	1200
	Average	1609	1540	1509	1454
	High	1900	1800	1700	1700
Period		2017	2018	2019	2020
Mineral Reserves	Low	1100	975	1200	1200
	Average	1226	1173	1291	1383
	High	1300	1250	1550	1750
Period		2017	2018	2019	2020
Mineral Resources	Low	1100	1200	1200	1250
	Average	1322	1300	1369	1459
	High	1500	1500	1550	1750

[102] Real term prices, escalated to Q1 2022%

[103] As per the 'Effective Date' of this Report, Price Waterhouse Coopers have not issued an update.

[104] Not specified whether gold prices are on real or nominal basis.

Based on this data, Endeavour's use of:

- USD 1500/ozt gold price for Mineral Resources is slightly towards the higher end of industry peer predictions in 2020; and
- USD 1300/ozt for Mineral Reserves is towards the lower end of industry peer predictions in 2020.

The impairment testing figures in Table 19-4 are the gold prices used by peers when performing a LoM cash flow analysis for a mine site which has indicators of impairment.

19.1.3 Silver Market

This section looks at historical silver pricing and analysts' forward projections. Silver reserves and resources are not modelled by Endeavour for PE 58 and thus the silver prices presented herein are for reference and internal budgeting purposes only.

Table 19-5, following, presents the historical silver price statistics on an annual basis from 2000 through 2022 Q1. For the 3-month period ended 31 March 2022, the silver price ranged from a low of USD 22.24/ozt to a high of USD 26.18/ozt, with an average of USD 23.99/ozt and a three-year moving daily average of USD 21.84/ozt.

Table 19-5: Historical Annual Silver Price Statistics (S&P Capital IQ, 2022a)

Period	Min (USD/ozt)	Max (USD/ozt)	Average (USD/ozt)	3YDMAV (USD/ozt)	Nominal Close (USD/ozt)	Real Close[105] (USD/ozt)	LTP Real,[106] (USD/ozt)
2000	4.57	5.50	4.95	5.15	4.59	6.71	5.25
2001	4.05	4.80	4.37	5.02	4.61	6.63	5.00
2002	4.23	5.07	4.60	4.78	4.76	6.69	5.00
2003	4.34	5.96	4.88	4.70	5.93	8.18	5.00
2004	5.54	8.22	6.66	5.13	6.79	9.07	5.08
2005	6.41	8.98	7.31	5.86	8.81	11.38	5.17
2006	8.72	14.74	11.56	7.60	12.87	16.21	7.75
2007	11.54	15.48	13.37	9.73	14.77	17.88	9.33
2008	8.95	20.75	14.93	11.79	11.30	13.66	10.58
2009	10.53	19.20	14.67	13.63	16.83	19.81	11.25
2010	15.01	30.86	20.16	15.78	30.86	35.79	13.08
2011	26.82	48.41	35.27	21.26	27.69	31.19	16.17
2012	26.34	36.89	31.13	25.31	30.31	33.56	19.58
2013	18.45	32.24	23.79	27.59	19.41	21.17	19.58
2014	15.33	21.96	19.03	27.31	15.66	16.95	18.75
2015	13.70	18.29	15.69	22.41	13.83	14.86	18.33
2016	13.79	20.61	17.08	18.90	15.93	16.78	19.50
2017	15.58	18.52	17.04	17.21	16.95	17.47	19.00

[105] Real terms prices as of 1 January 2022 money terms

[106] Historical Long-Term Price derived from median of Consensus Market Forecasts

Table 19-5: Historical Annual Silver Price Statistics (S&P Capital IQ, 2022a)

Period	Min (USD/ozt)	Max (USD/ozt)	Average (USD/ozt)	3YDMAV (USD/ozt)	Nominal Close (USD/ozt)	Real Close[105] (USD/ozt)	LTP Real,[106] (USD/ozt)
2018	13.96	17.57	15.68	16.37	15.48	15.66	17.50
2019	14.35	19.57	16.19	16.50	17.93	17.64	17.58
2020	12.01	28.89	20.53	17.51	26.49	28.35	18.75
2021	21.53	29.59	25.15	20.90	23.09	24.17	21.23
2022 Q1	22.54	26.18	23.99	21.84	24.82	25.98	22.50

Endeavour has leveraged CMF and historical prices for the purposes of determining silver price. Table 19-6, following, presents the analysis of silver CMF annually for periods from 2022 through to 2026, and long-term price assumptions in real terms (assumed 1 January 2022). The CMF LTP derived from the June 2022 analyst poll (seven analysts) indicates a median silver price of USD 22.50/ozt and a range of USD 20.00 to 26.39/ozt.

Table 19-6: Silver Consensus Market Forecast Analysis (S&P Capital IQ, 2022a)

Statistics	Units	2022	2023	2024	2025	2026	LTP
High	USD/ozt	25.76	27.00	26.40	26.09	26.39	26.39
Median	USD/ozt	24.50	24.00	23.42	22.50	22.50	22.50
Average	USD/ozt	24.23	23.66	22.94	22.40	21.94	22.57
Low	USD/ozt	20.00	18.00	18.00	18.36	18.20	20.00
STDEV	USD/ozt	1.16	1.79	2.06	1.96	2.25	2.02
Analysts	(No.)	31	29	26	17	10	7

19.1.4 Macro Economics

The Financial Model for the Issuer's interest in the Lafigué Project (the 'Project') has been determined in real terms, and as such, does not explicitly model the impact of inflation and purchase price or non-purchase price parity determination of nominal exchange rates. Notwithstanding this, macro-economic drivers may be modelled in the sensitivity analysis.

The following discussion includes a summary of key macro-economic factors which impact the projection of capital and operating costs.

19.1.4.1 Exchange Rates

The budgeting process and LoM expenditure forecasts incorporate assumed long-term and real exchange rates measured against the USD. Table 19-7, following, summarises the exchange rates used for the development of capital and operating costs estimates and are based on the Issuer's 2022 budget forecasts.[107]

[107] With the exception of ZAR, these numbers are from a Bloomberg Forecast, date 13 September 2021

Table 19-7: FOREX Rates

Currency (XXX)	Exchange Rates XXX:USD	Exchange Rates USD:XXX
USD	1.00	1.00
AUD	0.77	1.30
CAD	0.81	1.23
EUR	1.15	0.87
GBP	1.43	0.70
ZAR	0.07	14.29
XOF	0.0018	550

Table 19-8, following, presents annual historical exchange rates from 2000 to 31 March 2022. The purpose of the table is to demonstrate trends in FX rate movements and to justify rates used for the Project.

Table 19-8: Historical Exchange Rates (EUR,GBP,CAD,XOF, AUD, ZAR):One USD (S&P Capital IQ, 2022a)

Year	Annual Average						End of Period					
	EUR	GBP	CAD	XOF	AUD	ZAR	EUR	GBP	CAD	XOF	AUD	ZAR
2000	0.92	1.52	0.68	0.0014	0.56	0.145	0.94	1.50	0.667	0.0014	0.56	0.13
2001	0.90	1.44	0.65	0.0014	0.51	0.118	0.89	1.45	6.250	0.0014	0.51	0.08
2002	0.95	1.50	0.64	0.0014	0.56	0.094	1.05	1.61	0.637	0.0016	0.56	0.12
2003	1.13	1.64	0.71	0.0017	0.75	0.131	1.26	1.79	0.769	0.0019	0.75	0.15
2004	1.24	1.83	0.77	0.0019	0.78	0.154	1.36	1.92	0.833	0.0021	0.78	0.18
2005	1.24	1.82	0.83	0.0019	0.74	0.157	1.18	1.72	0.862	0.0018	0.74	0.16
2006	1.26	1.84	0.88	0.0019	0.79	0.148	1.32	1.96	0.855	0.0020	0.79	0.14
2007	1.37	2.00	0.93	0.0021	0.88	0.142	1.46	1.98	1.000	0.0022	0.88	0.15
2008	1.47	1.85	0.93	0.0022	0.71	0.122	1.40	1.46	0.820	0.0021	0.71	0.11
2009	1.39	1.57	0.88	0.0021	0.90	0.118	1.43	1.62	0.952	0.0022	0.90	0.14
2010	1.33	1.55	0.97	0.0020	1.02	0.136	1.34	1.56	1.000	0.0020	1.02	0.15
2011	1.39	1.60	1.01	0.0021	1.02	0.139	1.29	1.55	0.980	0.0020	1.02	0.12
2012	1.29	1.59	1.00	0.0020	1.04	0.122	1.32	1.63	1.010	0.0020	1.04	0.12
2013	1.33	1.56	0.97	0.0020	0.89	0.104	1.37	1.66	0.943	0.0021	0.89	0.10
2014	1.33	1.65	0.91	0.0020	0.82	0.093	1.21	1.56	0.862	0.0019	0.82	0.09
2015	1.11	1.53	0.78	0.0017	0.73	0.079	1.09	1.47	0.725	0.0017	0.73	0.06
2016	1.11	1.36	0.76	0.0017	0.72	0.068	1.05	1.23	0.746	0.0016	0.72	0.07
2017	1.13	1.29	0.77	0.0017	0.78	0.075	1.20	1.35	0.794	0.0018	0.80	0.08
2018	1.18	1.33	0.77	0.0018	0.70	0.076	1.15	1.28	0.735	0.0018	0.70	0.07
2019	1.12	1.28	0.75	0.0017	0.70	0.069	1.15	1.33	0.769	0.0017	0.70	0.07
2020	1.10	1.28	0.75	0.0017	0.61	0.061	1.10	1.24	0.709	0.0017	0.61	0.07
2021	1.18	1.38	0.80	0.0018	0.75	0.068	1.14	1.35	0.787	0.0017	0.72	0.06
2022Q1	1.10	1.31	0.79	0.0017	0.72	0.066	1.11	1.31	0.800	0.0017	0.75	0.07
Table 19-7 values	**1.15**	**1.43**	**0.81**	**0.0018**	**0.77**	**0.064**	**1.15**	**1.43**	**0.81**	**0.0018**	**0.77**	**0.064**

The FX rates presented in Table 19-7 are mostly reasonable based on historical FX rates illustrated in Table 19-8. The relative contribution of the various FOREX rates to the overall CAPEX and OPEX estimate for the Project, is as noted in Section 21.

19.1.4.2 Consumer Price Inflation

Historical Consumer Price Inflation (CPI) statistics for the period 2000 through 31 December 2021 for the principal corresponding country currencies are reflected in Table 19-9 following and are summarized below:

- For the 12-month period ended 31 December 2021, the YoY CPI for Côte d'Ivoire (CI) is 4.10%.
- For the 12-month period ended 31 December 2021, the YoY CPI for the United States (US) is 4.69%
- For the 12-month period ended 31 December 2021, the YoY CPI for Australia (AU) is 2.86%.
- For the 12-month period ended 31 December 2021, the YoY CPI for the Euro Zone (EZ) is 2.40%.
- For the 12-month period ended 31 December 2021, the YoY CPI for the United Kingdom (GB) is 2.51%.
- For the 12-month period ended 31 December 2021, the YoY CPI for South Africa (ZA) is 4.62%.

Table 19-9 provides context when determining Mineral Resource and Mineral Reserve gold prices as CPI provides a benchmark for variability in gold price due to inflation.

Table 19-9: Historical Consumer Price Inflation (S&P Capital IQ, 2022a)

Year	YoY 12-month CPI							Year Average CPI						
	CI (%)	US (%)	AU (%)	CA (%)	EZ (%)	GB (%)	ZA (%)	CI (%)	US (%)	AU (%)	CA (%)	EZ (%)	GB (%)	ZA (%)
2000	2.53	3.39	5.80	3.20	3.47	1.23	6.99	2.53	3.39	5.80	3.20	3.47	1.23	6.99
2001	4.36	1.55	3.12	0.72	3.12	1.35	4.59	4.36	1.55	3.12	0.72	3.12	1.35	4.59
2002	3.08	2.38	3.03	3.80	2.58	1.73	13.51	3.08	2.38	3.03	3.80	2.58	1.73	13.51
2003	3.30	1.88	2.37	2.08	2.31	1.31	-1.63	3.30	1.88	2.37	2.08	2.31	1.31	-1.63
2004	1.46	3.26	2.59	2.13	2.96	1.68	2.20	1.46	3.26	2.59	2.13	2.96	1.68	2.20
2005	3.89	3.42	2.80	2.09	2.65	2.16	2.02	3.89	3.42	2.80	2.09	2.65	2.16	2.02
2006	2.47	2.54	3.25	1.67	2.68	2.86	4.82	2.47	2.54	3.25	1.67	2.68	2.86	4.82
2007	1.89	4.08	2.96	2.38	4.42	2.30	7.57	1.89	4.08	2.96	2.38	4.42	2.30	7.57
2008	6.31	0.01	3.69	1.16	3.06	3.08	9.31	6.31	0.09	3.69	1.16	3.06	3.08	9.31
2009	1.02	2.72	2.11	1.32	0.34	2.07	6.16	1.02	2.72	2.11	1.32	0.34	2.07	6.16
2010	1.23	1.50	2.68	2.35	2.63	3.15	3.34	1.23	1.50	2.68	2.35	2.63	3.15	3.34
2011	4.91	2.96	2.99	2.30	3.17	3.60	6.32	4.91	2.93	2.99	2.30	3.17	3.60	6.32
2012	1.31	1.74	2.20	0.83	2.10	2.42	5.81	1.31	1.74	2.20	0.83	2.10	2.42	5.81
2013	2.58	1.50	2.75	1.24	0.56	1.95	5.24	2.58	1.50	2.75	1.24	0.56	1.95	5.24
2014	0.45	0.76	1.72	1.47	-0.24	0.71	5.34	0.45	0.76	1.72	1.47	-0.24	0.71	5.34
2015	1.25	0.73	1.69	1.61	0.17	0.50	5.18	1.25	0.73	1.69	1.61	0.17	0.50	5.18
2016	0.72	2.07	1.48	1.50	1.04	1.79	7.07	0.72	2.07	1.48	1.50	1.04	1.79	7.07
2017	0.69	2.11	1.91	1.87	1.50	2.74	4.50	0.69	2.11	1.91	1.87	1.50	2.74	4.50
2018	0.36	1.91	1.78	1.99	1.65	2.00	4.40	0.36	1.91	1.75	1.99	1.65	2.00	4.40

Table 19-9: Historical Consumer Price Inflation (S&P Capital IQ, 2022a)

Year	YoY 12-month CPI							Year Average CPI						
	CI (%)	US (%)	AU (%)	CA (%)	EZ (%)	GB (%)	ZA (%)	CI (%)	US (%)	AU (%)	CA (%)	EZ (%)	GB (%)	ZA (%)
2019	-1.11	1.18	1.57	2.25	1.46	1.31	4.03	-1.11	1.18	1.57	2.25	1.46	1.31	4.03
2020	2.40	1.00	0.80	0.70	0.30	1.00	3.20	2.40	1.00	0.80	0.70	0.30	1.00	3.20
2021	4.10	4.69	2.86	3.40	2.40	2.51	4.62	4.10	4.69	2.86	3.40	2.40	2.51	4.62

Uncertainty exists surrounding the forecasting of inflation rates as countries begin easing COVID-19 restrictions. Additionally, the ongoing Russia-Ukraine war creates further uncertainty, with increasing oil prices and increased economic pressures due to sanctions. Endeavour does not model inflation within the financial models; therefore, the uncertainty of inflation does not impact budgeting or impairment.

Table 19-10, following, summarizes the forecasts for CPI and producer-price inflation (PPI)[108] for countries relevant to the supply of goods for the Project/Mine.

Table 19 10: Forecast Consumer Price Inflation and Producer Price Inflation (S&P Capital IQ, 2022a)

Year	Consumer price inflation (CPI) forecast						Product price inflation (PPI) forecast					
	CI (%)	US (%)	AU (%)	CA (%)	GB (%)	ZA (%)	CI (%)	US (%)	AU (%)	CA (%)	GB (%)	ZA (%)
2022F	5.4%	7.7%	5.0%	5.7%	7.2%	5.8%	N/A	10.5%	13.0%	12.1%	11.3%	8.1%
2023F	4.8%	3.7%	2.6%	2.6%	4.4%	4.6%	N/A	4.0%	3.0%	2.2%	5.7%	2.0%
2024F	4.5%	1.6%	2.3%	2.1%	2.1%	4.5%	N/A	1.6%	1.8%	1.5%	2.0%	4.8%
2025F	3.4%	2.1%	2.1%	1.9%	2.0%	4.4%	N/A	2.0%	2.3%	1.6%	2.0%	5.5%
2026F	3.0%	2.1%	2.1%	1.8%	1.9%	4.3%	N/A	2.0%	2.2%	1.6%	2.0%	5.5%

As illustrated in Table 19-10, above, both CPI and PPI are forecasted to increase above historical trends through 2022 and 2023, however with rising interest rates globally, they should generally settle from 2024 onwards.

19.1.4.3 Consumable Commodity Input Costs

Fuels

CMF for crude prices are presented in Table 19-10. The data suggests a median LTP of USD 67/bbl. for West Texas Intermediate, and USD 73/bbl. for Brent. For the purpose of selecting an appropriate LoM fuel price to use for the Project, a Bent Crude price of USD 73/bbl. has been used as the basis.

[108] PPI measures the average changes in prices received by producers.

Table 19-10: Consensus Market Forecast Crude Oil and Fuel Pricing (S&P Capital IQ, 2022a)

Statistics	Units	2022	2023	2024	2025	2026	2027	2028	2029	2030	LTP
West Texas Intermediate											
High	(USD/bbl.)	86	94	86	75	75	78	78	78	78	78
Median	(USD/bbl.)	70	65	63	60	60	67	67	67	67	67
Average	(USD/bbl.)	69	68	63	60	63	66	66	66	66	66
Low	(USD/bbl.)	55	51	49	48	54	55	55	55	55	55
STDEV	(USD/bbl.)	8	11	9	7	7	8	8	8	8	8
Analysts	(No)	32	23	21	19	10	10	10	10	10	10
Brent Crude											
High	(USD/bbl.)	89	96	89	91	92	98	98	98	98	98
Median	(USD/bbl.)	75	71	66	65	64	73	73	73	73	73
Average	(USD/bbl.)	73	72	67	66	68	73	73	73	73	73
Low	(USD/bbl.)	58	54	52	51	54	60	60	60	60	60
STDEV	(USD/bbl.)	8	12	10	9	11	11	11	11	11	11
Analysts	(No)	30	22	20	19	11	10	10	10	10	10

Considering recent developments in Ukraine, historical WTI crude oil prices were reviewed to determine price impacts of historical events to best forecast the potential fuel price and likely impacts cost and duration of similar events. Figure 19-1, following, illustrates the impact of global and regional events on oil price.

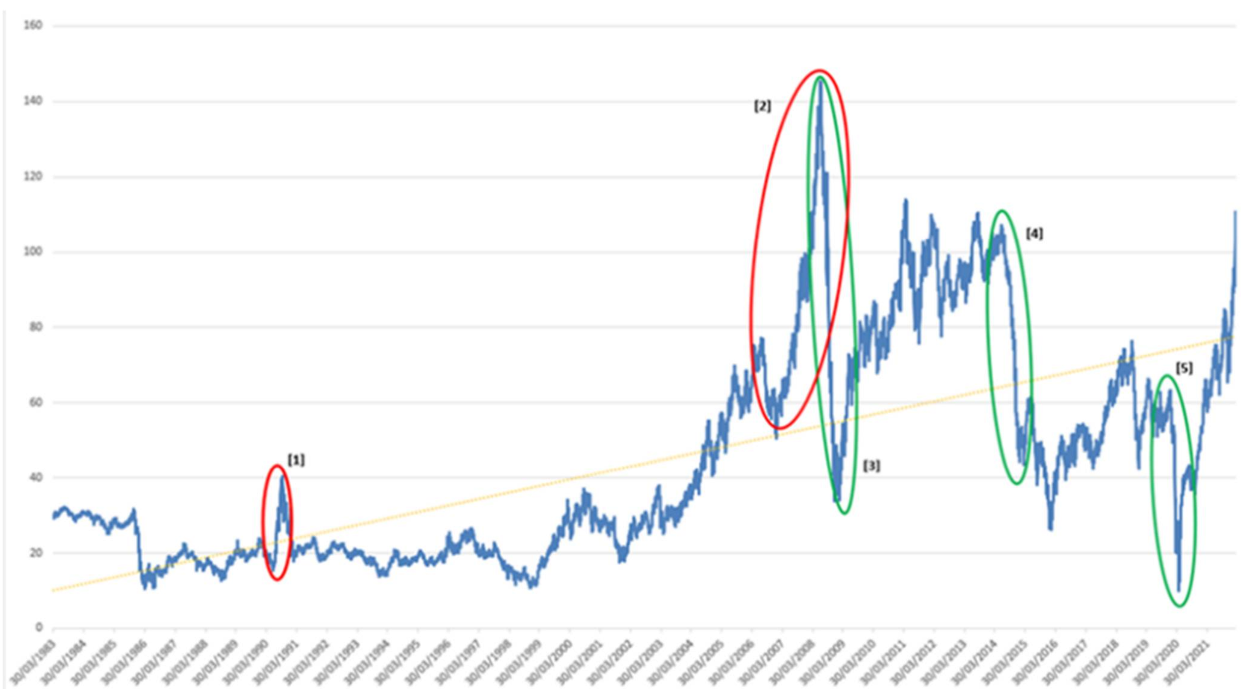


Figure 19-1: WTI Crude Oil Historical Prices (y-axis) and Key Events by Year (Bloomberg, 2022)

Key events associated with price volatility are summarised below:

- Point 1 - Gulf War (1990):

  The Gulf War restricted global supply with available fuel prioritised for the war. As a result, the price of WTI crude oil rose 55% over a 6-month period from July 1990 to January 1991.

- Point 2 - Decline in Oil Supply (2006):

  A global decrease in WTI crude oil supply, partially attributable to the war in Iraq, led to a 130% increase in oil price over an approximate 2-year period.

- Point 3 - Economic Recession (2008):

  The economic crisis of 2008 led to a 76% decrease in WTI crude oil prices from July 2008 to February 2009.

- Point 4 - Global Over Supply (2014):

  WTI crude oil prices fell 60% from September 2013 to March 2015 as a result of increasing global supply, driven by the United States, coupled with an improving geopolitical environment.

- Point 5 – COVID-19 (2020):

  WTI crude oil prices fell 77% over the 2-month period from February to April 2020 as a result of the emergence of COVID-19. Over this period, global demand fell sharply resulting in a price decrease.

Whilst there have been a series of significant swings in crude oil prices over the past forty years, the prices have gradually recovered after each of the five events illustrated in Figure 19-1, with the trendline showing that prices have been increasing with time. While there has been an immediate impact on oil prices as a result of the ongoing Russia-Ukraine war, the extent (price) and duration of the impact is unknown. Therefore, the current Russia-Ukraine war may not have a lasting impact on fuel prices. In Figure 19-1, it has been assumed that the short- to medium-term impact on fuel price could be price swings from the norm of between 35 and 100%.

Fuel pricing in CI is artificially controlled by the government and since Q1 2018 to present, there has been no to limited correlation between the fuel (diesel) and the Brent crude price, as evidenced in Figure 19-2, following. Figure 19-2 is based on data from Endeavour's ITY mine in CI, and is the price received at the mine gate, inclusive off all charges applicable to a mine in production[109]. It is notable that the fuel transport distance from Yamoussoukro[110] is approximately 220 km shorter for the Lafigué mine, which should result in some transport cost savings.

For the Project a LT DDP fuel price of USD 0.91/L has been selected as the price applicable to a Brent crude price of USD 73/bbl., albeit with limited to no correlation. It is evident that at the higher brent crude prices, the fuel price is likely being subsidised, and it is unclear for how long this could be sustained for. To address this concern, the upper range which the fuel price might reach (high-end of the analysts long-term prediction for Brent Crude USD 98/bbl.[111]), was used along with the HFO price[112] (which is allowed to float relatively freely) to estimate the LFO price. Based on this preliminary analysis, it was estimated that the diesel price could potentially reach USD 1.28/L if not artificially controlled. A similar analysis was undertaken for a low range analysis at USD 60/bbl. (Endeavour, 2022c).

[109] See Section 4 for the definition of 'Production'.

[110] Inland terminal, from where fuel is dispatched to the user (defined in further detail in Section 5).

[111] Table 19-10.

[112] HFO typically trades at a (26 to 30)% average discount to HFO in West Africa (high variability).

Based on the aforementioned analysis, three scenarios were identified for modelling in the financial analysis, namely[113]:

Low Scenario (Brent USD 60/bbl.):0.79 USD/L

Base Case Scenario (Brent USD 73/bbl.):0.91 USD/L

High Scenario (Brent USD 98/bbl.):1.28 USD/L

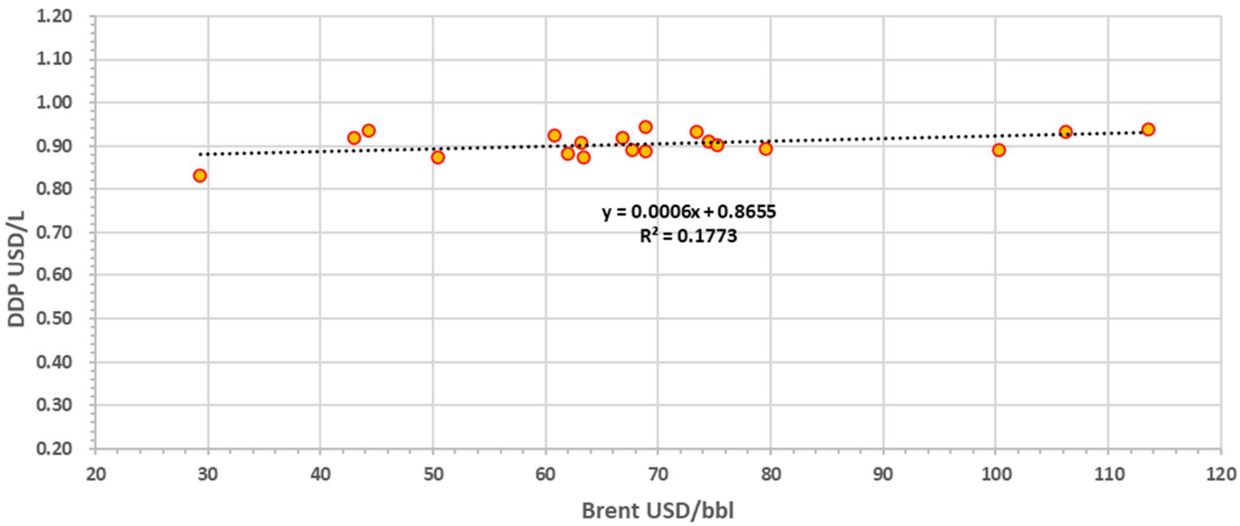


Figure 19-2: Historical Data for Brent Crude Price versus Diesel Price for Endeavour's Ity Mine (Endeavour, 2022c)

As noted in Section 5 of this Report, power generation in CI (2018) is largely a mix of natural gas-fired generation (60%) and hydropower (40%)[114]. Whilst there is a drive for more renewables and alternate energy sources in CI, the implementation of some of these projects has not met the proposed implementation schedule. CI has approximately 10 years of natural gas reserves remaining; however, recent finds are likely to extend this. The electricity price at USD 0.112/kWh is controlled by government and given that natural gas productions is not exported presently, there is nothing to suggest that the mine will see power price rises that will be detrimental to project economics.

19.1.4.4 Steel

Steel is a key commodity used in the mining industry, specifically impacting CAPEX, sustaining CAPEX and OPEX (grinding media/liners).

Figure 19-3, following, outlines the price of structural and stainless-steel from 2019 to 2021, with price forecasts from 2022 to 2024. Steel manufacture and transport is energy intensive and rising secondary raw material input costs and global insecurity may keep prices high.

[113] Crude estimates, to be used for directional guidance only.

[114] 27% capacity factor.

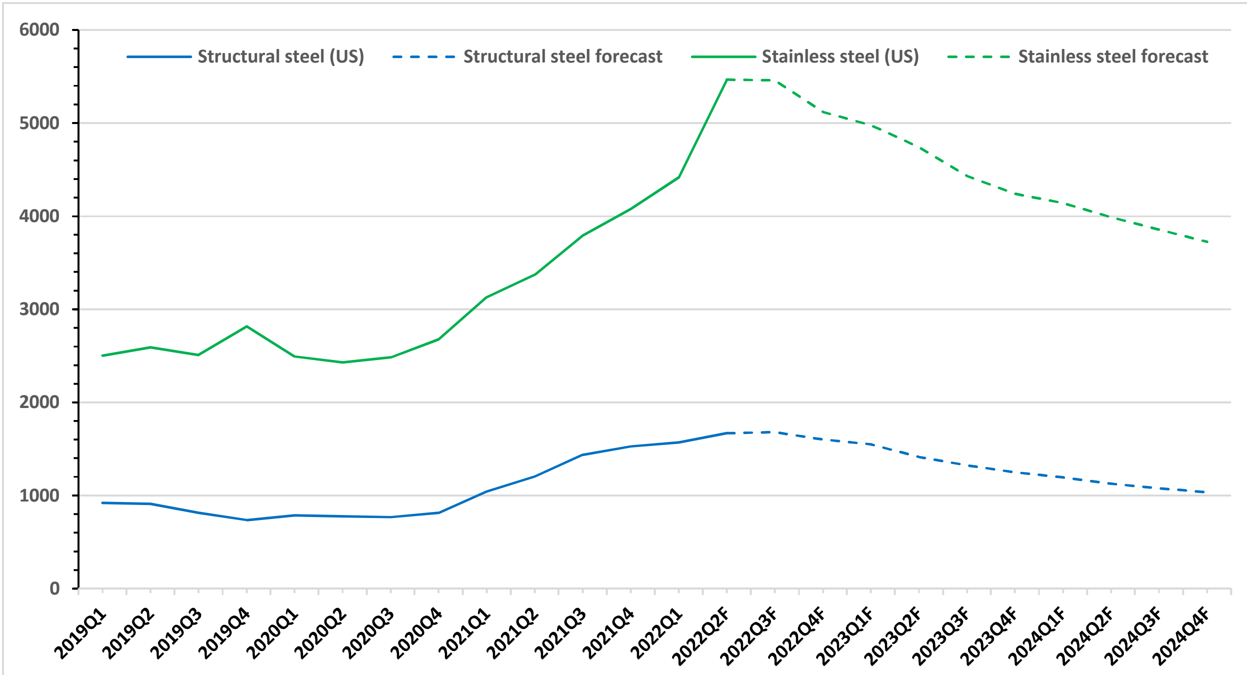


Figure 19-3: Actual and Forecast Stainless (304) and Structural Steel Pricing (USD/t) (S&P Capital IQ, 2022a)

19.2 Contracts

The following discussion pertains only to operational contracts, not contracts associated with construction activities.

Outside of the agreements stated in Section 4 of this Report, as of the 'Effective Date' of this Report, no contracts have yet been entered into for the operational/closure planning phases of the mine's Life cycle.

The Lafigué Mine, owned and operated by Société des Mines de Lafigué SA (SML) is to utilise a hybrid business model for operations, whereby where it makes techno-economic sense to do so, certain primary and secondary mine value chain functions will be outsourced to third parties.

SML falls under the greater Endeavour group, who provide certain technical and commercial corporate support services from Endeavour's head office in London and regional office in CI (Abidjan).

Whilst still to be finalised, the general principles for contract set-up are as summarised below:

- Ownership of 'fixed Infrastructure'[115] provided by contractors, will be amortised over five-years, and handed over to SML at the end of this period.
- Where the contractor provides facilities/infrastructure, they will be provided a graded terraced site, with utilities/services up to agreed battery limits. The terms of supply of utilities/services are still to defined.
- In certain instances, SML will provide buildings and the contractor will fit out (e.g., laboratory), whilst in other instances, the contractor may be provided fully fitted out buildings/facilities and equipment for them to provide their service (e.g., Clinic with ambulance provided).

[115] Structures and buildings

- Where it makes techno-economic sense to do so, SML or a third-party service provider, will provide services to both SML and other contractors (e.g., fuel, security, cleaning, catering and laundry, transport, medical, etc). The terms of supply are still to defined.
- Minimise contractors duplicating support services/roles on the mine, where it makes more sense for one party to provide a common service.

Some of the primary and secondary mine value chain business functions, the likely provider if known, and the commercial basis for cost development in the DFS, is summarised in (Table 19-11). As previous stated, as of the 'Effective Date' of this Report, no key outsourced services contracts are in place for operations.

Table 19-11: Mine Business Model/Basis (Endeavour, 2022d)

Business Area	**Operational Basis**	**DFS Estimate Basis**
Primary Mining Value Chain Functions		
• Exploration	Endeavour	Corporate cost (not back charged)
• Mining[116]	SML/Outsourced	Tender (non-expiring and subject to review/change)
• Processing	SML	DFS Level cost build up
• Tailings management	SML	DFS Level cost build up
• Gold transport, refining and sales	Endeavour/Outsourced	Email pricing
Secondary Mine Value Chain Support Functions		
• Fuel supply & dispensing[117]	Outsourced	Budget quotation/rates
• Power supply	CI Energies	Benchmark/factored pricing from other CI operations
• Emulsion and explosives supply	Outsourced	Benchmark/factored pricing from other Endeavour operations
• Laboratory/Analytical Services	Outsourced	Budget quotation/rates
• Rehabilitation (operational over LoM)	Not defined for DFS	Not developed
• Maintenance		
– Public road maintenance	Not defined for DFS	Allowance
– Site road maintenance	Not defined for DFS	Allowance
– Buildings and facilities	Not defined for DFS	Allowance
– Non mining mobile equipment maintenance	SML	DFS level cost build up
• Site Services		
– Bus services (People)	Outsourced	Benchmark/factored pricing from other Endeavour operations
– CI Aviation transport services (People)	Endeavour	Corporate cost (not back charged)
– Travel and visa services	Endeavour/SML	DFS level cost build up
– Cleaning, catering, and laundry	Outsourced	Benchmark/factored pricing from other Endeavour operations
– Recruitment[118]	Outsourced	Allowance

[116] Includes; waste rock management, grade control, drilling & blasting, dewatering, road dust suppression and maintenance and tailings lifts.

[117] Includes management of waste petroleum products.

[118] One or more of the services providers stated in Section 24, may be carried over to operations.

Table 19-11: Mine Business Model/Basis (Endeavour, 2022d)

Business Area	Operational Basis	DFS Estimate Basis
– Training	Not defined for DFS	Costs included in training budget, % of total labour cost.
– Medical/clinic	Outsourced	Benchmark/factored pricing from other Endeavour operations
– Logistics (Third-party provider)	Outsourced	Allowance
– Supply Chain	Endeavour/SML	DFS level cost build up (SML service not back charged)[119]
– General Waste Management	SML	Allowance
– Security	SML/Outsourced	Benchmark/factored pricing from other Endeavour operations
– Gendarmes	Outsourced	Benchmark/factored pricing from other Endeavour operations
• Technical and minor services contracts	Outsourced	Allowances
• Information and Communications Technology (ICT)		
– Cellular and internet	Outsourced	Benchmark/factored pricing from other Endeavour operations
– Satellite	Not required	Not applicable
• Off-site services		
– Local/Regional Hospitals/clinics	Not defined	Not defined
– Training	Not defined	Not defined
– Waste management	Not defined	Not defined

With respect to gold sales, Gold Doré (approximately 92% m/m Au) produced at the mine site, will be transported by air (250 to 300 kg consignments by Brink's Inc.)[120] to Switzerland (Zurich) for refining by Metalor Technologies SA (Metalor). Gold sales are contracted though one of the three following entities:

- METALOR Technologies SA.
- StoneX Group Inc.
- Endeavour's Syndicate Banks.

19.3 Independent Audits/Reviews

No independent audits and/or reviews have been undertaken for either the Marketing or Contracts sections of this Report.

19.4 Data Verification

The approach taken to verifying the data used and presented in Section 19 is discussed in Section 12 of this Report.

[119] Service not currently provided to outsourced service providers.
[120] Two shipments per month.

19.5 Comments on Section 19

19.5.1 Markets

Gold pricing used for resource and reserve modelling (reserves: USD 1300/ozt; resources: USD1500/ozt) are reasonable and in alignment with industry norms stated in Section 19.1.2. Based on the mean long-term nominal gold price indicated (USD 1746/ozt), the use of a gold price of USD 1500/ozt for revenue modelling is considered conservative.

Fuel prices and exchange rates are considered reasonable, albeit there may be short-term misalignment between budget and actual 2024/2025/2026 prices.

19.5.2 Contracts

The tender and budget quotation methodology used to develop costs for the DFS, is in accordance with standard DFS requirements. However, there are concerns that the basis of quotation/tender is not optimised with respect to the provision of services between SML/the contractor and other contractors. Potential savings may be realised in both costs and labour numbers if optimised. No opinion is offered in this Section on the pricing provided by said contractors, this is dealt with separately by the responsible QPs

The appropriate use of benchmarking/factoring to derive costs is likely to deliver costs in accordance with the requirements of a DFS, however missing out a commercial stage in the development of a project, makes the next stage of negotiation more complicated and business optimisation opportunities could be missed.

19.6 Interpretations and Conclusions

Interpretations and conclusions pertaining to Section 19, are presented in in Section 25 of this Report.

19.7 Recommendations

Recommendations pertaining to Section 19, are presented in in Section 26 of this Report

19.8 References

References cited in the preparation of Section 19, are presented in Section 27 of this Report

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

This Environmental and Social (E&S) Chapter has been informed by the Environmental and Social Impact Assessment (ESIA) process undertaken from 2019 to early 2021, and the final ESIA approval process undertaken for the Lafigué Project (the 'Project').

Cabinet ENVAL (Enval), an environmental and social consultancy based in Abidjan, Côte d'Ivoire (CI), was appointed by La Mancha Côte d'Ivoire (LMCI), a subsidiary of Endeavour Mining plc ('Endeavour' or the 'Issuer'), to carry out the ESIA Process for PR 329, in support of obtaining a signed-off ESIA by the Ministry of the Environment and Sustainable Development.[121] LMCI's ESIA was subsequently approved on 18 February 2021 (Ministere de L'Environnement, 2021). This approved ESIA formed the basis for the granting of the exploitation permit (PE 58), hereafter referred to as the Lafigué Mining License or Lafigué ML. The Lafigué ML is held by the Société des Mines de Lafigué SA (SML). The ownership history of the Issuer's permits relating to the Project (PR 329 and PE 58) is described more fully in Section 4 and 11 of the Technical and DFS Reports respectively.

Herein, where it is read that a Permit has been granted to LMCI, it is a given that the permit/authorisations are now held by SML.

The area covered by the ESIA was the 'zones d'études' or 'Study Area' illustrated in Figure 20-1 as well as key communities within the Dabakala district. The geographic relationship of the ESIA 'Study Area' to the Lafigué ML and the current Fetekro Exploration Permit (PR 329 or Fetekro EL) is also illustrated in Figure 20-1.

[121] www.environnement.gouv.ci

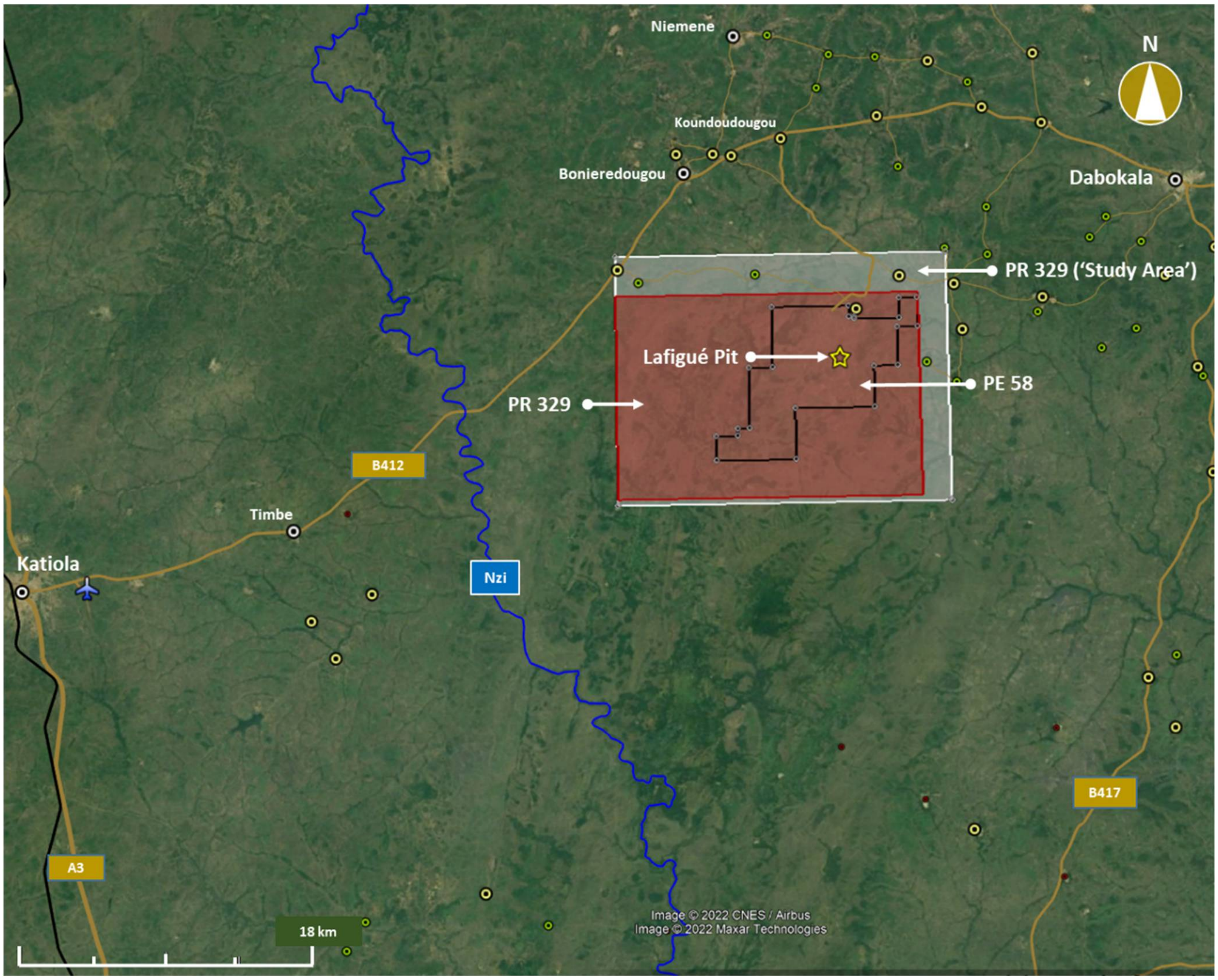


Figure 20-1: 'Zones d'études (Study Area) and PE 58 and PR 329 ((Endeavour Mining Plc, 2022))

Further to the ESIA, this section outlines the social, environment, legal and governance context in which the Mine/Project operates within, as well as the permitting process followed by Endeavour/LMCI/SML for the Lafigué ML or 'Site'.

The Project and the study, considers:

- all the pits, waste dumps and ancillary mine infrastructure associated with the Site;
- all processing facilities (the 'Plant') and the new tailings storage facilities (TSF);
- general infrastructure that supports mining and processing;
- the impacted environment and community;
- national and international legislation; and
- commitments made by Endeavour at a corporate level to Endeavour's and SML's stakeholders.

20.2 Environmental and Social Setting

20.2.1 Property and Region Description

PE 58 is located in north central CI (Section 5 of this Report), approximately 330 km north-northwest of the port city and economic capital city of Abidjan (approximately 470 km by road) and 175 km north-northeast of the political capital Yamoussoukro (approximately 230 km by road). PE58 is located 55 km east of a transnational infrastructure (road, rail, power and fibre) corridor that connects the port of Abidjan with Burkina Faso (BF) to the North.

The Project is located within the Hambol region, one of 31 regions (2014)[122] within CI. The Hambol region is further split into three departments, of which the Project is located within the Dabakala department. The relationship of the 'Study Area' to the regional and departmental boundaries is illustrated in Figure 20-2.

From Figure 20-1 it can be seen that the 'Study Area' is significantly larger than PE 58, and with the exception of the water harvest dam, the majority of the mine infrastructure is located in the northeast corner of PE 58.

The PFS mine Infrastructure that formed the basis of the PFS and the ESIA submission is shown in Figure 20-3, and the final DFS mine infrastructure layout is illustrated in Figure 20-4. There are some changes between the two, and the implications of these changes are discussed later in this section.

[122] https://en.wikipedia.org/wiki/Regions_of_Ivory_Coast

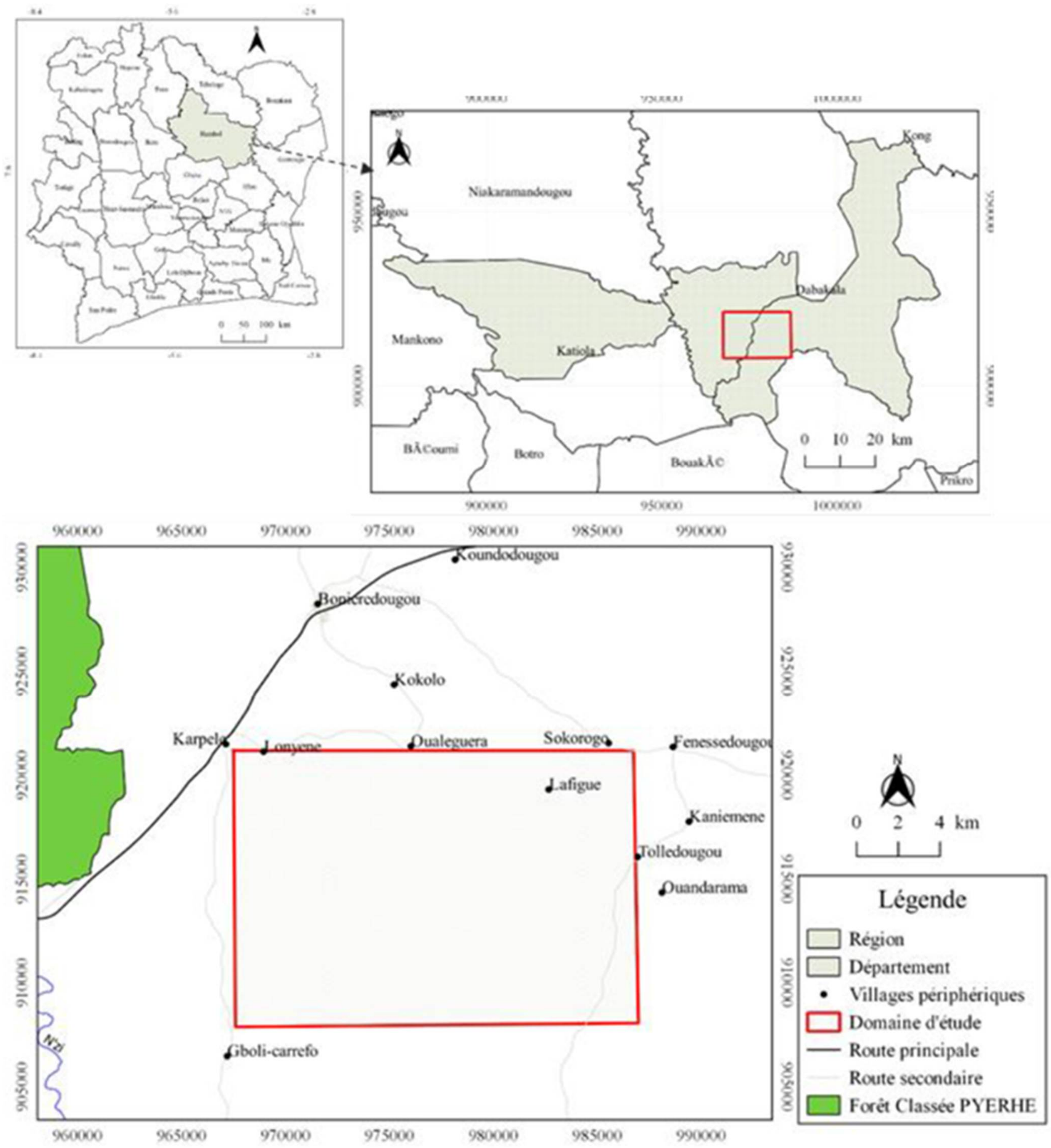


Figure 20-2: CI Regions and Hambol Districts in Relationship to the Study Area (Cabinet Enval, 2021)

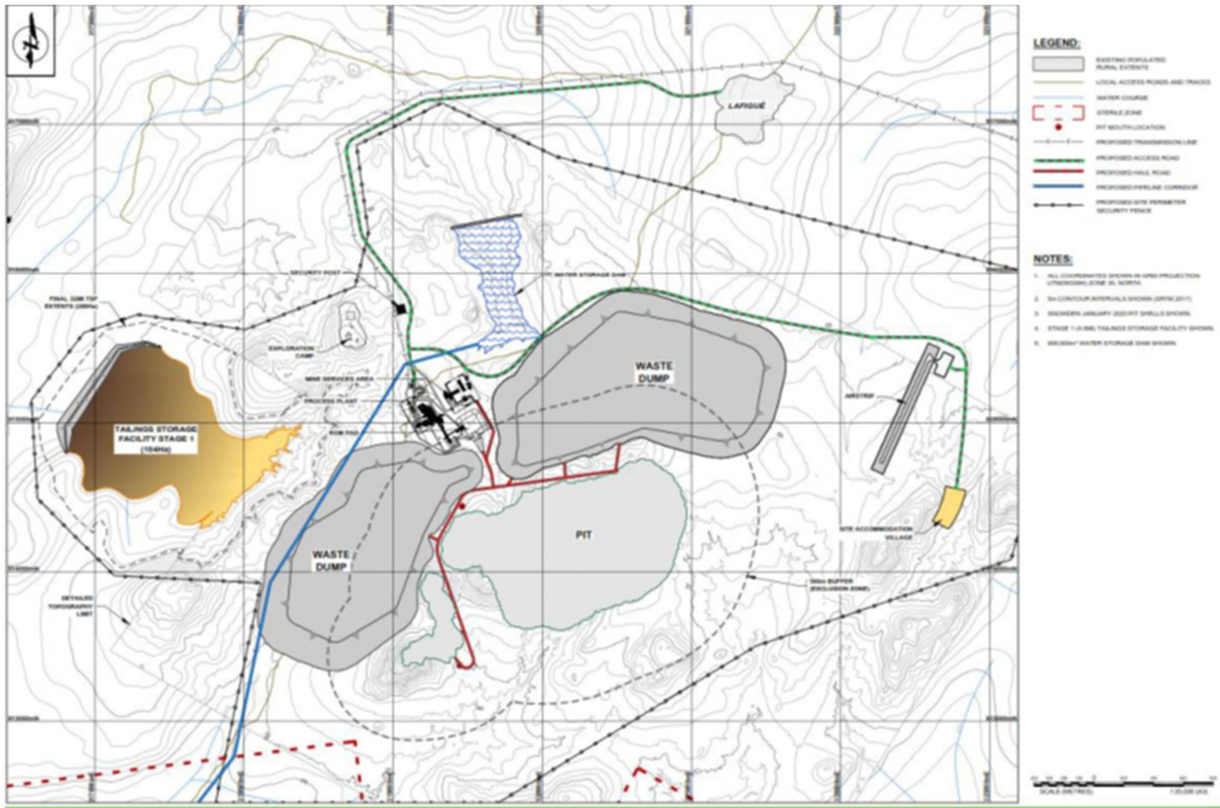


Figure 20-3 : PE 58, PFS/ESIA Mine Infrastructure Basis (Cabinet Enval, 2021)

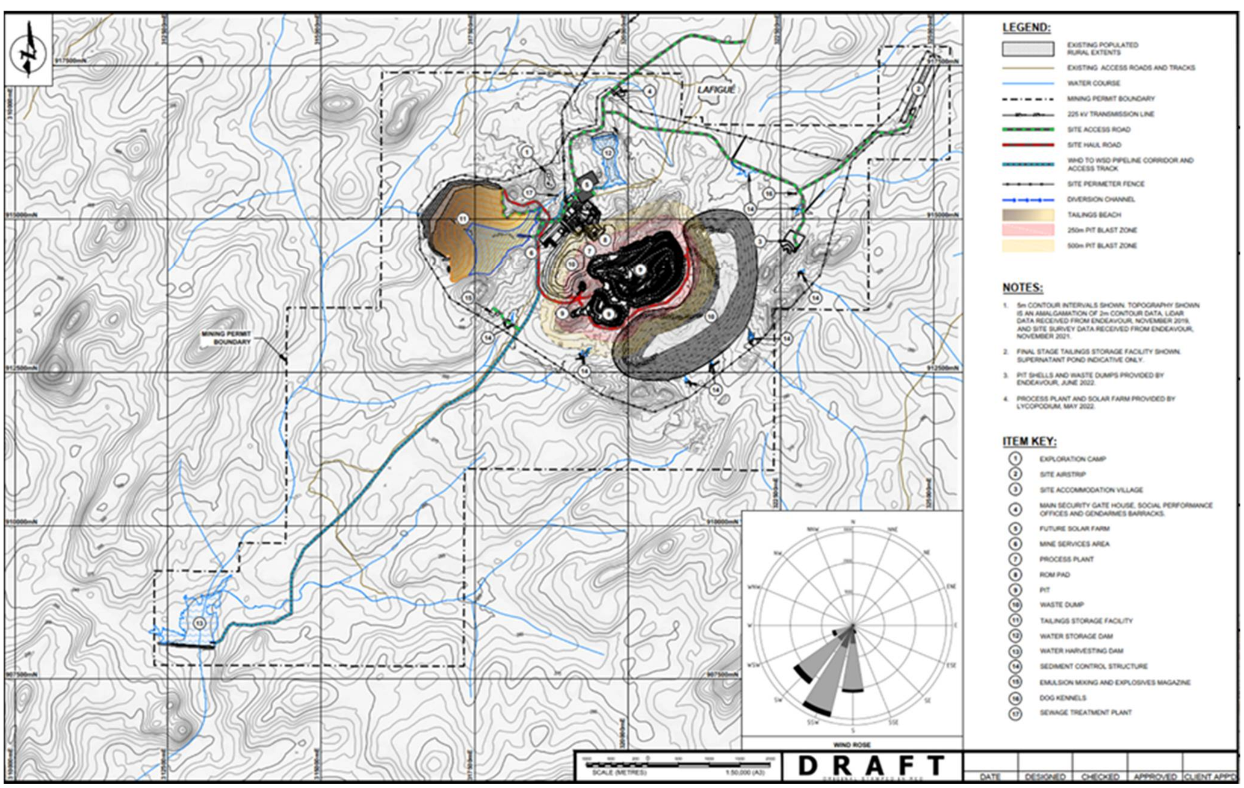


Figure 20-4: PE 58, DFS Mine Infrastructure Basis (Knight Piesold Consulting, 2021)

20.2.2 Regional Climate

The Study Area is located in the Sudanese climatic zone, a tropical regime of transition between the semi-arid sub-Saharan zones and the humid tropical zones of the Gulf of Guinea, regulated by the movement of the Inter-Tropical Convergence Zone (ITCZ) north and south of the equator.

The country is normally warm and humid with an annual average temperature of 28°C and annual average rainfall of 800 mm (Figure 5-4). The wet season largely occurs from April to October with average annual temperatures ranging from 24 to 28 °C. The dry season occurs from November to February and is dominated by the harmattan, a dry, cool wind that blows from the Sahelian zones. The dominant winds blow from the southwest, with average wind speeds of between 1 and 4 m/s.

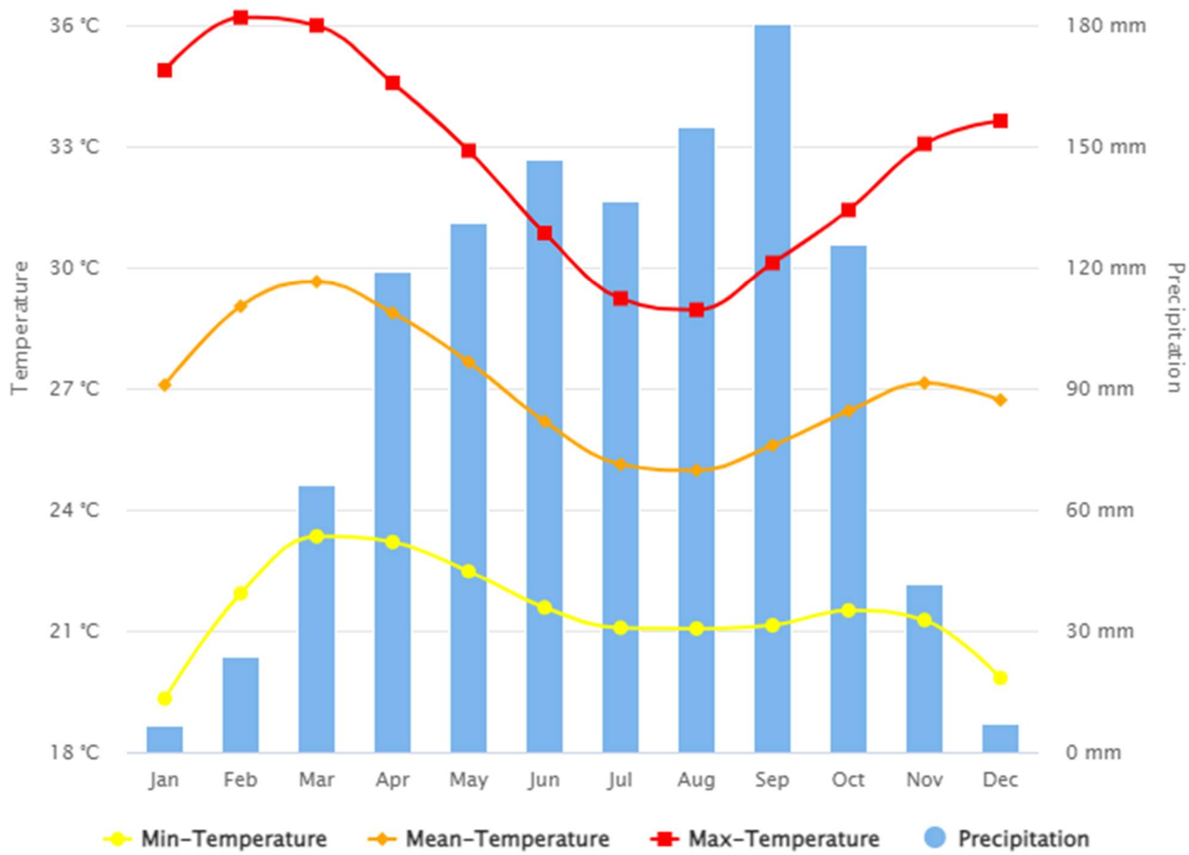


Figure 20-5: Vallée du Bandama[123] Average Monthly Temperatures and Rainfall (World Bank, 2020a)

20.2.3 Regional Topography

There are low lying hills to the north of PE 58, where the elevation increases to just above ~400 mamsl and on PE 58, the elevation varies from approximately 310 mamsl in the northeast corner of the permit, to approximately 230 mamsl in the southeast corner. In the area of the proposed pit, the elevation increases to approximately 400 mamsl.

[123] Historic district, replaced by the Hambol and Gbêkê regions (2014)

20.2.4 Regional Vegetation and Wildlife

The north-central region of CI is characterised by open forest and various savannah types (wooded, grassy etc). Gallery and riparian forests inhabit the edges of the permanent and temporary watercourses and their tributaries (Adjanohoun, Kammacher, Anoma, & Ake-Assi, 1971). To the south, Guinean savannas and dense semi-deciduous forests occur. The mammalian fauna of the savannah region is very diverse and is possibly home to various endangered and critically endangered species. In this part of north-central CI, approximately 34 species of large mammals are present. Historically, the savannah region was home to large iconic mammals such as; lion, elephant, and buffalo, which have now almost disappeared from unprotected areas. Twenty-five species of small mammals occur in the area with approximately 22 bat species (Kingdon, et al., 2013).

20.2.5 Population

The Project is located in the Hambol Region of central CI. The region covers an area of 19 280 km^2 (Wikipedia, 2022), with an estimated population of 429 977 in 2014 and 612 029 in 2021 (Wikipedia, 2022). The region has three departments, namely that of Katiola, Dabakala and Niakara. Table 20-1 provides population demographic information for the Hambol Region.

Table 20-1: Population Demographic Information Hambol Region

Departments:	Dabakala Katiola Niakara	
Population Origins:	Local inhabitants: • Tagbana • Djimini • Djamala • Mangoro • Malinké	Non-national inhabitants: • Burkinabés • Malians • Guineans • Beninese
Religion:	Islam Christianity Animism	

20.3 Legislation, Permitting and Project Development Framework

20.3.1 Overview

The Constitution of the CI recognises that everyone has the right to a healthy environment and as such, the Constitution is the basis for all national legislation. The following subsections deal with:

- applicable CI legislation (Section 20.3.2);
- applicable international conventions/legislation (Section 20.3.3); and
- Permits and agreements required for developing the Project and operating the mine (Section 20.3.4).

20.3.2 National Legislation

Table 20-2, following, summarises the key national mining and environmental legislation applicable to the Project and SML.

Table 20-2: Key Legislation

Thematic Area	Law
Atmospheric emissions	Law No. 96-766 of October 3, 1996 on the Environmental Code. Decree No. 64-212 of May 26, 1964, regulating the use of roads open to public traffic. Decree No. 2017-125 of February 22, 2017 relating to air quality. Order No. 01164/MINEF[124]/CIAPOL[125]/SDIIC of November 4, 2008 relating to the Regulation of discharges and emissions from installations classified for the protection of the environment.
Noise emissions	Law No. 96-766 of October 3, 1996 of the Environmental Code. Decree No. 64-212 of May 26, 1964, regulating the use of roads open to public traffic. Order No. 01164/MINEF/CIAPOL/SDIIC of November 4, 2008 relating to the Regulation of discharges and emissions from installations classified for the protection of the environment.
Blasting	Law No. 96-766 of October 3, 1996 on the Environmental Code. Decree No. 2014-397 of June 25, 2014 determining the terms of application of the law No. 2014- 138 of March 24, 2014 on the mining code.
Closure and Rehabilitation	Law No. 2014-138 of March 24, 2014 on the mining code. Decree No. 2012-1047 of October 24, 2012 setting the terms of application of the polluter-pays principle as defined by Law No. 96-766 of October 3, 1996.
Compensation	Constitutional Law No. 2020-348 of March 19, 2020 amending Law No. 2016886 of November 08, 2016 on Constitution of the Republic of Côte d'Ivoire. Law No. 2014-138 of March 24, 2014 on the mining code. Order interdepartmental No. 247/MINAGRI/MPMEF/MPMB of June 17, 2014 fixing the scale of compensation for destroyed crops. Interministerial Order No. 453/MINADER/MIS/MIRAH/MC LU/MMG/MORE/MPEER/ SEPTEMBER of August 01, 2018 fixing the scale of compensation for destruction or planned destruction of crops and other investments in rural areas and felling.
Compliance	Law No. 2014-138 of March 24, 2014 on the mining code. Decree No. 98-43 of January 28, 1998 relating to Installations classified for the protection of the environment. Decree No. 2005 - 03 of January 6, 2005 relating to Audit environmental. Decree No. 2015-346 of May 13 2015 determining the list of breaches of the water code that may give rise to a transaction and breach. Order No. 01164/MINEF/CIAPOL/SDIIC of November 4, 2008 relating to the Regulation of discharges and emissions from installations classified for the protection of the environment.
Financial	Law No. 95-620 of August 3, 1995 on the Investment Code as amended by Ordinance No. 2012-487 of June 07, 2012 on the investment code. Law No. 96-766 of October 3, 1996 on the Environmental Code. Ordinance No. 2012 – 487 of June 07, 2012 on the Investment Code amending Law No. 95-620 of August 3, 1995 on the Investments Code, Decree No. 95-712 of September 13, 1995 setting the terms of application of Law No. 95-620 of August 3, 1995 on the Investments Code and Order No. 0121 of December 22, 1995 setting the conditions of admissibility of investment declarations and application for investment approval.
Hazardous materials handling	Law No. 92-469 of July 30, 1992 on the repression of fraud in the field of petroleum products and violations of technical safety requirements.
	Order No. 13 SEM. CAB. DH. of February 27, 1974, regulating the development or extension of oil depots and establishments.

[124] Ministry of Water and Forests
[125] Centre Ivorian Antipollution

Table 20-2: Key Legislation

Thematic Area	Law
Health and safety	Law No. 99-477 of August 2, 1999 on the social security code and its decrees amended by Ordinance No. 2012-03 of January 11, 2012. Law No. 2015-532 of July 20, 2015, on the Labour Code. Decree No. 67-321 of August 21, 1967 taken for the application of Title VI "Health and Safety - Medical service" of Law No. 64-290 of August 1, 1964 on the Labour Code. Decree No. 96-206 of March 7, 1996 relating to the health, safety and working conditions committee work. Decree No. 98-505 of 16 September 1998 defining emergency plans in the event of an accident or disaster. Decree No. 2012-980 of October 10, 2012 banning smoking in public places and transport. Decree No. 2014-397 of June 25, 2014 determining the terms of application of the law No. 2014- 138 of March 24, 2014 on the mining code.
Labour	Law No. 99-477 of August 2, 1999 on the social security code and its decrees amended by Ordinance No. 2012-03 of January 11, 2012. Law No. 2015-532 of July 20, 2015, on the Labour Code. Decree No. 96-204 of March 7, 1996 relating to night work. Decree No. 98-38 of January 28, 1998 relating to general hygiene measures in an industrial environment.
Natural Resources	Constitutional Law No. 2020-348 of March 19, 2020 amending Law No. 2016886 of November 08, 2016 on Constitution of the Republic of Côte d'Ivoire. Law No. 2016-886 of November 8, 2016 establishing the Republic of Côte d'Ivoire. Law No. 96-766 of October 3, 1996 on the Environmental Code.
Permitting Procedure	Law No. 2014-138 of March 24, 2014 on the mining code. Law No. 2019-675 of July 23, 2019 on the Forest Code. Decree No. 96-894 of November 8, 1996, determining the rules and procedures applicable to studies relating to the environmental impact of development projects. Decree No. 97-393 of July 9, 1997, creating and organizing a Public establishment of an administrative nature called the National Agency for the Environment or De Agence Nationale de l'Environnement (ANDE). Decree No. 98-43 of January 28, 1998 relating to Installations classified for the protection of the environment. Decree No. 2014-397 of June 25, 2014 determining the terms of application of the law No. 2014- 138 of March 24, 2014 on the mining code.
Social	Law No. 2003-208 of July 7, 2003 on the transfer and distribution of powers from the State to local authorities (in terms of environmental protection and natural resource management). Law No. 2014-390 of June 20, 2014 on the orientation of Sustainable development. Order interdepartmental No. 247/MINAGRI/MPMEF/MPMB of June 17, 2014 fixing the scale of compensation for destroyed crops.
Stakeholder engagement	Law No. 2003-208 of July 7, 2003 on the transfer and distribution of powers from the State to local authorities (in terms of environmental protection and natural resource management). Law No. 2014-390 of June 20, 2014 on the orientation of Sustainable development. Interministerial Order No. 453/MINADER/MIS/MIRAH/MC LU/MMG/MORE/MPEER/ SEPTEMBER of August 01, 2018 fixing the scale of compensation for destruction or planned destruction of crops and other investments in rural areas and felling.
Sustainable development	Law No. 2014-390 of June 20, 2014 on the orientation of Sustainable development.
Waste management	Law No. 96-766 of October 3, 1996 on the Environmental Code. Law No. 98-755 of December 23, 1998 on the water code. Decree No. 2012-1047 of October 24, 2012 setting the terms of application of the polluter-pays principle as defined by Law No. 96-766 of October 3, 1996.

Table 20-2: Key Legislation

Thematic Area	Law
	Order No. 131/MSHP/CAB/DGHP/DRHP/ of 03 June 2009 regulating the management of sanitary waste in Côte d' Ivore Order No. 1240 of October 28, 2009 on the procedure for issuing approval to service providers for the recovery, recovery and/or disposal of waste. Order No. 0012/MINEDD/DGE/PFCB of 15 March 2012 on the authorization border movement and transfer of waste.
Water quality and use	Law No. 98-755 of December 23, 1998 on the water code.

20.3.3 International Legislation

The international legislative context applicable to the development of the Project and the operation of the mine is presented in Table 20-3.

Table 20-3: International Legislation, Conventions and Agreements (Cabinet Enval, 2021)

Name	Date of Ratification by Ivory Coast	Purpose	Impacts on Project
UN Montreal Protocol on Substances that Deplete the Ozone Layer, Montreal, 16 September 1987	November 30, 1992	Protection of the ozone layer	SML shall abstain from using any equipment containing substances that deplete the ozone layer.
United Nations Vienna Convention 1985 1990 London Amendment	November 30, 1992	Protection of the ozone layer	SML shall abstain from using any equipment containing substances that deplete the ozone layer.
United Nations Framework Convention on Climate Change (UNFCCC) 1992	November 24, 1994	stabilisation of greenhouse gas concentrations in the atmosphere at a level that will prevent dangerous human interference with the climate system,	SML to limit greenhouse gases.
Basel Convention on the Control of Transboundary Movements of Hazardous Wastes and Their Disposal, 22 March 1989	June 9, 1994	International treaty to reduce the movements of hazardous waste between countries	SML must limit the export of its waste products or comply with the provisions of this agreement in the event of treatment outside the country.
Bamako Convention on the Prohibition of Import into Africa hazardous waste and the Control of Transboundary Movement and Management of Hazardous Wastes within Africa	June 9, 1994	Defines the rules applicable to imports and waste movement	SML shall refrain from importing hazardous waste from non-contracting parties.
UN Convention on Biological Diversity (CBD), Rio, June 1992	November 24 1994	Develop national strategies for the conservation and sustainable use of biodiversity	Protecting biodiversity in the vicinity of the site. SMLI to implement a protection system in the event of the presence of endangered species.
Paris Agreement on Climate Change, 2015	25 October 2016	First universal climate agreement. The text sets the objective of limiting global warming to less than 2°C, aiming for the 1.5°C mark	Limiting greenhouse gas emissions

Other international initiatives includes:

- Equator Principles - Mentioned in the 2014 CI Mining Code, but not a tangible instruction to apply Equator/IFC Principles/Guidelines.
- Extractive Industries Transparency Initiative (EITI) - Reporting required as per the 2014 CI Mining Code.

20.3.4 Permits and Agreements

The status of permits and agreements required to develop and operate the SML mine are discussed in Section 4 of this Report.

The conditions for the environmental authorisation (No. 00044/MINEDD/ANDE dated 18 February 2021) are detailed in Table 20-4.

Table 20-4: Environmental authorisation No. 00044/MINEDD/ANDE dated 18 February 2021

Article	Implications
Article 1	Granting of the environmental permit following the submission of the ESIA by LMCI is in conformity with the decree n0 96-894 of November 1996.
Article 2	The present authorization is granted to LMCI under the conditions that the company adheres to the recommendations formulated in the ESMP.
Article 3	The present authorization cannot be used as an authorization to start developing the mine. Such decision has to be obtained from a competent technical department.
Article 4	ANDE is in charge of following up the implementation of the recommendations formulated in the ESMP. In that respect ANDE has the right to access the mine site to conduct its environmental follow up and provide the necessary observations.
Article 5	In cases where ANDE observes a non-alignment/adherence with the environmental prescriptions formulated it has the right to bring these to the attention of LMCI for corrective measures within 15 days. After expiration of the said 15 days, the following actions can be undertaken: • ANDE will implement the corrective measures and LMCI will bear the cost. • ANDE can legally suspend the development of the activities up until the corrective measures are undertaken. • ANDE can remove the environmental authorization from LMCI.
Article 6	Any modification to the initial scope of the validated ESIA must be brought to the attention of ANDE.
Article 7	LMCI is held responsible for any environmental damage taking place outside the scope of the ESIA. LMCI will be subject to payment of a fine and will support all rehabilitation costs in line with the regulatory requirements in force.
Article 8	This authorization will become null and void if the project is not developed within 3 years starting from the date of signature of the present document.
Article 9	LMCI is subject to an environmental audit, every three years starting from the end of the construction activities.
Article 10	LMCI is required to inform ANDE about the start of the activities in order to enable ANDE to conduct environmental follow up as prescribed in the ESMP. LMCI is required to produce bi-annual reports on the implementation of the ESMP that will be addressed to ANDE.
Article 11	The head of ANDE is in charge of the execution of the present decision that will be published wherever needed.

20.4 Environmental and Social Impact Assessment

20.4.1 Study Methodology

An ESIA was undertaken by Enval for the proposed Project, culminating with the ESIA being signed off by the Ministry of the Environment and Sustainable Development (Section 20.1). The 'Terms of Reference' for the ESIA were:

- geographical 'Zones d'études' (Study Area);
- key towns within the Dabakala District; and
- legislation applicable to the development and operation of the mine.

The 'Study Area' was based on the historical permit for PR 329, issued in 2013 (approximately 355 km^2). This permit was subsequently renewed in 2017 and 2020 (approximately 250 km^2) and as of 2022, currently in the process of being re-renewed again (approximately 184 km^2). PE 58 occupies approximately 64 km^2 within the historical PR 329. The 'Study Area' coordinates are given in Table 20-5, whilst the Study area in relationship to PE 58 and PR 329 is presented in Figure 20-1, above.

Table 20-5: Study Area (PR 329) (SODEMI, 2013)

Boundary Points	Latitude	Longitude
A	8°19'35.00"N	4°45'40.00"W
B	8°19'35.00"N	4°34'10.00"W
C	8°11'0.00"N	4°34'10.00"W
D	8°11'0.00"N	4°45'40.00"W

20.4.2 ESIA Objectives

The main objectives of the ESIA Process were to:

- provide a clear description of the Project, which was subjected to Environmental Permitting;
- characterise the biophysical and socio-economic baseline conditions of the project's area of influence;
- identify and evaluate the potential negative and positive environmental and social impacts which may result in implementing the Project;
- detail the process undertaken to appropriately engage with all relevant project stakeholders on issues that could potentially affect them; and
- provide an Environmental and Social Management Plan (ESMP) with practical mitigation and management measures to address and/or minimise the identified potential impacts.

20.4.3 Approach to the ESIA (Enval, 2020)

The ESIA for the Project involved several stages from the development of inventories to the approval of the ESIA on 18 February 2021 (Ministere de L'Environnement, 2021).

Infield investigations were carried out by Enval within the Project's area of influence (AoI) between November 2019 to early January 2021, against a Project layout provided by Endeavour. Enval's goal was to assess the potential impacts of the future mining activity on the baseline physical and human environment.

A draft ESIA was developed and submitted for review to Endeavour and to ANDE on 5 October 2020.

A public consultation exercise was held in the Dabakala Department during the second half of November 2020, with a large consultation open to elected officials, customary authorities, administrative authorities, and the local populations concerned. This public consultation was held over a ten-day period and was finalised with an official report giving the opinions of the populations impacted by the Project.

During these public consultation activities, the Prefet of the Department of Dabakala appointed an investigating commissioner who was responsible for collecting the opinions and observations of affected populations over the consultation period. His role was to draft the public consultation report which provides the population's official opinions on the Project.

A major technical review meeting for the ESIA was held in Abidjan in January 2021, following the submission of the Draft ESIA report, to present the issues identified in the ESIA to an inter-ministerial commission assigned for the Project's appraisal. As part of this review meeting, the measures taken to consider environmental, social, legal, and regulatory provisions in effect. This meeting, organised in the presence of the experts from the different ministries who are also stakeholders in the environmental and social (E&S) context of the Project, consequently compiled a recommendation report, and ultimately, appraised the Final ESIA for Environmental Authorisation approval.

At the end of this technical review, an environmental authorisation order was signed by the Minister of the Environment. This environmental authorisation order granted to LMCI was attached to the Mining License application file for the Project, which was submitted at the beginning of February in 2021, and was subsequently presented to the Ministry in charge of Mines to obtain the Lafigué Exploitation Permit (PE 58). Additional information in support of the application was provided on the 19 March 2021, and PE 58 was granted on 22 September 2021 (Presidency of the Republic of CI, 2021).

The following main tasks were undertaken to develop the ESIA and Environmental Social Management Plan (ESMP) and subsequently, obtain the Environmental Authorisation:

- Screening study;
- Preparation of the Terms of Reference;
- Completion of the ESIA;
- Application Appraisal;
- Authority Decision; and
- Environmental monitoring.

The 'Study Area' and the baseline survey points for water (surface- and groundwater), noise and air data collection, are illustrated in Figure 20-6.

To account for seasonal variation (i.e., dry and wet season), the collection of field data was carried out between 16 November 2019 and 16 August 2020.

20.4.4 ESIA and ESMP

To inform the ESIA and ESMP the following environmental and social specialist studies were undertaken:

- Air Quality Study;
- Terrestrial Biodiversity (flora and fauna) Study;
- Hydro-biological Study;

- Hydrological Study;
- Noise Study;
- Socio-economic Study; and
- Compilation of a Conceptual Rehabilitation and Closure Plan.

Environmental and social baseline survey results are presented in Section 20.4.5, following.

20.4.5 Physical Environmental

20.4.5.1 Background

The physical and biological environment ESIA results are discussed in the Sections 20.4.5.2 to 20.4.5.9. Survey/data collection points for; water, noise, dust and key local villages, and geographical terms of reference are illustrated in Figure 20-6. All baseline data was collected from November 2019 to early 2021 by Enval. Survey points for flora and land use are presented in the respective sections.

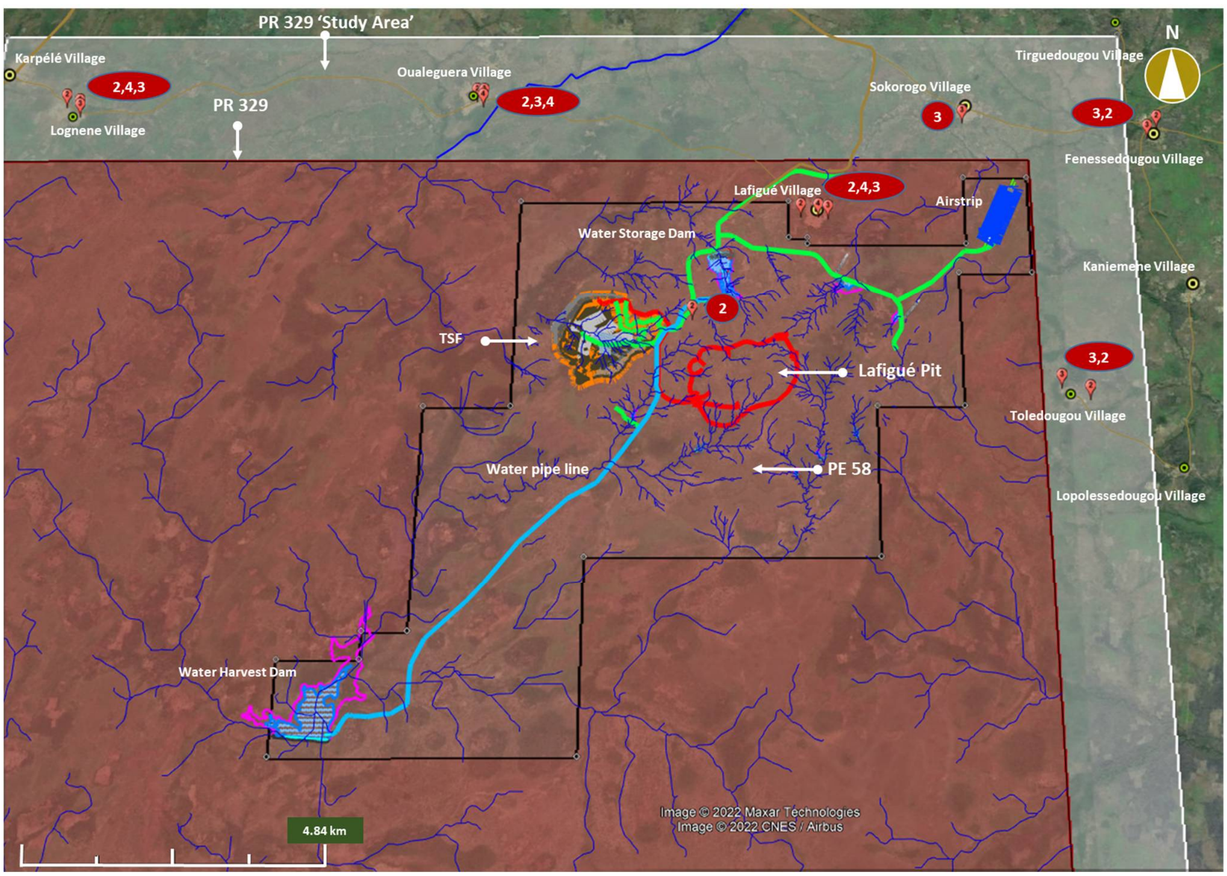


Figure 20-6: Infrastructure, Data Collection Points & Study Area (GoogleEarth®) (Endeavour Mining Plc, 2022)

Figure 20-6 notes:

- Image created by Endeavour utilising DFS Google Imagery and survey points provided by Enval (Cabinet Enval, 2021)
- (1) - Surface water sampling (two points, downstream coordinates incorrect and not shown)
- (2) - Noise sampling (six points)
- (3) - Dust sampling (six points)
- (4) - Ground water sampling (three points)

20.4.5.2 Protected Areas and Internationally Recognized Areas of Biodiversity Importance

One classified forest occurs directly to the south of the proposed Study Area, namely the 'Nangbyon'[126] which is a designated forest. In addition, 12 classified forests are located within a 50 km buffer of the Project 'Study Area'.

Furthermore, the N'Zi River Lodge Voluntary Nature Reserve (IUCN[127] Category II) is also located within this buffer. This reserve covers 41 000 ha of herbaceous savannah.

No World Heritage Sites occur within 50 km of the Study Area.

20.4.5.3 Surface Water

This section provides a baseline of the surface water conditions and does not provide an overview of the water infrastructure or the proposed stormwater management infrastructure and systems. For this information refer to Section 8 of the DFS report and Section 18 of this Report.

The Hambol region is drained by three main rivers and their tributaries, namely the:

- Bandama blanc;
- N'zi; and
- Comoé.

The N'zi, Bandama and the Comoé are all perennial systems. The N'zi drains the Study Area through its tributaries. The Nz'i River at its closest point is 8 km from the western edge of PR 329 and 15 km from the southwest edge of PE 58.

The M'bé River is found in the southern part of the region, from where it continues until it joins the N'zi near M'bahiakro.

The river systems in relationship to PE 58 and PR 239 are illustrated in Figure 20-7.

[126] Reference coordinates not provided.

[127] International Union for Conservation of Nature (IUCN), www.iucn.org

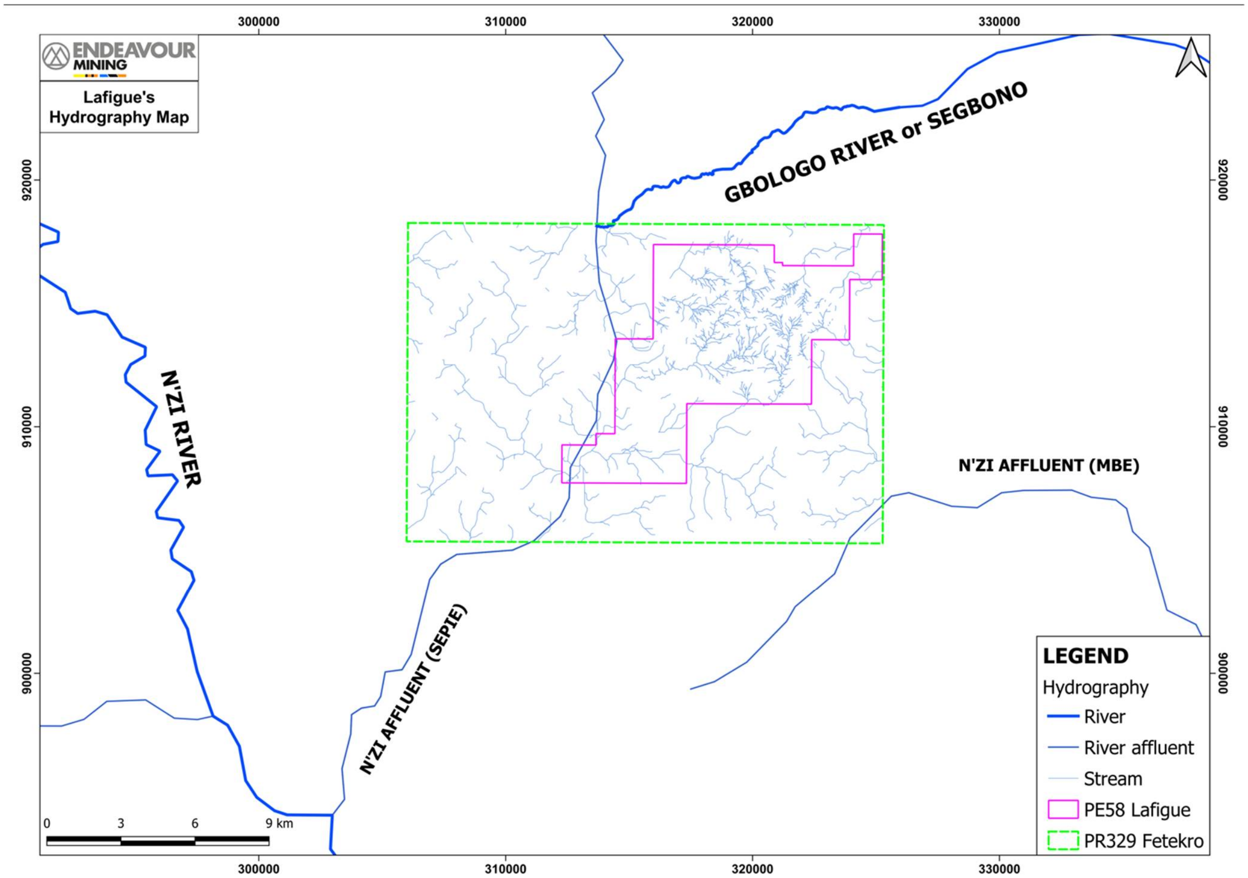


Figure 20-7: Hydrological Setting (Endeavour Mining Plc, 2022)

Only one upstream and one downstream water sample was collected to assess the baseline water quality conditions within the vicinity of the Study Area, the locations of which are described in Table 20-7 and in Figure 20-8. The results from the ESIA indicate that the samples analyzed, comply with the French Regulation Decree No. 2001-1220 of 20/12/2001 relating to the quality limits of raw water used for the production of water intended for human consumption. The results of the microbiological analyzes of the Gbologo River (Table 20-6) indicate that both the upstream and downstream water do not meet the European/World Health Organisation (EU/WHO) Drinking Water Guidelines (World Health Organisation, 2017).

Table 20-6: Microbiological results for the Gbologo River

	Method	Units	Gboloko River (upstream)	Gboloko River (downstream)
Coliform bacteria	ISO 9308-1:2014	CFU/100 mL	>8.104	>8.104
E. coli	ISO 9308-1:2014	CFU/100 mL	<1.103	<1.103
Intestinal enterococci	ISO 7899-2:2000	CFU/100 mL	N=3, 3.103	N=3, 7.103
Salmonella spp.	ISO 19250:2013		Detected	Detected

Table 20-6 notes: CFU (colony forming unit)

Table 20-7: Surface Water Quality (Gbologo River) (Cabinet Enval, 2021)

Parameters	Units	Date of execution	Methods	Results		French Regulation Decree No. 2001-1220 of 20/12/2001*
Coordinates/Date GPS (UTM 30P)	-	25/06/2020	-	P1, 320464 mE, 921683 mN	P2 (unknown)	-
Position				Upstream	Downstream	
Sampling time*	-	25/06/2020	-	18H23	18H06	-
Nitrates*	mg NO_3/L	15/07/2020	ISO 7890-3:1988	3.187	1.098	50
TSS	mg/L	03/07/2020	NF EN 872: 2005	260.00	137.14	-
COD	mg O_2/L	13/07/2020	NFT90-101 : 2001	227	100	-
BOD_5*	mg O_2/L	10/07/2020	NFEN 5815-2 :2012	132.4	60	-
Turbidity	NTU	25/06/2020	ISO 7027-1 : 2016	2.89	2.83	-
pH*		25/06/2020	ISO 10523 : 2008	6.9	6.8	-
Temperature*	°C	25/06/2020		26.4	26.7	-
Conductivity	µS/cm	25/06/2020	NF EN 27888: 1994	174.1 at 25.5°C	173.1 at 25.4°C	-
Choride	mg Cl/L	14/07/2020	NF ISO 9297 :2000	<5	<5	200
Sulfates	mg SO_4/L	16/07/2020	NF T 90-040 :1986	65.75	83.5	250
Phosphate	mg PO_4/L	16/07/2020	NF EN ISO 6878: 2005	0.57	1.00	-

Table 20-7 notes:

- P2 coordinates were incorrectly stated by Enval and have hence not been reported in this table.
- Ion valency not shown
- BOD_5 (Five-day Biological Oxygen Demand)
- COD (Chemical Oxygen Demand))
- TSS (Total Suspended Solids)
- *The French Regulation Decree No. 2001-1220 of 20/12/2001 which specifies the quality limits of raw water was used as there are no criteria for surface waters in CI and EU/WHO standards apply only to drinking water.

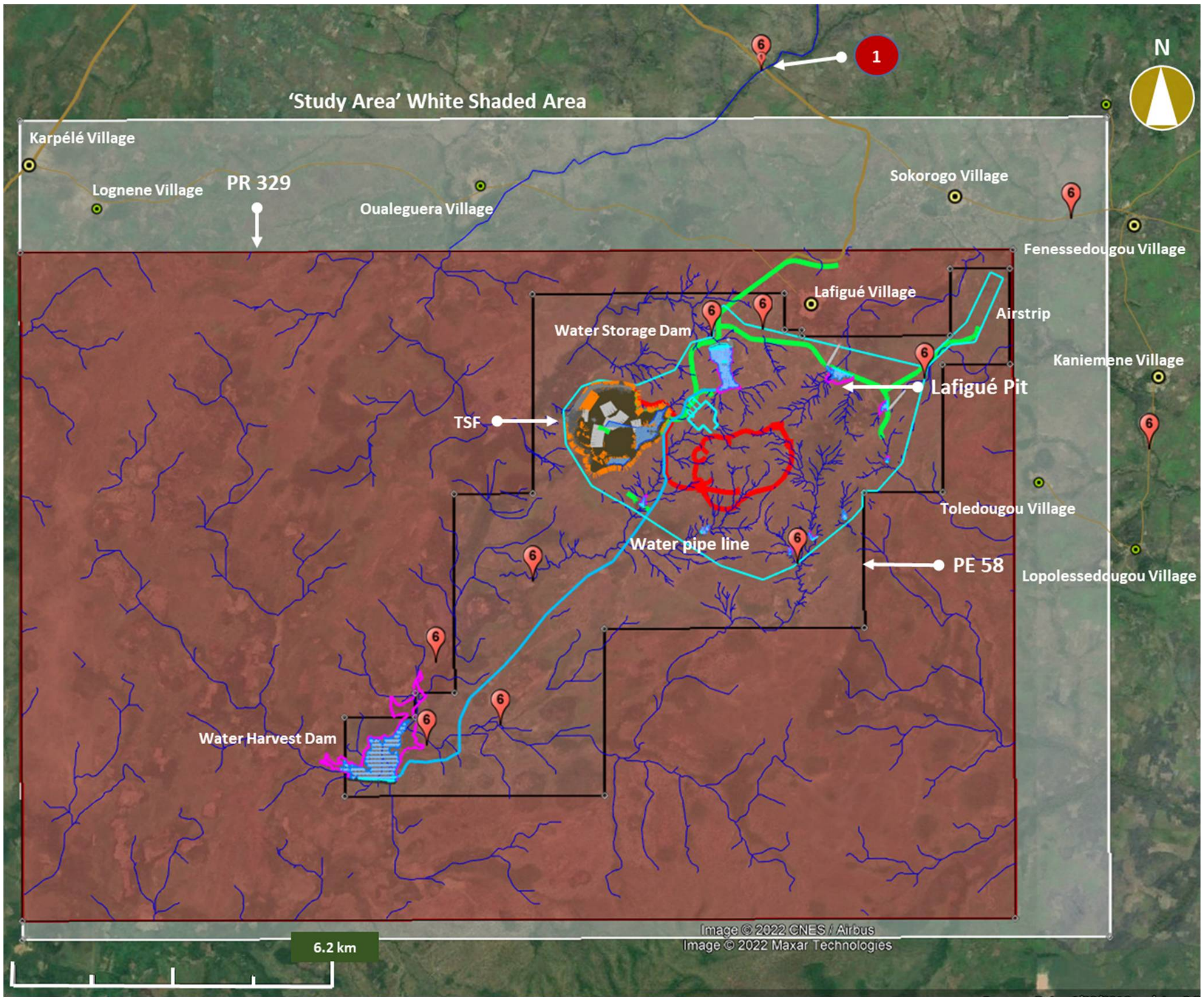


Figure 20-8: ESIA and Current Surface Water Sampling Points (Google Earth®) (Endeavour Mining Plc, 2022)

Figure 20-8 notes:

- Image created by Endeavour utilising DFS Google Earth® Imagery and survey points provided by Enval (Cabinet Enval, 2021)
- (1) ESIA sample point P1
- (2) ESIA sample point P2, coordinates not known, but taken downstream to the southwest of 1 (Gbologo River)
- (6) Current environmental sampling points for surface water

20.4.5.4 Groundwater

The geology at the Lafigué deposit is composed of mafic intrusive, mafic volcanic and felsic intrusive (Cabinet Enval, 2021). This complex is affected by a transpressive deformation and intruded by quartz-porphyry dykes. The mean hydraulic conductivity for the water-bearing fractures at Lafigué is estimated at 0.039 m/d. Groundwater occurrence is associated with open fractures in the mafic intrusive and mafic volcanic. Where no mafic volcanic exists, the mafic intrusive typically have yields less than 5 m^3/h. Open fractures at rock contacts between mafic intrusive and mafic volcanics yields water over 10 m^3/h, especially southwest of the pit shell. Before encountering the bedrock there is a thin laterite layer (1 to 2 m thick), underlain by 2 to 15 m of saprolites. Natural seepage from the saprolites and overburden is not predicted to occur. However, transient seepage from the overburden and saprolites should be expected during the rainy season as the groundwater system recharges from incidental rainfall and surface runoff (EMS, 2021).

The depth of the water level in the pit area is on average 18 mbgl. No groundwater ingress is expected while the pit bottom is shallower than this. The ingress will increase as the pit is deepened from 0 m^3/h in the first year, to 66 m^3/h by Year 11 (life of mine) (EMS, 2021).[128]

The inflow rate is predicted using a numerical model, assuming passive drainage. A passive dewatering entails the collection of draining groundwater at a centralised sump within the pit bottom. This ingress should be considered as a lower limit if active dewatering from out-pit boreholes is to be conducted.

Testwork and geographical location of bores tested within the 'Study Area' are presented in Table 20-8 and Figure 20-6. It was noted that the baseline groundwater quality is generally good, with maximum TDS being less than 350 mg/L. Metals such as Mn, Fe, and Zn are below WHO drinking limits, while As, Ni, and Pb are below detection limits and also below standards.

It is notable that the baseline assessment did not sample to the south, east and west of the Lafigué deposit.

[128] PFS Mine Plan

Table 20-8: Physio-chemical Analysis of Selected Ground Water Sites (Cabinet Enval, 2021)

Parameters	Method	Unit	Date/Execution	Sample References/Results			Standards OMS 2017
				Pump Lognene	Pump Oualeguera	Pump Lafigué	
Coordinates				307657 mE/ 919240 mN	315192 mE/ 919307 mN	321418 mE/ 0917079 mN	
Sampling Time			25/06/2020	8H28	11H00	15H44	
pH	ISO 10523:2008		25/06/2020	6.3	7	6.3	
Temperature		°C		27.8	27.4	30	
Conductivity	25*NF EN 27888-1994	µS/cm		988 at 27.5°C	1158 at 26.6°C	619 at 29.5°C	
Turbidity	ISO 7027-1:2016 NO			0.7	0.47	1.59	
Dissolved Oxygen	NF EN ISO 5814:2012	mg O_2L		4.82	4.63	4.69	
		%		56.3	53.9	55.3	
TSS*	NF EN 872:2005	mg/L	3/7/2020	<2	<2	<2	250
COD*	NF T 90-101:2001	mg O_2/L	13/07/2020	<30	<30	<30	
BOD5 *	ISO 5815-2 : 2003	mg O_2/L	10/7/2020	13	10	8	
Sulfates	NF T 90-040: 1986	mg SO_4/L	16/07/2020	<6	57.4	<6	250
Nitrates	ISO 7890-3 : 1988	mg NO_3/L	15/07/2020	6410	6 269	4 569	50
Phosphates Total	NF EN ISO 6878:2005	mg P/L	16/07/2020	0.28	0.16	0.16	
TDS	HACH	ppm	3/7/2020	480	560	300	
Ammonium	NF T90-015-2:2000	mg NH_4/L	14/07/2020	0.33	0.32	0.24	0.5+
Nitrites	HACH 8507	mg NO_2/L	3/7/2020	0.351	0.305	0.245	
Raw Colour	HACH	mg PtCo/L	16/07/2020	<15	<15	<15	15
Filtered Colour				<15	<15	<15	15
Oil and Greases	NF T 90-202:1979	mg/L	16/07/2020	<0.5	<0.5	<0.5	
Cadmium	ISO 11885:2007	µg Cd/L	8/7/2020	<0.5	<0.5	<0.5	
Manganese		µg Mn/L		<5	23.36	<5	

Table 20-8: Physio-chemical Analysis of Selected Ground Water Sites (Cabinet Enval, 2021)

Parameters	Method	Unit	Date/Execution	Sample References/Results			Standards OMS 2017
				Pump Lognene	Pump Oualeguera	Pump Lafigué	
Copper	ISO 11885:2007	µg Cu/L	8/7/2020	<5	<5	<5	2000
Iron		µg Fe/L		<50	<50	<50	
Lead		µg Pb/L		<5	<5	<5	10
Zinc		µg Zn/L		<50	<50	<50	
Nickel		µg Ni/L		<5	<5	<5	70
Arsenic		µg As/L		<5	<5	<5	10
Cyanides	HACH	mg CN/L		<0.1	<0.1	<0.1	
Mercury	ISO 12846:2012	µg Hg/L		<0.1	<0.1	<0.1	
Total Hydrocarbons	NFT90-202:1979	mg/L	7/7/2020	<1	<1	<1	

20.4.5.5 Soil, Land Use and Land capability

The area is dominated by Ferralitic soils, which are generally located on the low and high plateaus with mineral and organic hydromorphic soils generally found in the vicinity of watercourses and marshy areas. Tertiary formations or Neogene sands are made up of clayey-sandy soils.

No soil land use or land capability study was undertaken as part of the ESIA and therefore exact soil types and fertilities are unknown.

20.4.5.6 Ambient Air Quality

The baseline ambient air quality was established through monitoring across the five mine-affected localities (Figure 20-6). The results of are discussed in the subsections below.

Pollutants

The pollutants measured were: CO; CO_2; SO_2; and NO_2.

The air quality measurements are tabulated in Table 20-9. The CO_2 concentrations measured, varied between 1150 and 1700 ppm which is below WHO guidelines.

Table 20-9: Air Pollutants (Cabinet Enval, 2021)

Element	Project Site	Lafigué	Toledougou	Fenessedougou	Lognene	Oualeguera	CI Limit values µm/m³	WHO air quality guidelines µm/m³
NO_2	ND	ND	ND	ND	ND	ND	40	10 annual average (VEM) 25 (24 hour average)
CO	ND	4	ND	ND	ND	ND	10 000	10 000 (over 8 hours)
SO_2	ND	ND	ND	ND	ND	ND	20	40 (24 hour average)
CO_2	1150	1700	1650	1550	1550	1450	-	

Table 20-9 notes:

- UTM coordinates of sampling location not presented

Dust

Ambient air quality monitoring considered particulate matter, specifically size (PM_{10} and $PM_{2.5}$), Total Suspended Particle (TSP) concentration. These were compared to the CI limits set out in Decree No. 2017-125 of February 22, 2017 relating to air quality. The following results were obtained:

- 9.6 and 46.9 µg/m³ for $PM_{2.5}$ (CI limit is set at 25 µg/m³ for a 24-hour period).
- 20.5 and 319.1 µg/m³ for PM_{10} (CI limit is set at 50 µg/m³ for a 24-hour period).
- 0.037 and 0.899 mg/m³ for total dust (TSP) (CI limit is set at 100 mg/m³ for a 24-hour period).

The dust measurements for $PM_{2.5}$, PM_{10} and TSP are tabulated in Table 20-10. The values exceeding set these limits are highlighted in the red cells. With the exception of Lafigué village, and the Project Site, all other areas had $PM_{2.5}$, PM_{10} and TSP levels within the limit set by Decree No. 2017-125 of February 22, 2017 and the WHO standards.

Table 20-10: Dust Measured at the Sampling Points (Cabinet Enval, 2021)

	$PM_{2,5}$ µg/m³	PM_{10} µg/m³	TSP mg/m³
Project Site	24.6	94.2	0.474
Village Lafigué	46.9	319.1	0.899
Village Toledougou	9.6	20.5	0.037
Village Fenessedougou	13.09	38.6	0.062
Village Lognene	14.37	33.1	0.057
Village Oualeguera	16.36	34.4	0.056
CI Limit values relating to air quality set by decree No. 2017-125 of February 22, 2017	25	50	100
WHO/IFC Standards	**15 (24-hour average)** **5 (annual average)**	**45 (24-hour average)** **15 (annual average)**	-

Table 20-10 notes: UTM coordinates not specified by Enval for each sample point (indicative locations shown in Figure 20-6

20.4.5.7 Ambient Noise Levels

One daytime and one night-time noise level measurements were taken over five mine-affected localities (Figure 20-6). The duration for the noise measurements was five (5) minutes. The assessment of sound levels was carried out according to the methodology of ISO 1996-1: 2003. Sampling locations and results are presented in Figure 20-6 and Table 20-11 following.

The recorded sound level values in the areas surveyed, vary between 42.2 dB(A) and 61.7 dB(A). These values are compared respectively to the CI limit set at 45 dB(A) for daytime noise levels by Order 01164/MINEEF/CIAPOL/SDIIC of 04 November 2008 on the regulation of discharges and emissions from installations classified for the protection of the environment (SDIIC) and then at 55 dB(A) for daytime noise levels by the IFC (IFC, 2007). When compared to the CI guidelines, the noise limits were exceeded in three villages and on the project site, but only one village when compared to the IFC limit. No discussion on night-time noise limits, was provided by Enval.

Table 20-11: Noise Measurement Data (Cabinet Enval, 2021)

Sampling Area	Co-ordinates (P30 UTM)	Average Values dB (A)	SDIIC Daytime Limit for Rural Areas 45 dB(A)	IFC Daytime Limit for Rural Areas 55 dB(A)
Project Site	319136 mE/915127 mN	46	x	
Village Lafigué	321097 mE/917082 mN	61.7	x	x
Village Toledougou	326134 mE/913636 mN	42.2		
Village Fenessedougou	327739 mE/918798 mN	44.7		
Village Lognene	307401 mE/919339 mN	45.8	x	
Village Oualeguera	315072 mE/919413 mN	46.4	x	

Table 20-11 notes : 'X' exceeds limit

20.4.5.8 Terrestrial Biodiversity

The Study Area is in the Sudanian Region of CI, which is typically characterised by wooded savannahs, shrubby savannahs, as well as fallow lands and perennial crops, including cashew tree plantations (Cabinet Enval, 2021).

Further:

- the savannas generally have a woody component, with trees growing among the tall grasses, comprising largely *Andropogon sp*. within the graminoid layer. Both natural and human-induced bush fires burn up to 80% of the savanna areas annually; and
- gallery and riparian forests typically run along the permanent or temporary stream network.

Description of Directly Affected Habitats

A large proportion of the terrestrial vegetation within the 'Study Area' has been severely degraded by subsistence agriculture and artisanal/small-scale mining (ASM) activities.

The five vegetation communities identified within the Study Area include the following:

- Wooded savannah, which has a canopy that reaches about 15 m with species such as *Daniellia oliveri* and *Parkia biglobosa* scattered throughout the graminoid layer.
- Grassy savannah is dominant with a grass cover having a height of approximately 2 m and found generally on rocky soils.
- Gallery forests narrowly following streambeds Common species include: *Uapaca togoensis Vitex grandifolia, Isoberlinia doka, Anthocleista djalonensis, Carapa procera, Elaeis guineensis,* and *Morus mesozygia*.
- Fallow land, which consists of land cultivated in the past and now occupied by invasive species e.g. *Chromolaena odorata*.
- Cultivated land including cashew plantations, yam fields and other food crops.

The vegetation present by 'type' in the Study Area and associated land use is geographically positioned in Figure 20-9 following. The data supporting this figure along with degree of land degradation is presented in Table 20-12.

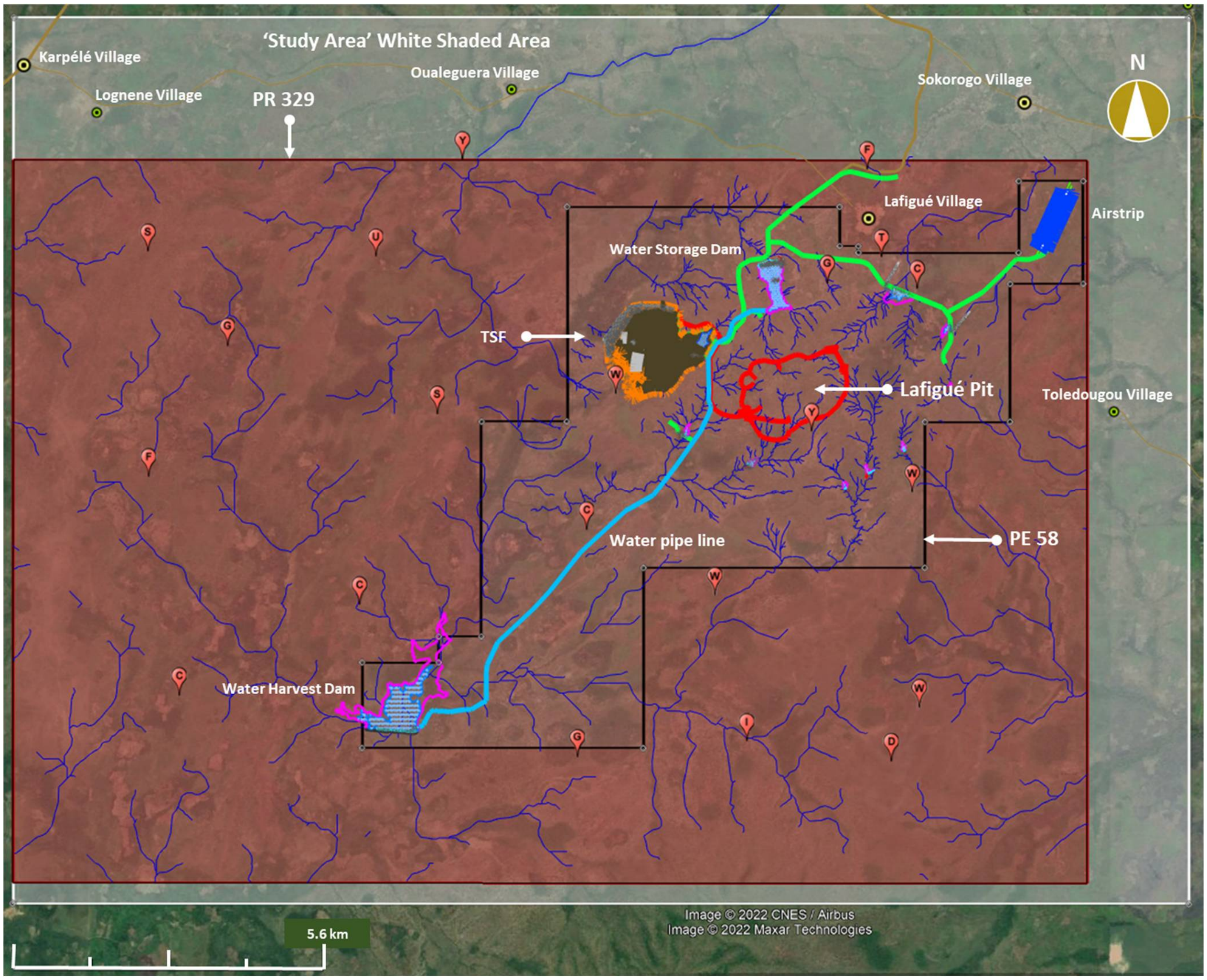


Figure 20-9: Distribution of Inventory Plots at the Study Site During the Rainy Season (Endeavour Mining Plc, 2022)

Figure 20-19 notes:

- Image created by Endeavour utilising DFS Google Earth® Imagery and survey points provided by Enval (Cabinet Enval, 2021)
- (C) - Cashews;
- (Y) - Yam cultivation;
- (G) - Gallery Forest (Dry Bed);
- (S) - Shrub Savannah;
- (F) - Fallow;
- (W) - Wooded Savannah;
- (U) Wooded Savannah in Uapaca;
- (D) - Wooded Savannah in Daniella,
- (I) - Wooded Savannah in Isoberlina;
- (T) - Savannah with Trees

Table 20-12: Coordinates (UTM) of Survey Plots and Level of Degradation by Habitat Type

Point	Habitat	Y	X	Degree of disturbance*
1	Cashew	915903.337	322262.644	5
2	Cashew	911621.34	316313.149	4
3	Cashew	910299.568	312230.555	5
4	Cashew	908688.322	308987.986	4
5	Yam cultivation	913369.115	320358.532	5
6	Yam cultivation	918244.585	314106.246	5
7	Gallery forest (Dry bed)	916000.393	320651.274	2
8	Gallery forest (Dry bed)	914923.953	309877.225	3
9	Gallery forest (Dry bed)	907562.352	316135.448	2
10	Fallow	918038,768	321373.656	3
11	Fallow	912604.74	308449.479	4
12	Wooded savannah	908432.089	322274.715	1
13	Wooded savannah	912273.117	322161.82	1
14	Wooded savannah	910444.777	318613.253	1
15	Wooded savannah	914051.674	316846.282	2
16	Wooded savannah in Daniella	907464.442	321765.298	2
17	Wooded savannah in Isoberlina	907835.173	319174.754	1
18	Savannah with trees on a base	916468,452	321623.526	2
19	Wooded savannah in Uapaca	916515.942	312546.524	3
20	Shrub savannah	913701.405	313639.478	2
21	Shrub savannah	916619.067	308449.479	1

Table 20-12 notes: *Where 0 is pristine and 5 is cultivated.

For the Study Area, a biodiversity screening assessment predicted the occurrence of six Critically Endangered (CR), eight Endangered (EN), and no range-restricted species within a 50 km buffer of the boundary of the proposed Project (Table 20-13). The IUCN list (IUCN, 2019) that informed the ESIA biodiversity screening assessment has recently been updated (2021); and is therefore marginally different to the IUCN list that was utilised for the study (2019).

Table 20-13: Priority Species Potentially Occurring within a 50 km Radius of the Project

Species Name	Common Name	Threat Status*
Aves (Avifauna/Birds)		
Gyps africanus	White-backed Vulture	CR
Necrosyrtes monachus	Hooded Vulture	CR
Polemaetus bellicosus	Martial Eagle	EN
Terathopius ecaudatus	**Bateleur**	**EN**
Torgos tracheliotos	Lappet-faced Vulture	EN
Trigonoceps occipitalis	White-headed Vulture	CR

Table 20-13: Priority Species Potentially Occurring within a 50 km Radius of the Project

Species Name	**Common Name**	**Threat Status***
Magnoliopsida (Flowering Plants)		
Pterocarpus erinaceus	**Kosso**	**EN**
Mammalia (Mammals)		
Colobus vellerosus	White-thighed Colobus	CR
Loxodonta cyclotis	African Forest Elephant	CR
Pan troglodytes	Chimpanzee	EN
Phataginus tricuspis	White-bellied Pangolin	EN
Smutsia gigantea	Giant Ground Pangolin	EN
Reptilia (Reptiles)		
Mecistops cataphractus	Slender-snouted Crocodile	CR
Actinopterygii (Bony Fish)		
Epiplatys olbrechtsi ssp. dauresi	-	EN
Table 20-13 notes: *IUCN Threat Status Categories: CR – Critically Endangered, EN – Endangered; and Species in bold were identified in the Study Area.		

The confirmed presence of several threatened species, as well as nationally protected and partially protected species, suggests that significant biodiversity values may be present within the 'Study Area'. Further detail regarding the inventories undertaken in April 2020 and August 2020 as part of the terrestrial flora and fauna studies. is presented in the following discussion.

Terrestrial Flora

With respect to terrestrial flora, six threatened species were confirmed during the fieldwork:

- *Afzelia Africana* (African Mahogany) is a widespread tree of woodland and dry forest in the savannah belt of West Africa, but has suffered a decline in numbers because of overexploitation of its timber for the commercial market. It is classified as **Vulnerable** and **protected** according to Decree No. 66-122 of March 31, 1966.
- *Khaya anthotheca* (East African Mahogany) is a large canopy tree (up to 60 m high) normally found in riverine fringe forest and floodplains. Khaya Anthotheca is also heavily exploited for timber, resulting in its **Vulnerable** and nationally **Protected** status.
- *Vitellaria paradoxa* (Shea Tree) is characteristic of the west African savannah. It is a small-medium sized tree which is utilised for a variety of purposes i.e., food, medicines, timber, soap, oil and latex, thereby resulting in it being classified as **Vulnerable**.
- *Entandrophragma candollei* (West African Cedar) is also classified as **Vulnerable** and is exploited for its timber as well as for medicinal purposes. It is considered to be more of a forest species. It is also nationally **Protected**.
- *Pterocarpus erinaceus* (Kosso) is an **Endangered** species due to overexploitation for timber, environmental degradation, and climatic change. It is usually found in groves scattered throughout west African savannahs. This species was not on the 2019 IUCN Red List and therefore no locations are available.
- *Khaya senegalensis* (Senegal Mahogany) is a **Vulnerable** species that inhabits riparian forest and moist savannahs. It is threatened by overexploitation due to its medicinal properties, and for its timber.

One endemic species was recorded *Amorphophallus accrensis*.

The location of these species within the Study Area, are presented graphically in Figure 20-10, with supporting detail provided in Table 20-14.

Table 20-14: List of Flora Species within the Study Area on the IUCN list (IUCN, 2019)

Point	Species	X mE (UTM)	Y mN (UTM)	Status
1	*Afzelia africana*	321180	916225	Vulnerable
2	*Afzelia africana*	320730	914685	Vulnerable
3	*Afzelia africana*	320237	913792	Vulnerable
4	*Afzelia africana*	320102	913995	Vulnerable
5	*Afzelia africana*	320201	914243	Vulnerable
6	*Entandrophragma candollei*	314093	918111	Vulnerable
7	*Khaya senegalensis*	317540	914421	Vulnerable
8	*Khaya senegalensis*	317540	914421	Vulnerable
9	*Khaya senegalensis*	321364	916664	Vulnerable
10	*Khaya senegalensis*	321175	916223	Vulnerable
11	*Khaya senegalensis*	321182	916257	Vulnerable
12	*Pterocarpus santalinoides*	321190	916252	Least Concern
13	*Vitellaria paradoxa*	321197	916066	Vulnerable
14	*Vitellaria paradoxa*	320683	915941	Vulnerable
15	*Vitellaria paradoxa*	319897	915488	Vulnerable
16	*Vitellaria paradoxa*	317604	914329	Vulnerable
17	*Vitellaria paradoxa*	321364	916664	Vulnerable
18	*Vitellaria paradoxa*	313698	916972	Vulnerable
19	*Vitellaria paradoxa*	320880	915179	Vulnerable
20	*Vitellaria paradoxa*	320725	914661	Vulnerable
21	*Vitellaria paradoxa*	320344	913844	Vulnerable
22	*Vitellaria paradoxa*	319964	913870	Vulnerable
23	*Vitellaria paradoxa*	320201	914243	Vulnerable
24	*Vitellaria paradoxa*	320839	914725	Vulnerable

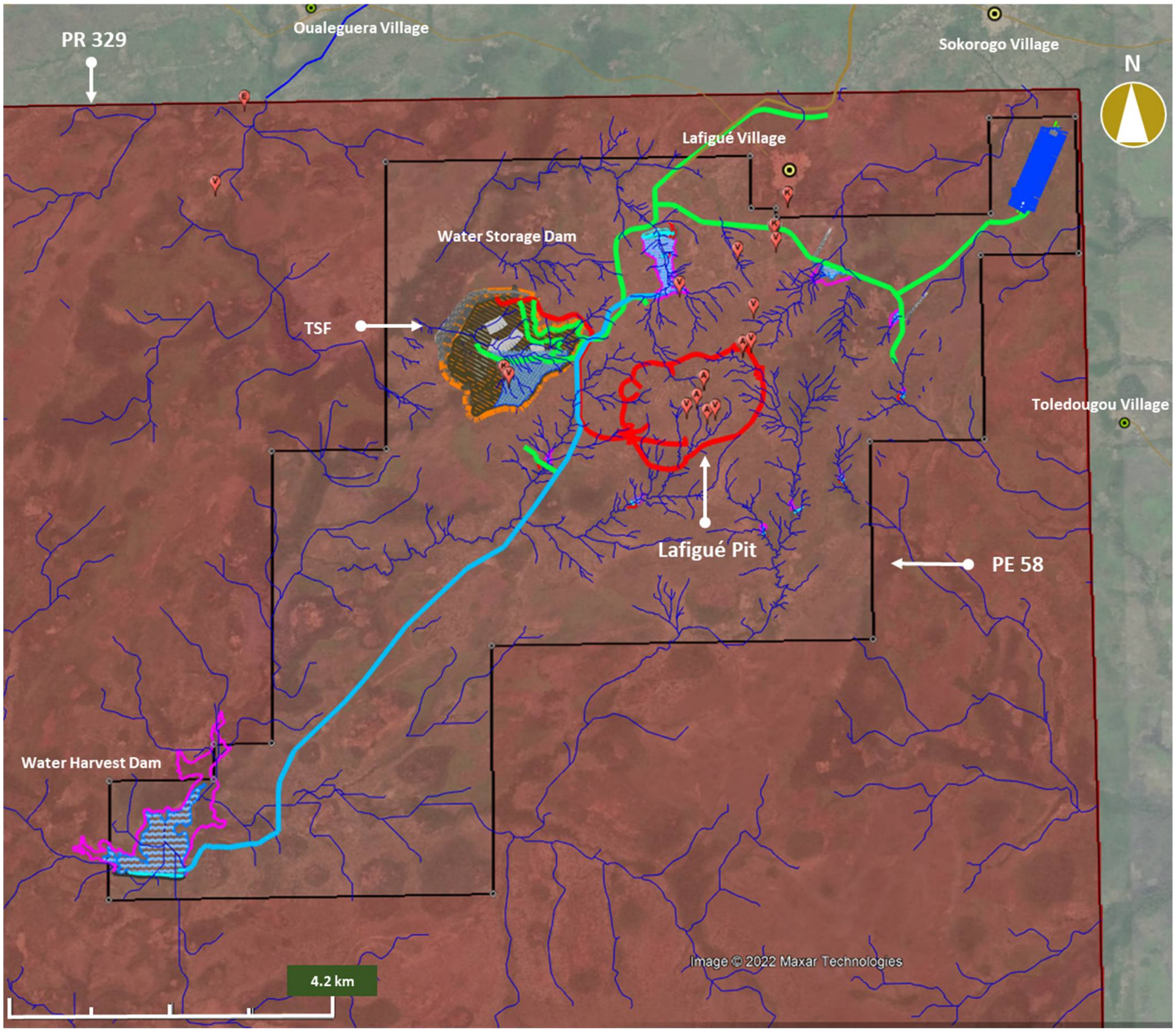


Figure 20-10: Distribution of Species on the IUCN Red List (IUCN, 2019) (Endeavour Mining Plc, 2022)

Figure 20-10 notes:

- Image created by Endeavour utilising DFS Google Earth® Imagery and survey points provided by Enval (Cabinet Enval, 2021).
- (A) *Afzelia africana* (Vulnerable), 5 instances noted in study by Enval.
- (K) *Khaya senegalensis* (Vulnerable), 5 instances noted in study by Enval.
- (V) *Vitellaria paradoxa* (Vulnerable), 12 instances noted in study by Enval.
- (E) *Entandrophragma candollei* (Vulnerable), 1 instance noted in study by Enval.
- Outside of PE 58, Enval previously showed three further species of 'V' to the south of the eastern 'V' illustrated in Figure 20 9. This could not be reproduced using the coordinates given in Table 20-14. Thus, there is a risk that species V is over-represented in PE 58.

Terrestrial Fauna: Mammals

A baseline survey of mammals within the Study Area was conducted during wet season fieldwork in August 2020 and dry season fieldwork in April 2020. In the surveys; 23 large mammals, 15 species of small mammals, and seven species of chiroptera (bats) were recorded.

Of the 23 confirmed large mammal species, only *Erythrocebus patas* (Common Patas Monkey, listed as Near Threatened) was regarded as a threatened species. A further five species (over and above the 23 identified) were reported through interviews (or local knowledge surveys), of which only *Phataginus tetradactyla* (Black-bellied Pangolin, listed as Vulnerable) was flagged as a conservation concern.

According to Law 94-442 of 16 August 1994, two partially protected species were confirmed in the Study Area, namely *Chlorocebus sabaeus* (Green Monkey) and *Cephalophus niger* (Black Duiker). Six species are partially protected. *Orycteropus afer* (Aardvark), which was not confirmed, yet reported by the community, is fully protected, while the unconfirmed species *Mellivora capensis, Nandinia binotata, Potamochoerus porcus* and *Phataginus tetradactyla* are partially protected.

Terrestrial Fauna: Avifauna

A total of 129 bird species were identified. 76 bird species were recorded in the Study Area during wet season fieldwork, while the dry season fieldwork identified 93 species. *Terathopius ecaudatus*, an endangered species, was observed in both surveys.

Terrestrial Fauna: Herpetofauna

Seven amphibian species and eight reptile species were confirmed in the Study Area. Reptiles comprised of three snakes, four lizards and one tortoise species.

No threatened reptiles were observed in the Study Area. However, *Mecistops cataphractus*, is expected to occur in the area, and protected *Python regius* and *Python sebae* is believed to be present within the Study Area.

20.4.5.9 Freshwater Ecosystems

The N'zi River and its associated tributaries drain the survey area. Most of the systems are non-perennial and are therefore completely desiccated in the dry season. These systems seem to be surface water driven and therefore seasonal changes will likely be significant. ASM activities were observed to take place within these systems.

It is important to note that the current ASM activities made surveys difficult. The watercourses have been dammed by the ASM miners and dried out so that mining can commence, hence the low number of sampling points (two). An example of ASM activities along water courses is illustrated in Figure 20-11, following.

Figure 20-11: ASM Mining Activities Along a Watercourse (Endeavour Mining, 2022)

Aquatic Biodiversity

Fish sampling during the dry season resulted in the confirmation of 17 species, with 30 species identified in the rainy season. This distribution for both seasons includes one introduced species (*Oreochromis niloticus)* and one hybrid species (*Coptodon zilii X Coptodon guineensis*). All species are considered as Least Concern in the IUCN Red List. The endangered species *Epiplatys olbrechtsi ssp. dauresi* expected to occur in the Study Area and surrounds was not observed.

Aquatic macroinvertebrate sampling in the dry season yielded 11 taxa belonging to 11 families, 7 orders and 3 classes. In the rainy season, 24 taxa were observed, divided into 18 families, 8 orders and 4 classes. Sixty taxa of microflora were also observed in the dry season and 69 taxa in the rainy season. Examples of the river/aquatic systems in the Study Area are illustrated in Figure 20-12.

Figure 20-12: Example Aquatic/River Systems Within the Study Area (Cabinet Enval, 2021)

Wetland Ecosystems

No wetlands are referred to in the ESIA. Aerial imagery does however indicate that some wetlands may be present in the Study Area. Wetlands provide a wide range of ecosystem services, which are required to be investigated in accordance with IFC Performance Standard (PS) 6. Wetlands are also considered to be sensitive environments subject to EIAs by the CI Environmental Code. Examples of potential wetland sites within the Study Area are illustrated in Figure 20-13. It is noted that a level of degradation is expected as wetlands are commonly used for cultivation and ASM activities, which impacts the overall biodiversity value of the Study Area as evidenced in Figure 20-13.

Figure 20-13: Examples of Watercourses in the Study Area (Cabinet Enval, 2021)

20.4.6 Human Environment

20.4.6.1 Local Administrative Structure

The Hambol region (19 122 km²) one of 31 autonomous regions in CI, has an estimated population of 429 977 inhabitants (RGPH, 2014) or 17 inhabitants/km^2. It is bordered to the north by the region of Tchologo, to the south by the region of Gbêkè, to the east by the regions of Iffou, Gontougo and Bounkani, and to the west by the regions of Poro and Béré.

The Hambol region has three Departments/districts, namely; Katiola, Dabakala and Niakara. The ESIA assessment undertaken covers eight villages (Lafigué, Sokorogo-bobosso, Fenessedougou, Toledougou, Oualeguera, Lognene, Karpele and Gboli Carrefour) likely impacted by the Project and the mine[129] within the Dabakala Department.

20.4.6.2 Demographic and socio-economic setting

Human population demographic data, as well as the socio-economic setting with respect to socio-economic infrastructure/services in the Dabakala department, is summarised in Table 20-15, following.

Table 20-15: Population Information for the Dabakala District (Cabinet Enval, 2021)

Population (RGPH, 2014)	189 254 people	
Population Origins:	Local inhabitants: • Djimini • Djamala • Mangoro • Malinké	Non-nationals inhabitants: • Burkinabés • Malians • Guineans • Senegalese
Religions:	Islam Christianity Animism	
Access to Land:	Inheritance Loan Gift	
Farm Management:	Capital community asset managed by Chief	
Water:	Water distribution Network (SODECI) Hydraulic pumps Wells Rivers	
Electricity:	Some villages within the Department are electrified. However, the eight villages located in the Study Area are not connected to the grid. Alternative sources of energy include wood, solar and coal.	
Schools:	46 private schools	97 public schools
Agriculture:	Crops: • Cashew Cultivation • Maize • Cassava • Millet • Groundnuts • Rice • Bananas • Market gardens	Livestock: • Cattle • Goats • Pigs • Poultry • Beekeeping

[129] With the issuance of PE 58, this has now been reduced to five (Lafigué, Sokorogo-bobosso, Fenessedougou, Toledougou and Oualeguera)

Economic:	Main income generating activities: • Agriculture (mainly cashews which is widely grown as a cash crop for export) • ASM (Artisanal Small-scale Mining) • Trade
Vulnerable groups:	Elderly (neglect) Children (child labour in ASM) Women (Gender based violence)

20.4.6.3 Archaeology and Cultural Heritage

According to Enval, the socio-economic surveys did not reveal the presence of archaeological or heritage data (graves, etc.). However, Enval recommended that a Cultural Heritage Management Plan (CHMP) to deal with the rare cases of incidental finds be developed. The CHMP forms part of the ESMP and is presented in Table 20-16.

Table 20-16: Cultural and Heritage Management (CHM)

Objectives	• To protect cultural heritage and archaeological sites. • To ensure that operations do not inhibit traditional land management practises. • To minimize the impact on aesthetic values and the cultural landscape.
Targets	• No unapproved cultural heritage sites and graves to be disturbed. • Prevention of unplanned economic social impacts on the local community. • No adverse effect to traditional land management practises. • No inadvertent impact on aesthetic and cultural values for local communities.
Actions/measures/land management	• SML will maintain a register of sites and these sites will be marked on maps, development plans and advertised at the site (unless the location of the site is deemed confidential by the local community or authorities). • All employees and contractors will be advised of cultural and heritage sites, as well as exclusion zones. • SML employees will adhere to a "IF IN DOUBT – MARK IT OUT" principle to areas of potential new sites. Should an employee identify a potential site he will mark it out and not disturb the site until an archaeological survey can be arranged by the Social Performance department. • Ensure all practises are in line with the ESIA and Social Impact Management Plan (SIMP). • Ensure infrastructure and operations aim at maintaining aesthetic and cultural values for local communities, where practicable. • Designing of waste rock dumps and tailings dams to integrate into the surrounding environment upon closure.
Performance indicators/criteria	• No unauthorized disturbance of archaeological and community designated sacred sites. • No recorded disturbance of cultural heritage sites. • No unplanned economic social impacts on local community. • Level of feedback and concerns raised at meetings on the operations effect on cultural values and traditional land management practises.
Monitoring	• Regular meetings and communication with the community to ensure all practises are in line with ESIAs, SIMP and legal agreements with the community. • On an annually basis the HSE and Social Performance manager will monitor all sites for disturbance. • Notification of any accidental disturbance to the relevant authorities. • Evidence of notification of the new sites to the community and the relevant authorities.
Corrective actions	• A register of all external communications relevant to the mine's operations will be maintained in EDV through ISOMETRIX's Incident and Complaints Register. This will incorporate any details from on-country meetings. • This will lead to an appropriate action to overcome the problem.

Table 20-16: Cultural and Heritage Management (CHM)

Responsibilities	• Project Director • HSE • Social Performance manager
Response timing	• Immediate response to any cultural issues time taken to overcome the problem reflects the nature of the matter.
Review and reporting	• SML will notify the relevant authorities if deemed appropriate, should any new sites be identified. • Any relevant complaints will be reported in the MMP and the community. • SML will maintain mechanisms to update this plan, particularly recognizing new and emerging issues and implementation into the plan. These mechanisms include: – reporting annually to the board on any issues of cultural heritage risk or impact; and – allowing open and honest dialogue with administrative authorities, and traditional owners.
Relevant legislation and standards	• National laws and decrees on cultural heritage management. • Mining license granting condition. • Best Management Practises (BMP) on cultural heritage management.

20.4.6.4 Resettlement

As of March 2022, full inventories for all economically displaced people impacted by the project footprint at the time have been prepared. The initial focus was on those affected by the loss of access to land within the perimeter fence and water harvest dam. These inventories were audited by a third-party (H&B Consulting) and endorsed by the local authorities Dabakala's Prefet and sub-prefet. Initial compensation payments for all affected/impacted parties, was paid between November 2021 and February 2022. Some recipients have elected to receive their payments in instalments, so these will continue until 2023. The associated Livelihood Restoration Plan (LRP) is discussed in Section 25.17.7.2.

In December 2021, additional impacts were identified associated with the extension of the perimeter fence around the TSF and waste dumps to the south, as well as the access road bypass, around the Lafigué village.

This section of the report does not cover resettlement associated with the construction of the new transmission line between Dabakala and the Mine. This is controlled and managed by Compagnie Ivoirienne d'Électricité CI Energies.

20.4.6.5 Artisanal Mining

Available documentation contains several references to a substantial artisanal mining presence in the 'Study Area'.

Prior to the company acquiring the exploitation permit for the Lafigué Mine, there were an anticipated 8000 to 9000 ASM miners on the property. Once the exploitation permit had been granted, Endeavour/SML started enforcing it rights to the property by erecting a fence and moving the miners off the Site, with the help of the Gendarmes from the Dabakala Department.

SML has said that the process evolved relatively peacefully and that they are unaware of; any reports of injuries or deaths, or any cases opened against them, protesting at the way in which this exercise was undertaken.

The miners who were on Site have moved to other areas, or to a site nearby which has been demarcated for exploitation by Endeavour/SML in conjunction with the authorities.

Key dates for these movements are:

- 13 to 17 September 2021: start of the voluntary eviction campaign. The CI security forces met with the gold miners, village chiefs and community representatives. Estimated that 7000 to 8000 miners left the site.
- 20 to 23 September: involvement of the gendarmes, 90 % clearance from future pits and 30 % from the area to be fenced.
- 15 to 17 December 2021: gendarmes assistance, 99 % clearance from future pits and 60% from area to be fenced.
- 23 to 25 January 2022: combination of communication with communities and gold miners, construction of the fence and removal of artisanal miners by the gendarmes, 99% clearance from future pits, and 98 % from fenced areas.
- 27 to 28 April 2022 and 30 April to 3 May 2022: intervention outside the fenced area and on PR 58.

20.4.6.6 Impacts of artisanal mining

Due to the large-scale nature of the previous activities (mining, processing, and accommodation), there has been a substantial impact on the environment. Impacts include; pits which had been used for the cyanide extraction of gold, the use of mercury and physical waste dumps, and excavations, all of which have adversely impacted biodiversity and water resources. The extent of the disturbance and the baseline conditions were mapped by a team from Endeavour/SML as well as government representatives to ensure that the conditions of the site were well defined when Endeavour took over operations, in terms of starting to develop their mining infrastructure. These pits and processing sites have been mapped in reports and by means of photographs.

Some soil and plant samples were taken in July 2022 to determine and map baseline conditions following a recommendation from CIAPOL. CIAPOL were concerned about potential impacts to the population, the environment, and to biodiversity. The soil samples were generally below guideline values, except some for zinc, copper and cyanide.

The results of these investigations were shared with the Prefet and Sub-Prefet of the Dabakala region, CIAPOL, ANDE, and the Department of Geologie.

Current and historical ASM sites as provided by Endeavour are presented in Figure 20-14. It is noted that there are historic ASM sites within the PR 329 and PE 58 permits as well as gazetted artisanal mining lands in close proximity to PR 329 (see Section 23, Adjacent Properties).

The work undertaken to capture the impact of artisanal miners post the ESIA, is discussed in Section 4.

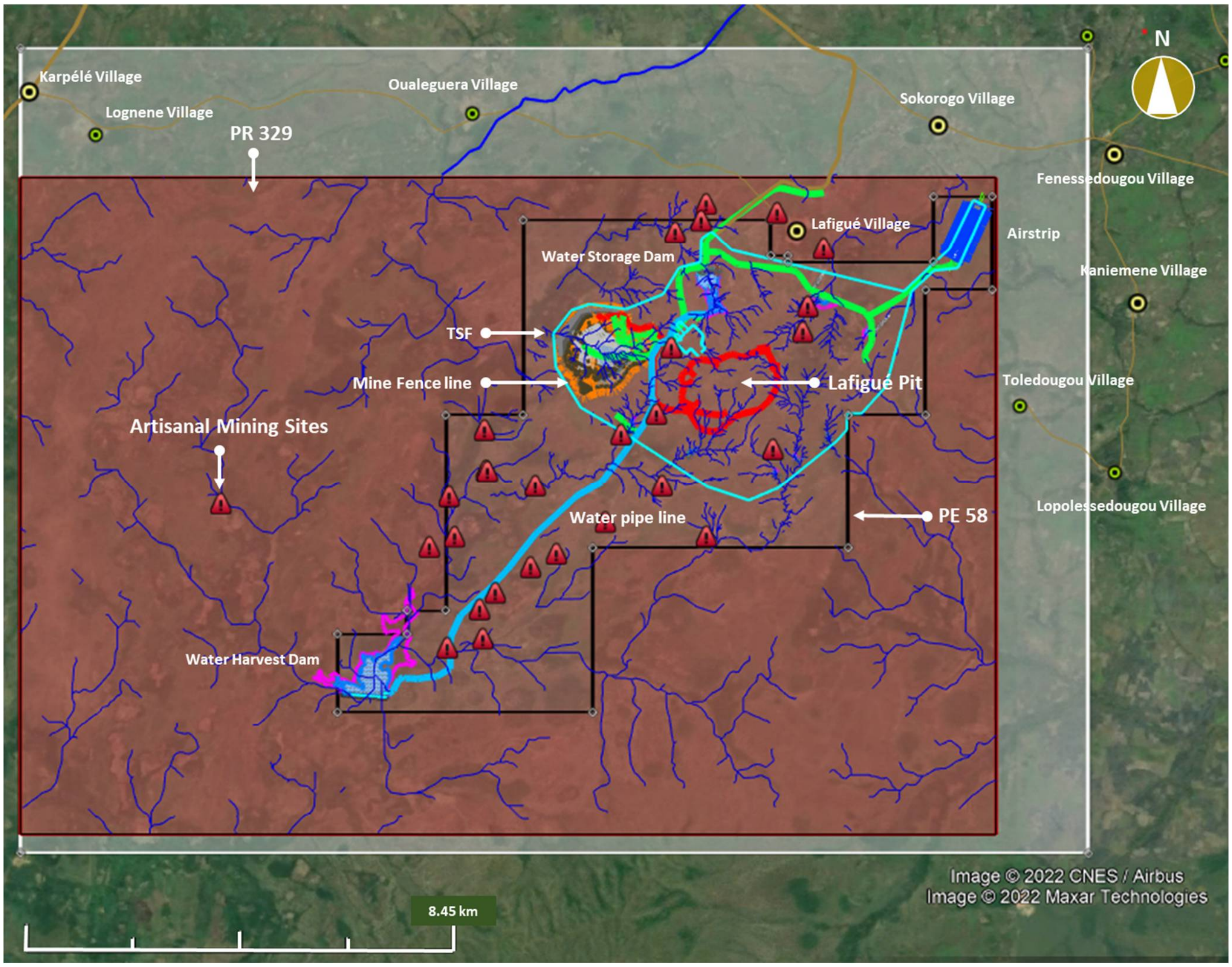


Figure 20-14: Current/Historical Artisanal Mining Sites (Google Earth®), (Endeavour Mining Plc, 2022)

20.4.7 Waste Management

The following section briefly reviews the environmental component of waste management, specifically:

- Waste Rock Geochemistry.
- Tailings Geochemistry.
- General Waste Management.

20.4.7.1 Waste Rock Geochemistry

Waste rock geochemical characterisation was completed by Knight Piésold (KP) to a DFS level of development (Knight Piesold, 2021). The Project is expected to produce some 330 Mt of waste rock and, in terms of understanding potential water related issues, a detailed breakdown of waste rock according to lithology and oxidation was provided to inform the waste rock geochemistry assessment.

Sample Selection and Tests

In order to assign an appropriate number of samples to each waste rock lithology, the total intercept length for each lithology in the drilling database was summed and the percentage of the overall waste rock tonnage determined. This was multiplied by the total waste rock tonnage of 330 Mt, to determine the approximate tonnage of each lithology. Samples were then allocated based on the proportion of each lithology. However, slight modifications to the sample allocation were also made, to ensure a minimum of the three samples for each lithology to allow meaningful interpretation. Minor lithologies (each accounting for less than 0.4% of the total waste) were not sampled as part of this study.

The distribution of waste rock samples by lithology is provided in Table 20-17

Table 20-17: Waste Rock Sample Summary (Knight Piesold, 2021)

Lithology Code	Lithology Description	Logged Intercepts (m)	Lithology Proportion (%)	Sample Allocation (No.)
IMAF	Mafic Intrusion	29 436	59%	50
VMAF	Mafic Volcanic	13 379	27%	35
ARGL	Overburden	3046	6%	10
MBSL	Metavolcanic	1244	2.5%	5
SCHT	Schist	845	1.7%	5
VFEL	Felsic Volcanic	658	1.3%	5
IFEL	Felsic Intrusion	223	0.4%	5
DIOT	Diorite	203	0.4%	5
QZVN	Quartz Vein	161	0.3%	0
NRCV	No recovery	137	0.3%	0
QZTM	Quartz Vein	134	0.3%	0
QZON	Quartz Zone	111	0.2%	0
QRTZ	Quartz Vein / Zone	109	0.2%	0
LATR	Laterite	108	0.2%	0
OVBD	Overburden	95	0.2%	0
ND	Not Logged	83	0.2%	0
MTLZ	Mottled Zone	80	0.2%	0
DYKE	Dyke	40	0.1%	0
IQFP	Quartz Porphyry	19	0.04%	0
TCBC	Mylonite	17	0.03%	0
CHRT	Chert	16	0.03%	0
GRAN	Granite	15	0.03%	0
LATG	Laterite	3.6	0.01%	0
IMM	Mafic Intrusive	1.3	0.002%	0
VCSD	Volcanic-Sedimentary	1.0	0.002%	0
Total		**50 163**	**100%**	**120**

Acid base accounting (ABA) and multi-element (ME) screening on 120 samples, with distilled water extract (DWE) testing conducted on 60 of these samples. The scope of work is considered appropriate for the current design phase. However, additional characterisation will likely be required throughout subsequent design phases, to build up a robust geochemical database, and provide confidence in the waste rock management plans.

Test Results

The acid neutralising capacity (ANC) of the samples was determined along with the estimated carbonate content (based on total carbon in the absence of inorganic carbon test data). The two results (MPA and ANC) can be used as a check against one another and to identify the contribution of ANC from carbonates and other non-carbonate minerals. The ANC/MPA ratios were typically very high, ranging from 1.4 to 850, averaging 129. This indicates a very high factor of safety against acid generation for the majority of samples. Net acid generation (NAG) testwork is a direct measure of a sample's ability to produce acid through sulphide oxidation. The addition of hydrogen peroxide to samples causes rapid oxidation of the contained sulphides to produce sulfuric acid. The results of the NAG test indicate that no samples produced measurable acid under extreme oxidising conditions, with the final pH varying from 5.1 to 11.4. Most samples resulted in a highly alkaline pH with only one sample recording a pH less than 6.5.

The acid formation potential is calculated based on the ABA results and the NAG test results summarised above. The classification of the samples is summarised below:

- 50 samples (42%) classified as Acid Consuming (AC).
- 70 samples (58%) classified as Non-Acid Forming (NAF).

The results of the ABA and NAG test results show that the samples analysed have a very high factor of safety against acid generation.

Distilled water extract tests are conducted to assess the potential for leaching of environmentally significant elements from samples, which could have a detrimental effect on the seepage water quality. To allow assessment of the results of the distilled water extract analyses, three sets of reference values have been established as follows:

- Reference Set 1: Côte d'Ivoire water discharge limits (Centre Ivoirien Antipollution, 2008).
- Reference Set 2: IFC guidelines for release of water from mining operations (IFC, 2007). As these guidelines only cover a limited number of elements, KP has supplemented the guidelines with ANZECC livestock drinking water guidelines (ANZECC, 2000) to allow a more comprehensive assessment.
- Reference Set 3: WHO drinking water standards (World Health Organisation, 2017), supplemented with Australian drinking water standards (NHMRC, 2011) have also been applied to provide a more comprehensive list of assessment criteria.

The results were also compared with world health organisation (WHO) drinking water guidelines, supplemented with Australian drinking water guidelines to cover a wider range of parameters. KP is not aware of any country specific drinking water guidelines for Côte d'Ivoire. Exceedances for metal(loid)s were recorded for both aluminium (23% of samples) and arsenic (23%), with a small number of exceedances for iron (two samples) sulphate (one sample) and antimony (one sample). The average concentration of aluminium from all samples is 0.61 mg/L, less than the guideline value of 0.9 mg/L. The same also applies for arsenic, which has an average concentration of 0.008 mg/L compared to the guideline value of 0.01 mg/L. In addition, pH ranged from 7.3 to 9.7 and was found to be elevated above the upper bound limit of pH 8.5 in over 77% of samples.

Implications for Waste Rock Management

The testwork conducted to date indicates that acid generation from the waste rock is unlikely to be a risk to the project based on the typically low sulphur contents and high ANC values, provided the samples tested to date are representative of the overall waste rock material to be mined. Further, given the significant proportion of carbonate in the waste rock and strong alkalinity, the development of acidic drainage from placement of a limited amount of PAF waste (if encountered) is considered unlikely.

The results of the multi-element analysis and comparison to average crustal abundance indicates that the samples have very low levels of element enrichment. The most commonly enriched element was arsenic recording enrichments in 29% of samples, however, only two samples were highly enriched with a maximum concentration of 141 ppm. The results of the multi-element analysis have also been compared to a set of soil quality screening guidelines, which indicated that the samples met the human health guidelines (i.e. recreational/non-agricultural land use), with around 91% of samples meeting the soil intervention guidelines. However, only a small number of samples are indicated to have met the ecological guidelines.

Based on the multi-element results, it is envisaged that a basic cover system of benign waste and a growth medium will be required to rehabilitate the waste rock landform on closure.

The distilled water extract testing indicated that all but one samples met the reference mining release/surface water quality values for metal(loid)s, although pH was commonly out of compliance due to elevated pH (i.e. alkaline). The average metal(loid) concentrations from all samples met the guideline values. Comparison of the distilled water extract with drinking water guidelines indicated exceedances of aluminium and arsenic for around a quarter of the samples. The average concentrations from all samples met the health-based guideline values (where applicable). However, pH was recorded above the alkaline threshold in around three quarters of samples.

Assuming that these samples are representative of the overall waste rock to be mined, the runoff and seepage flows from waste dumps may be suitable for release, depending on the regional pH regime and regulatory requirements.

20.4.7.2 Tailings Physical and Geochemical Testwork

Physical and geochemical testing of combined tailings samples derived from the different ore bodies was conducted during the study. A fresh ore composite sample 'CIP (Fresh)' described as fresh ore tailings, and an oxide blend composite 'CDO (Blend)' described as 35% oxide/65% fresh were tested. The tailings testing results were incorporated into the water balance modelling (and thus the TSF design) by interpolating tailings properties during the operation based on the process schedule.

CIP (Fresh) consisted of 36% sand, 61% silt[130] and 3% clay. The testing indicates the material is non plastic sandy silt with trace clay. The sample has a P_{80} of 111 µm.

CDO (Blend) consisted of 33% sand, 63% silt and 4% clay. The testing indicates the material is non-plastic sandy silt with trace clay. The sample has a P_{80} of 107 µm.

The grading curves indicate the samples fall inside the boundary for potentially liquefiable soils and therefore liquefaction of the tailings mass was considered in the stability assessment for the TSF.

[130] Silt is classified as particle sizes between (0.002 and 0.075) mm.

CIP (Fresh) tailings sample exhibited very rapid settlement and achieved a moderately high dry density from settlement before air drying or consolidation. The sample achieved a maximum dry density from air drying of 1.46 t/m^3 (adb). The dry density is considered moderate for gold tailings.

CDO (Blend) settled quickly and achieved a moderately high dry density from settlement before air drying or consolidation. The sample achieved a maximum dry density from air drying of 1.51 t/m^3 (adb). The dry density is considered moderate for gold tailings.

The rate of supernatant release for all samples sample was quick and reached typical dry densities, with a good increase due to drying and consolidation. Assuming that the facility is efficiently operated, it is estimated that the average settled density for the tailings mass will be approximately 1.44 t/m^3 (based on density and consolidation modelling completed).

The results of the geochemistry testing are summarised below:

- The tailings samples are considered to be Non-Acid Forming (NAF), with the fresh tailings sample further considered as Acid Consuming (AC). On the basis of these results, there is no perceived risk of the tailings samples generating acid.
- The samples had a low number of enrichments, with the level of enrichment ranging from slight to highly enriched. Boron was the only metal(loid) classified as highly enriched, with bismuth, molybdenum, silver and sulphur recorded as slightly to significantly enriched in the samples.
- Comparison of the multi-element analysis results with soil quality screening guidelines indicated that the samples met the human health criteria (recreational land use). However, the samples did not meet the ecological guidelines due to exceedances in chromium, copper, nickel, sulphur and zinc. In addition, the blended sample met the intervention[131] values, whilst the fresh sample did not (elevated nickel).
- The supernatant from both samples was found to exceed Côte d'Ivoire water discharge guidelines due to elevated total cyanide. Similarly, the blended sample exceeded the IFC guideline values for total cyanide, while the fresh sample met the reference release guidelines. However, neither sample met international drinking water guidelines due to a range of elevated metal(loid)s and total cyanide.

The TSF design incorporates sufficient measures to prevent the discharge of tailings and supernatant liquor from the facility based on the expected tailings geochemistry. It is considered that the number of samples tested (two) is typical for a DFS. Ongoing geochemistry testing of tailings and supernatant water should be conducted during the operation to confirm the proposed closure design.

20.4.7.3 General Waste Management

Good general non-mining/non-process waste management practises for construction and operations, needs to be developed early and refined/improved with time. This continuous improvement will not be limited to the evaluation of waste treatment and disposal processes, but also considers the use of technical solutions for waste reduction at source. The General Waste Management Plan is described in Table 20-18.

[131] Soils with contaminant concentrations exceeding the 'intervention' values require remediation, as the functional properties of the soil for humans, plant and animal life are seriously impaired or threatened. Thus, in a mining context this means that it would not be an acceptable outcome to have materials which have concentrations of contaminants above the intervention values, left exposed on the outer faces of landforms post closure (Thus, would cover the material to isolate it from the environment)

Table 20-18: General Waste Management Plan

Objectives	Achieve the best possible environmental outcome by minimizing waste generation, maximizing waste re-use, maximizing recycling and safely treating and disposing of non-recyclable materials. Prevent wastes from contaminating the surrounding environment.
Targets	Achieve efficient waste management by: • optimizing the processes/products that produce zero or minimal waste requiring disposal. • not contaminating the surrounding environment. • maximizing the principles of avoid, reduce, reuse, and recycle wherever possible. • safely disposing of non-reusable and recyclable materials.
Actions/measures/ land management	Waste management actions will be in accordance with the EDV group waste management procedure (EDV-HSE-V1.01-STA014). That procedure includes as actions: • segregating wastes that are recyclable and reusable, and will endeavor to recycle wastes in appropriate recycling facilities or use on site if applicable. • re using or recycling wastes such as oil, scrap metal and timber pallets while others will be disposed of on-site. • recyclable waste will be regularly collected for recycling, by recycling companies approved by the Ministry of the Environment through CIAPOL where applicable. Waste collection contracts with these companies will be confirmed after verification of the acceptability of their practises in terms of environmental, health and safety management. • utilizing processes/products that produce zero or minimal waste requiring disposal. • utilizing processes/products which minimize contamination of the surrounding environment. • placing contaminated wastes, including materials that have been in contact with lubricants, greases, hydrocarbons and other hazardous chemicals in designated disposal bins for transporting and disposal. • installing sewage treatment facilities. • installing burning/incineration facilities.
Performance indicators criteria	• Appropriate waste management resulting in minimal environmental effects. • Effective recycling and reuse of appropriate materials to reduce resource use.
Monitoring	• Regular inspections/audits of camp and operational areas, to ensure that waste is being managed appropriately. • Records kept of waste disposal activities. • Monitoring will include the recording of waste types and volumes generated on-site (e.g. general waste, contaminated waste, scrap metal and recyclables) and being transported off-site.
Corrective actions	If an incident occurs including improper waste disposal, investigate incident, then instigate procedural change.
Responsibilities	• HSE manager and Environmental Superintendent • Project Manager
Response timing	• Immediate response to any waste management incident. If training is required implement over required training timeframe.
Review and reporting	• Records will be reviewed on a regular basis and appropriate corrective actions formulated to reduce or eliminate waste generation or impacts associated with waste. • Reporting of any incidents internally and to authorities. • Include summary of inspections/audits and waste management activities (including recycling) in annual reports.
Relevant standards and legislation	• Country national laws on environment and health • Mining license granting conditions • Best Management Practises on waste management

20.5 Key Environmental and Social Impacts

20.5.1 Identified Potential Impacts

A list of the environmental and social impacts for the Project are provided in Table 20-19, including the initial significance ratings. It is noted that measures to avoid or reduce impacts inherent to the Project design are reflected in the initial impact rating, however the implementation of additional management measures are not reflected, as no residual impact rating was undertaken.

A detailed list of infrastructure that will be developed as part of the project and that will lead to some of the impacts noted, is provided in Section 18 of this Report.

Table 20-19: Summary of the Identified Potential Impacts

Aspect	Potential Impact	Initial Impact Significance
Development Phase		
Topography and Visual	Alteration of topography and change in landscape as a result of the development of infrastructure.	Minor (negative)
Soils and Land Capability	Loss of topsoil resources through land clearance and potential soil erosion/compaction	Minor (negative)
	Contamination by hydrocarbons	Moderate (negative)
Biodiversity	Reduced biodiversity value from direct and indirect loss of vegetation	Minor (negative)
	Reduced biodiversity value from direct and indirect loss of fauna due to habitat loss	Minor (negative)
Water	Reduced water quality and quantity	Moderate (negative)
Air Quality	Site clearance and construction activities may result in fugitive dust generation which may reduce ambient air quality.	Moderate (negative)
Noise	Increased noise levels from vehicles/machinery may affect nearby sensitive receptors.	Moderate (negative)
Cultural Heritage	Potential direct disturbance/destruction of a places of cultural importance	Minor (negative)
Socio-Economics	Physical and economic displacement of artisanal miners	Major (negative)
	Population influx of speculative job seekers results in increased pressure on natural resources and public infrastructure/services as well as potentially giving rise to social ills, gender based violence (GBV) and illnesses.	Moderate (negative)
	Job creation for the execution of construction activities and associated business opportunities.	Minor (positive)
	Construction activities may increase community health and safety risks and impacts. (accidents)	Moderate (negative)
	Cultural mixing and cohesion	Major (positive)

Table 20-19: Summary of the Identified Potential Impacts

Aspect	Potential Impact	Initial Impact Significance
Construction Phase		
Topography and Visual	Alteration of topography and change in landscape as a result of the development of infrastructure.	Minor (negative)
Soils and Land Capability	Loss of topsoil resources through land clearance and potential soil erosion/compaction	Minor (negative)
	Contamination by hydrocarbons	Moderate (negative)
Biodiversity	Reduced biodiversity value from direct and indirect loss of vegetation	Minor (negative)
	Reduced biodiversity value from direct and indirect loss of fauna due to habitat loss	Minor (negative)
Water	Reduced water quality and quantity	Moderate (negative)
Air Quality	Site clearance and construction activities may result in fugitive dust generation which may reduce ambient air quality.	Moderate (negative)
Noise	Increased noise levels from vehicles/machinery may affect nearby sensitive receptors.	Moderate (negative)
	Population influx of speculative job seekers results in increased pressure on natural resources and public infrastructure/ services as well as potentially giving rise to social ills GBV and illnesses.	Moderate (negative)
	Job creation for the execution of construction activities and associated business opportunities.	Moderate (positive)
	Construction activities may increase community health and safety risks and impacts. (accidents)	Moderate (negative)
Operational Phase		
Topography and Visual	Risk of landslides and destabilisation.	Major (negative)
Soils and Land Capability	Loss of topsoil resource through potential soil erosion/compaction or contamination.	Moderate (negative)
	Soil contamination	Major (negative)
	Modification of the soil structure	Moderate (negative)
Biodiversity	Cyanide deposits in TSF to result accidental injury or death of fauna.	Major (negative)
Surface water	Seepage from mine waste storage facilities affecting surface water resources.	Major (negative)
	Potential AMD contamination.	Major (negative)

Table 20-19: Summary of the Identified Potential Impacts

Aspect	**Potential Impact**	**Initial Impact Significance**
Groundwater	Pit dewatering resulting in groundwater drawdown in the vicinity of the open pits.	Moderate (negative)
	Seepage affecting groundwater resources (AMD).	Major (negative)
	Accidental contamination	Major (negative)
Air Quality	Operational activities may result in fugitive dust generation which may reduce ambient air quality.	Major (negative)
Noise	Increased noise levels from vehicles/machinery which may affected nearby sensitive receptors.	Moderate (negative)
Socio-Economic	Payments of taxes and royalties to the Government.	Major (positive)
	Contribution to local economic development.	Major (positive)
	Direct and indirect employment.	Major (positive)
	Pressure on sanitary infrastructure	Minor (negative)
	Operational activities may increase community health and safety risks and impacts.	Moderate (negative)
	Development of socio-economic infrastructure	Major (positive)
Decommissioning and Closure		
Soils and Land Capability	Loss of topsoil resource through potential soil erosion/compaction or contamination.	Major (negative)
Biodiversity	Reduced biodiversity value from indirect loss of vegetation and associated faunal habitat.	Major (negative)
	Rehabilitating disturbed area and creating habitat for flora and fauna	Major (positive)
Surface Water	Surface water contamination	Major (negative)
Ground Water	Groundwater contamination	Major (negative)
Air Quality	Decommissioning and rehabilitation activities may result in fugitive dust generation which may reduce ambient air quality.	Moderate (negative)
	Improved air quality after mine closure	Major (positive)
Noise	Increased noise levels from vehicles/ machinery which may affected nearby sensitive receptors.	Moderate (negative)
Socio-Economics	Reduction in accidents due to equipment and machinery	Major (positive)

20.5.2 Mineral Resources and Reserves

There is nothing to suggest that environmental and/or social considerations will impact resource and reserve calculations. This statement does not cover any permitting related issues, which are discussed separately in Section 4.

20.5.3 Energy and Greenhouse Gases

From project development to closure, Endeavour forecasts and tracks its energy usage across its operations and across the mine value chain. This enables the group to develop energy/cost reduction strategies, including; reducing energy consumption, and changing the types of energy used.

Endeavour measures and reports annually in its 'Sustainability Report' its Scope 1 emissions (which are direct emissions produced from its operations), Scope 2 emissions (which are indirect emissions that result from the generation of purchased energy such as electricity), and Scope 3 emissions (which are indirect emissions that occur in the value chain, such as the emissions resulting from employee commuting, business travel, refining the gold produced, etc). The Scope 1 to 3 emissions criteria are represented graphically in Figure 20-15.

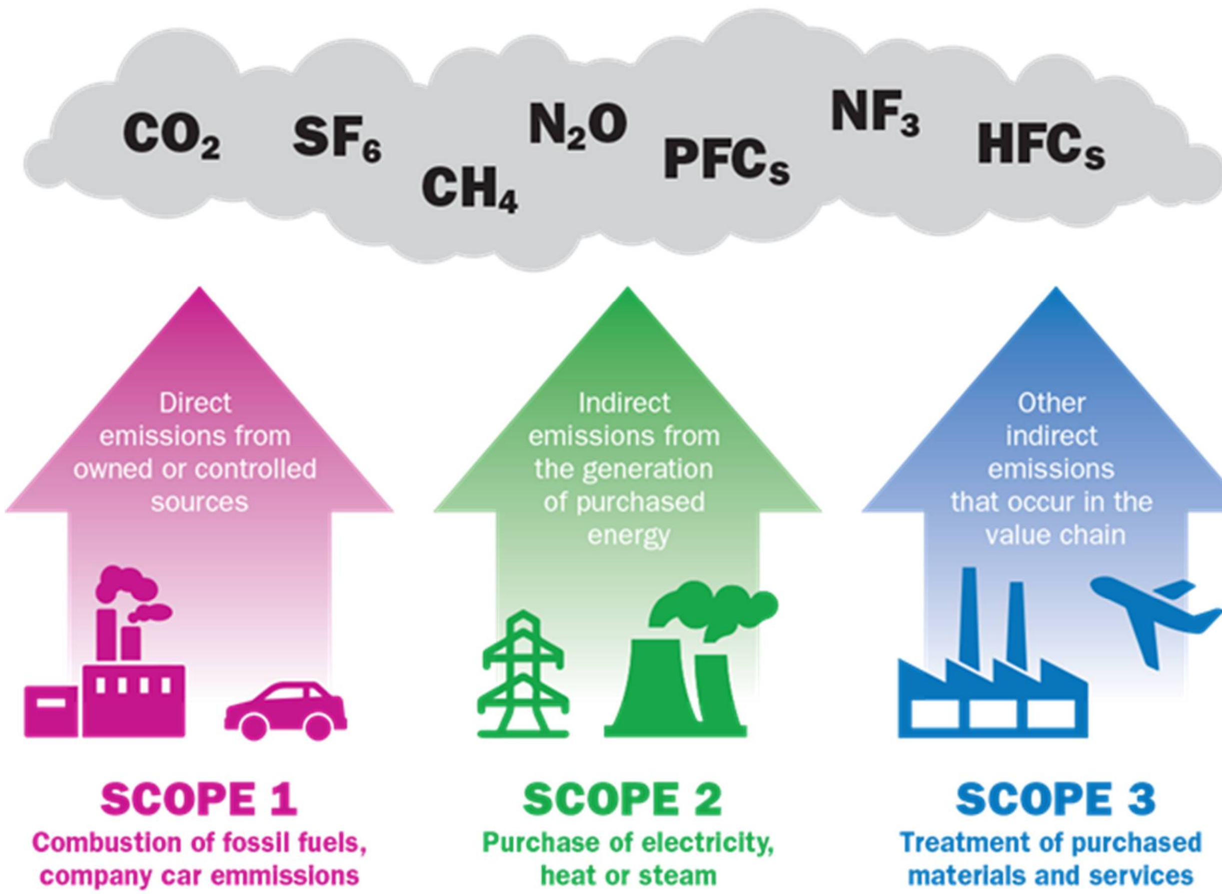


Figure 20-15: GHG Scope 1 to 3 (Endeavour, 2022)

The Project is connected to the national grid, which relies on both renewable and fossil fuels for power generation. Current and future generation sources are discussed in Section 5 of this Report.

Key energy and CO_2 metrics/Key Performance Indicators (KPI's) for the Project and for Operations are summarised in Table 20-20 and detailed by year in Table 20-21.

Table 20-20: Key Energy and Emissions Metrics for the Project and Mine (Issiyakou, 2022), (Thomson, 2022)

Area	Units	PFS	DFS Total/Avg
Production			
• Tonnes mined (Open Pit)	Mt (db)		491.615
• Tonnes Processed (Total)	Mt (db)		49.813
• Gold Produced	koz		2584
Emissions (CO_2)			
• Total (S1 & S2)	kt CO_2-e		1538
• Total (S1 to S3, excluding C2 and C4)	kt CO_2-e		1921
Emissions Intensity (per 'tonne' mined)			
• Total (S1 & S2)	t CO_2-e/t		0.0031
• Total (S1 to S3, excluding C2 and C4)	t CO_2-e/t		0.0039
Emissions Intensity (per oz of gold produced)			
• Total (S1 & S2)	t CO_2-e/oz	0.36	0.595
• Total (S1 to S3, excluding C2 and C4)	t CO_2-e/oz		0.743
Energy used	GJ		18 993 112
• Energy Intensity (per 'tonne' processed)	GJ/t		0.381
• Energy Intensity (per 'oz' of gold produced)	GJ/oz	6.99	7.35

Table 20-20 notes:

- S1 – Scope 1 emissions
- S2 – Scope 2 emissions
- S3 – Scope 3 emissions.
- C2 – Category 2 emissions (Capital Goods) for Scope 3 (not included and/or not applicable)
- C4 – Category 4 emissions (Upstream Transportation & Distribution) for Scope 3 (not included and/or not applicable)
- Scope 3 emissions are forecasts/general estimates only and subject to refinement.

There are number of reasons for the change in metrics between the PFS and DFS, including but not limited to:

- a DFS has a greater level of technical definition than a PFS, particularly around power and energy/fuel usage;
- errors and omissions; and
- the drop in the overall weighted average gold grade between the PFS (2.0 g/t) and DFS (1.69 g/t).

Table 20-21: Lafigué Mine Emissions Forecasts (Construction and Operations) (Issiyakou, 2022)

Description		2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Production																
• Tonnes mined (Open Pit)	Mt (db)	0	0	47.072	54.750	55.480	54.518	55.632	54.385	46.926	46.070	35.816	23.430	9.401	8.097	0.037
• Tonnes Processed (Total)	Mt (db)	0	0	2.279	4.000	4.098	4.000	4.011	4.008	4.000	4.000	4.011	4.000	4.000	4.000	3.405
• Gold Produced	koz	0	0	139	223	177	230	233	231	215	214	240	222	223	192	45
Emissions																
• Total (S1 & S2)	kt CO_2-e	5.75	24.12	117.61	123.34	126.38	130.31	130.42	133.43	134.56	131.04	126.39	112.82	93.48	75.57	72.95
• Total (S1 to S3, excluding C2 and C4)	kt CO_2-e	7.06	29.59	145.68	154.88	158.66	163.40	163.54	167.24	168.58	164.36	158.69	142.20	118.60	96.63	81.58
• Scope 1	kt CO_2-e	*5.75*	*24.12*	*66.31*	*72.04*	*75.07*	*79.00*	*79.12*	*82.13*	*83.26*	*79.74*	*75.08*	*61.52*	*42.18*	*24.27*	*21.30*
• Scope 2	kt CO_2-e	*0.00*	*0.00*	*51.30*	*51.30*	*51.30*	*51.30*	*51.30*	*51.30*	*51.30*	*51.30*	*51.30*	*51.30*	*51.30*	*51.30*	*51.65*
• Scope 3	kt CO_2-e	1.31	5.47	28.07	31.54	32.28	33.09	33.11	33.82	34.02	33.31	32.31	29.38	25.11	21.05	8.63
– Category 1 (Purchase Goods & Services)	*kt CO_2-e*	*0.000*	*0.000*	*3.021*	*5.280*	*5.411*	*5.283*	*5.300*	*5.292*	*5.287*	*5.276*	*5.285*	*5.251*	*5.236*	*5.234*	*3.794*
– Category 2 (Capital Goods)	*kt CO_2-e*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*
– Category 3 (Fuel and Energy Related Products (exc. S1 and S2)	*kt CO_2-e*	*1.306*	*5.471*	*23.822*	*25.008*	*25.632*	*26.559*	*26.560*	*27.273*	*27.489*	*26.789*	*25.772*	*22.880*	*18.627*	*14.575*	*4.831*
– Category 4 (Upstream Transportation & Distribution)	*kt CO_2-e*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*	*0.000*
– Category 6 (Business Travel)	*kt CO_2-e*	*0.000*	*0.000*	*0.787*	*0.787*	*0.787*	*0.787*	*0.787*	*0.787*	*0.787*	*0.787*	*0.787*	*0.787*	*0.787*	*0.787*	*0.000*
– Category 7 (Employee commuting)	*kt CO_2-e*	*0.000*	*0.000*	*0.414*	*0.418*	*0.414*	*0.414*	*0.414*	*0.414*	*0.414*	*0.414*	*0.414*	*0.414*	*0.414*	*0.414*	*0.000*
– Category 9 (Downstream transport & distribution)	*kt CO_2-e*	*0.000*	*0.000*	*0.021*	*0.033*	*0.026*	*0.034*	*0.034*	*0.034*	*0.032*	*0.032*	*0.035*	*0.033*	*0.033*	*0.028*	*0.000*
– Category 10 (processing of sold products)	*kt CO_2-e*	*0.000*	*0.000*	*0.009*	*0.014*	*0.011*	*0.015*	*0.015*	*0.015*	*0.014*	*0.014*	*0.015*	*0.014*	*0.014*	*0.012*	*0.003*

Table 20-21: Lafigué Mine Emissions Forecasts (Construction and Operations) (Issiyakou, 2022)

Description		2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Emissions Intensity (Total Mined)																
• Total (S1 & S2)	t CO_2-e/t	0.000	0.000	0.0025	0.0023	0.0023	0.0024	0.0023	0.0025	0.0029	0.0028	0.0035	0.0048	0.0099	0.0093	1.9476
• Total (S1 to S3, excluding C2 and C4)	t CO_2-e/t	0.000	0.000	0.0031	0.0028	0.0029	0.0030	0.0029	0.0031	0.0036	0.0036	0.0044	0.0061	0.0126	0.0119	2.1779
Emissions Intensity (per oz of Gold Produced)																
• Total (S1 & S2)	t CO_2-e/oz	0.000	0.000	0.845	0.554	0.714	0.566	0.559	0.578	0.625	0.611	0.527	0.508	0.420	0.393	1.613
• Total (S1 to S3, excluding C2 and C4)	t CO_2-e/oz	0.000	0.000	1.047	0.696	0.896	0.710	0.701	0.725	0.784	0.766	0.662	0.640	0.533	0.503	1.804
Energy Consumption																
• Consumed (Total)	GJ	80 373	80 373	1 471 027	1 554 873	1 599 503	1 653 161	1 655 683	1 696 518	1 713 701	1 661 303	1 594 919	1 399 267	1 124 840	874 555	833 015
• Self-generated energy (S1)	GJ	80 373	80 373	938 933	1 022 779	1 067 409	1 121 067	1 123 589	1 164 424	1 181 607	1 129 208	1 062 824	867 173	592 746	342 461	297 320
• Purchased energy (S2)	GJ			532 094	532 094	532 094	532 094	532 094	532 094	532 094	532 094	532 094	532 094	532 094	532 094	535 694
Consumed Energy per 'tonne' processed (S1 and S2)	GJ/t	0.000	0.000	0.645	0.389	0.390	0.413	0.413	0.423	0.428	0.415	0.398	0.350	0.281	0.219	0.245
Consumed Energy per 'oz' of gold produced (S1 and S2)	GJ/oz	0.00	0.00	10.57	6.98	9.03	7.18	7.10	7.35	7.97	7.75	6.65	6.30	5.05	4.55	18.42

Table 20-21 notes:

- S1 - Scope 1 emissions.
- S2 - Scope 2 emissions.
- S3 - Scope 3 emissions.
- C2 - Category 2 emissions (Capital Goods) for Scope 3 (zero values, thus excluded).
- C4 - Category 4 emissions (Upstream Transportation & Distribution) for Scope 3 (zero values, thus excluded).
- Scope 3 emissions are forecasts/estimates only and subject to refinement/change.
- Table by (Issiyakou, 2022) is based in part on DFS data provided by Lycopodium covering; power, diesel usage, reagents and consumables, transport of goods, people and products et.al. (Thomsoon, 2022b).

20.5.4 Environmental Impact Statement

Site clearance for the establishment of the mine infrastructure is the source of several potential impacts, resulting in the direct loss of undisturbed areas with consequences to the landscape, terrestrial biodiversity, water resources and air quality (dust).

Clearing vegetation will have a direct impact on the wildlife species present by removing all or part of their habitat. Some species may slightly alter their habitat use; however, other territorial species will be more intensely affected by these activities. This could lead to the loss of life.

General construction work will generate noise pollution and will impact air quality (dust generation and the release of other atmospheric pollutants). Nearby residents and road users could potentially be locally affected by dust and fumes from machinery.

General construction and mining may lead to the modification of the soil profile, the formation of gullies, the instability of the slopes, and compaction while the loss of crops will have an impact on the local communities. The possible accidental releases of waste, namely; oil, grease, hydrocarbons and other contaminants from construction and operational activities may impact soil and water resources.

Open pit mining in the operational phase of the mine's life cycle will have significant impact on the landscape. In terms of nuisance impact, air quality, noise and visual are expected issue, but may be reduced with appropriate mitigation measures. Significant impacts are related to water and include deterioration of surface and groundwater quality and quantity.

Positive impacts associated with the implementation of this Project include direct and indirect employment, training and skills development, and payments of taxes and royalties, all of which, contribute to the improvement of the local economy directly and indirectly.

20.5.5 Rehabilitation and Closure Cost Assessment

The legislation of Côte d'Ivoire provides for provisions regarding the cessation of mining works and site rehabilitation. Articles 144, 145, 146, 147 and 148 of Law No. 2014-138 of 24 March 2014 on the Mining Code specify all the provisions relating to the rehabilitation and closure of a mine.

All mining projects must complete a Mine Rehabilitation and Closure Plan (MRCP), which is incorporated in the ESIA. The MRCP ensures that rehabilitation is considered early on in the planning process and an adequate Closure Bond is determined and applied to the developer.

The commercial/payment terms of the Closure Bond are defined in Section 4 of this Report.

The cost estimate for the Closure Bond was developed at a conceptual level, using the Endeavour standard closure cost model and the PFS mine plan and layouts. The MRCP considers a two-year 'Pre-closure Stage' (2034 to 2035), followed by a 'Closure Stage' (2036 to 2037), and a final 'Post Closure Stage' that starts in 2038 and runs for a period of five years (2038 to 2042).

The holder of the exploitation permit retains civil liability for damages and accidents that could be caused by the mine over the five (5) year 'Post Closure Stage'.

Additional studies and characterization of the impacted area, will provide additional information that will allow the evaluation and detail of proposed closure actions that should be incorporated in future MRCPs.[132]

The MRCP typically considers:

- cleaning of the operating site;
- dismantling and removal of mining facilities;
- on-site treatment and rehabilitation;
- post-rehabilitation monitoring of the site;
- conversion possibilities of the site to other agreed upon land uses; and
- official transfer of the site to the competent authorities.

The closure costs are calculated to be of the order of USD 24.4 M, with a cost breakdown presented in Table 20-22 (Endeavour, 2022b). The basis for the costs presented are summarised below (Endeavour, 2022a).

- Pits:
  - Limit access to the pit area.
  - Construction of drainage channels to manage surface water and provide a safety berm around the pit.
  - Revegetation of the resloped berm, as well as the area between pit and safety berm.
  - Reshaping of the upper bench is generally recommended but is subject to a geotechnical assessment.
- Waste Rock Dumps:
  - Geotechnical stabilization of lower slopes.
  - Construction of drainage channels to manage surface water.
  - Cover with growth material and revegetation of the surface.
  - Progressive rehabilitation will be started during operations, leaving only the final closure activities at the end. For instance, only the last 20 m bench is to be resloped for the waste dumps, assuming the lower benches were rehabilitated as the open pit operations progressed.
- Tailings Storage Facilities:
  - Embankment reshaping to minimize infiltration and promote water drainage, soil cover of final tailings surface and embankment and revegetation.
  - It has been assumed that the tailings are non-acid generating and non-leachable.
- Water Management:
  - Rock cladding of the embankment area and improvements on the diversion channels.
  - Decommission, dismantle, and restore all drainage/pollution control basins and, unless proven useful post-closure, backfill, grade, and revegetate the basins.

[132] To be updated every three years after the start of production.

- Infrastructure:

  Benchmarked against similar site estimates for: decommissioning, demolition and disposal of buildings and equipment; concrete demolition and metals dismantling; remediation of impacted soil and groundwater and reshaping and revegetation of the area.

- Roads:
  - Selected roads that belong to the mine and can be used by the community, will be offered to the government or respective local authority.
  - All roads and disturbed roads that are no longer needed will be rehabilitated. This will include removal of drains and culverts, ripping to a depth of 25 to 50 cm and revegetation with soil cover and native vegetation. It also includes a characterisation of the soil to determine the present of potential contaminants.
- Supporting Rehabilitation Facilities:
  - Provision of a revegetation nursery.
- Contingency:
  - A 20% contingency was applied on the direct costs, based on the level of technical definition/development.
  - As the estimations are refined with investigations, trials, and other studies, the contingency allowance will be reassessed. The closure stage of the Mine also has an impact on the contingency, which should decrease; as the Site is getting closer to its planned closure date, the MRCP gets operationalised, and progressive rehabilitation gets implemented.
- Studies and Research:
  - Benchmarked against similar site estimates.
  - Studies associated with deconstruction activities.
- Post Closure/Maintenance & Monitoring:
  - Monitoring program benchmarked against similar sites.
  - Five-year period for post-closure monitoring as per CI National legislation.
- Owner's costs:
  - Benchmarked against similar sites.
  - 10% on direct costs for general EPCM fees.
  - Indirect costs associated with deconstruction activities benchmarked against similar sites.
- Exclusions:
  - Update to latest DFS layouts/plans.
  - No social costs nor retrenchment costs are included in the Closure estimate.
  - No inflation or discount rates have been applied.

Table 20-22: PFS Mine Closure Cost (USD (M))

Direct Costs	**13.82**
• Pits	0.45
• Waste Dumps, Stockpiles	1.69
• Storage Facilities-Tailings, Rejects, Slimes	2.28
• Water Management	0.40
• Buildings and Infrastructure	4.75
• Processing Plant	1.81
• Roads and other disturbed areas	0.07
• Site Wide	0.06
• Social Cost	0
• Owners Cost	0
• Contingency	2.30
Indirect Costs	**10.60**
• Studies and Research	2.40
• Post Closure/Maintenance & Monitoring	2.21
• Social Costs	0
• Owner Costs	5.98
• Retrenchment costs	excluded[133]
Total	**24.39**

20.6 Stakeholder Engagement

20.6.1 Overview

One of the objectives of the Social Performance Management System is to minimise risks to Endeavour/SML and its stakeholders, by building trusting relationships to prevent conflicts, sharing information, and addressing issues that arise through ongoing dialogue.

In order to achieve this objective, the following principles apply:

- Identify and understand the views, influences, and interests of key stakeholder groups.
- Ensure that stakeholders understand the activities and impacts of Endeavour's/SML's operations.
- Ensure that Endeavour/SML addresses and responds to stakeholder concerns and grievances and takes stakeholder perspectives into account when making decisions and designing plans that may affect them.

[133] Estimated to be USD 2.1 M for the DFS.

20.6.2 Guiding Principles of Stakeholder Engagement

The principles that guide community engagement for Endeavour/SML are as noted below:

- Inclusion:
  - The community shall be engaged in the participatory process that promotes inclusion.
  - Ensure that dominant interest groups are not the only voices heard.
- Respect:
  - Respect the time and contribution (opinions, contributions, points of view) of the participants during the engagement process, while considering the needs of the community as a whole.
  - Be sincere and respect the commitments made to the communities.
  - Respect the culture of the host population and avoid any language, speech, discussion, or action that might offend their culture or beliefs.
- Transparency:
  - Promote open and honest communication when interacting with the community and/or stakeholders.
- Communication:
  - Ensure that communities have all the necessary information they need, as soon as possible, to provide informed feedback and recommendations.
  - Communicate openly, honestly, and responsibly with those with whom we seek to engage and create channels so that stakeholders can contact the mine based on their own needs.
  - Avoid creating false expectations about what community engagement can achieve.
  - Communicate well with peers and avoid duplication process.

20.6.3 Engagement with Communities

With respect to engagement with local communities, the following principles are applied.

- Ensure that vulnerable groups are identified and considered in all engagement activities and ensure that engagement is culturally appropriate and accessible to all stakeholders.
- Identify and analyse stakeholders and community dynamics and review at least once a year with key internal stakeholders.
- Assess the ability of stakeholders to engage significantly and work collaboratively to build this capacity over time.
- Establish a formal engagement mechanism for stakeholders to provide feedback and receive feedback on sustainable development and management issues.
- Ensure that this mechanism is shared between the Company and all stakeholders.
- Develop an annual stakeholder engagement plan with objectives, target groups and activities, based on operational activities, risks, impacts and opportunities, and stakeholder input.
- Consult with relevant departments to identify current and planned activities and potential issues.
- Ensure that community engagement activities are well planned, coordinated, accessible and inclusive and provide reasonable deadlines for contributing to all these activities.

- Clarify internal and external engagement objectives; how stakeholder contributions will be considered for relevant activities and decisions.
- Record the engagement activities as well as the issues/risks identified in these activities.
- Ensure that issues are monitored and addressed internally, and that feedback is provided to stakeholders.
- Conduct perception surveys at least every two years.
- Obtain and maintain broad support from the relevant stakeholder's community for major projects, based on the principles of free, prior, and informed consent.
- Produce internal reports on the implementation of the engagement plan, community perception, issues, and risks.
- Produce external reports and feedback on managing risks, impacts and opportunities, how their contribution was considered and how it influenced the outcome.

All engagement activities of stakeholders subject to payment of a per diem shall be in accordance with the procedure 'EDV Anti-Bribery and Corruption (ABC) Procedure – per diem Allowances_2021'.

20.6.4 Engagement with Government

Any engagement with Government must be conducted through official channels and in accordance with the laws and regulations. All engagement activities of stakeholders subject to payment of a per diem shall be in accordance with the procedure 'EDV-ABC Procedure – per diem Allowances_2021'.

20.6.5 Engagement with Entities Providing a Mine Support Services Function

The entities providing a mine support services function must be aligned with Endeavour's/SML's policies in terms of stakeholder engagement as well as Endeavour's/SML's 'ABC procedure'. Significant training is often required to align Endeavour's/SML's expectations with that of the service provider. Service level reviews with respect to quality and adherence to standards and KPIs are undertaken quarterly

20.6.6 Other Interested and Affected Parties

Stakeholder mapping and analysis is carried out to establish stakeholders by category, interest, influence. Subsequently, a specific engagement plan dedicated to each identified group is elaborated and implemented, with relevant actions taken to make the process of engaging these stakeholders fluent and effective.

20.6.7 Non-Governmental Organisations and Pressure Groups

Transparency is the key principle in Endeavour's/SML's relationship with NGO's and pressure groups. There are no issues to report.

20.6.8 Grievance Procedure

Addressing complaints is an integral part of the stakeholder dialogue. Even if impacts and mitigation measures have been identified during the ESIA process, unexpected impacts and complaints may arise. Endeavour/SML has in place a known and accessible grievance mechanism for stakeholders to submit complaints in a manner that ensures that they are effectively addressed, thereby maintaining confidence in the process, and continuing effective dialogue toward amicable resolutions.

The livelihood restoration plan (LRP) project's complaint management system complies with Endeavour's/SML's Grievance Standard Operating Procedures.

In addition, an external resolution mechanism will be set up with the support of the administrative authorities to deal with complaints that are not resolved amicably (the 'Conciliation Commission').

20.7 Environmental and Social Management Plans

The Environmental and Social Management Plan (ESMP) allows the implementation and monitoring of ESIA measures to remove, reduce, and possibly compensate for the consequences of the Project and operations on the environment and community. LMCI's now SML's ESMP that formed part of the ESIA submission, was approved by the Government. Subsequent to this, an internal ESMP that builds upon the approved ESMP, is in the process of being approved for use.

These plans reflect Endeavour's commitment to universal precautionary principles and constitutes proof of the consideration of the demands from the Ivorian authorities and international bodies.

The various Environmental and Social plans to be put in place for the Project and for operations, and their associated development status is presented in Table 20-23, following.

Table 20-23: SML Environmental and Social Management Plans

Management Plan	Description	Status
Biodiversity	With regards to the management of biodiversity, the plan aims to protect natural environments, terrestrial and aquatic flora, and fauna, to limit disturbances of natural habitat resulting from the activities of the Project and to protect endangered and protected species, critical habitats and areas housing certain plant species listed in the IUCN Red List as threatened nationally and internationally. Biodiversity management must consider land clearing permit establishment, bypassing sensitive environments and sites, minimizing the physical footprint of the Project in sensitive and natural habitats, and promote works and activities in areas with a lower ecological value. The plan must be integrating land disturbance optimisation in the global mine planning activities, setting clear objectives for preservation and protection of sensitive biodiversity areas. The plan must also consider compliance to international standards that endeavour mining corporation has subscribed to.	In progress
Water	The water management plan will help to limit the consequence on surface water, soil and groundwater (anti-erosion measures and good practises, management of hazardous products, effluents and waste, etc.), to promote earthworks activity to the dry season period and in hydromorphic zones. The water management plan will cover the protection of soil, surface water and groundwater and will include measures to monitor water consumption, reduce the impact of drainage water coming from working areas, and address the acid generating potential of material. This plan will also deal with wastewater and the appropriate method of treatment of the effluent before releasing into the environment. The water management plan will cover; dewatering and runoff management, spill prevention measures, and treatment plans in the event of a spill to limit the potential impacts on soil and water (especially in case of hydrocarbon spill). The integrated water management plan will aim at reducing water consumption of the site, allowing the control of site's water balance, avoiding competition between the operations needs and the community rights in term of access to water; develop a wastewater treatment system, with the goal to monitor the quality of the water discharged to ensure compliance with applicable standards.	In progress

Table 20-23: SML Environmental and Social Management Plans

Management Plan	Description	Status
Topsoil	Topsoil management is a key component of the reclamation tools required for the successful implementation of the progressive rehabilitation programs at SML. This procedure shall be used in close conjunction with Procedure for Land Clearing/Land Disturbance. Topsoil recovery should be planned to allow the stockpiling/re-handling of the topsoil (as necessary after stripping) to be completed as soon as practicable after the stripping/recover operations. An approved SML Land Clearing/Land Disturbance is required prior to any land disturbance activities. The Permit may contain conditions regarding topsoil recovery and management. The HOD responsible for the project will ensure that the permit conditions are followed during clearing and topsoil recovery activities. Topsoil stripping and stockpiling on areas involving the Construction Department shall be done under close supervision to ensure that the standards and conditions of the permit are followed and will arrange and coordinate equipment necessary for the stripping and stockpiling operations. The stripped material will be piled at locations that will allow ready haulage to the stockpile sites and prevent erosion of the material by rainstorms. As much as practicable, all stripped topsoil material shall be hauled away from the stripped site before the proposed project operation commences. If necessary, because of safety or high-cost reasons, the HSE Manager may be consulted to allow a deviation from the requirement. The topsoil shall be stockpiled at areas that have been indicated on the clearing permit or decided in the field through agreements between the HSE Manager or Environmental Superintendent and the Department responsible for the topsoil movement. Topsoil stockpiles will be constructed in such a way as to minimize the potential for erosion of topsoil by storm water and to preserve the biological quality of the materials. Wherever practical, the pile heights should not be more than 2 m. The SML Supervisor/Superintendent/Manager responsible for the project may need to establish stockpile specific plans for design/construction including storm water controls (storm water diversion, windrows, grading, etc.) as required. The side slopes of stockpiles shall not be steeper than 4H:1V (1 vertical in 4 horizontal). Haul trucks or heavy equipment shall not be allowed to travel over the top of topsoil being stockpiled. Compaction will result in the destruction of important biological and physical properties of the soil. Topsoil stockpiles will be inspected regularly to ensure that: no unauthorized use of the topsoil has occurred; and that erosion or storm water control features remain functional. Repair of protective features will be initiated immediately if found to require maintenance for proper erosion controls. The department carrying out the project shall ensure that the stockpiled topsoil is surveyed, with the results made available to the HSE Department. The HSE Department will maintain records of locations and volumes available.	Approved
Hazardous Substances	The hazardous substances management plan will: • Help ensure a safe handling of hazardous substances; • Help prevent the infiltration of pollutants into surface and groundwater; • Promote the substitution of hazardous products by an equivalent but less dangerous products as far as possible; • Help in identifying all hazardous chemical substances used and stored on the site; • Promote the installation of refuelling station with an impoundment system and spill recovery; • Ensure that a proper maintenance system of fuel tanks is in place; • Avoid the installation of underground tanks or piping for the storage or delivery of hydrocarbons or other dangerous products;	In progress, only Hydrocarbon Management procedure is approved

Table 20-23: SML Environmental and Social Management Plans

Management Plan	Description	Status
	• Ensure that a regular inspection schedule of hazardous products network is in place (supply pipes, connection pipes), is conducted by qualified employees and that a log of inspection reports is available; and • Ensure that an adequate training program is offered to employees, in order to maintain good practises in terms of storage and handling of hazardous products in order to prevent the risks associated.	
Air quality	The air quality management plan covers the management of dust induced by mining activities and construction work and the management of atmospheric emissions from potential diesel power generators. The air quality management plan involves the establishment of a program to measure the air quality in the initial state during the pre-construction and construction phases, in order to obtain the baseline data of the air quality in the area over a period of at least one year (to characterise the potential seasonal variability). The air quality management plan will include in particular: • mapping of dust sources; and • identification of sensitive receptors and areas where air quality must be particularly monitored and controlled. To prevent excessive air quality degradation, mitigation measures will be put in place, such as use of dust suppression systems at the processing plant, watering the haulage roads, stockpiles during activities, blast muckpile, limiting the trucks speed at sensitive areas (to avoid excessive dust emissions in those areas), and ensure that all equipment, vehicles, and machinery are regularly maintained and kept in good working order.	In progress
Noise	The purpose of the noise management plan is to integrate into the ESMP the noise management that is part of the HSE regulations and guidelines of IFC. It concerns the noise generated by all construction and operation activities of the Project, in particular at the places closest to the villages. The plan will identify the Project activities and places in which noise-related impacts are to be anticipated, and what are the sensitive receptors and areas where noise must be particularly controlled. Noise management will also involve keeping equipment in good condition, restricting some operational hours our working areas during sensitive times and will define a clear mechanism for handling complaints about noise.	In progress
General Wastes	The waste management plan is an operational process to be implemented and continuously improved on the basis of experience feedback. This continuous improvement will not be limited to the evaluation of waste treatment and means of disposal, but will also focus on the use of technical solutions for reducing waste generation. The waste management system will be updated in order to identify the consumption of products, to have traceability of the waste disposal process and to identify overconsumption. The plan will aim at quantifying all the waste that is generated, and providing guidance for disposal (type and volumes), allow safe handling of both hazardous and non-hazardous waste on site. The plan will be implemented through waste avoidance, minimization and segregation at source, temporary storage, transport, reuse, recycling and final disposal. This includes the appropriate management of any hazardous (restricted) or special wastes that may be generated. The plan will establish targets for reducing the quantities of waste production, this in accordance with health and safety rules and the plan for the management of hazardous substances and the prevention of spills. Dedicated waste storage centres will be built and designed to prevent liquid waste from seeping into the ground, these sites will be fenced, well maintained, clean, with waste separated by type and risk classification, in order to limit pollution risks, proliferation of vermin. When waste is sent off-site, suitable transport vehicles will be used (possibly using a service provider) in order to comply with the regulation and ensure the load is safe and correctly labelled.	Approved
Traffic and Transport	The transport management plan describes the principles of the procedure for road and river transport, during the construction and operation phase and relates to the transport of construction materials & equipment, transport of personnel, and transport of consumables.	Not started

Table 20-23: SML Environmental and Social Management Plans

Management Plan	Description	Status
	The standards applicable to the transport of freight and personnel will be respected, included maximum load, compliance with speed limits on roads and tracks, special measure in place for oversized loads (escort vehicles leading and tailing the convoy, coordination with local authorities to agree on the routes to be taken and when). The personnel will be trained on a regular basis in road safety, and traffic code in the country. In the event of a traffic accident involving one of the Project vehicles, the manager responsible for supervising the activity will inform the emergency services as soon as possible. It is expected that outside of public roads, the emergency vehicle dedicated to the Project will be mobilised to the accident area, while on the public road the public emergency services will be involved. The details of the incident will be recorded in a specific incident report.	
Hazardous Substances and Spill Response	The hazardous substances management and spill response plan aims at reducing the impacts of any accidental spills of hazardous products into the environment, in the event of a leak or breach from the storage area. A spill management plan and a fire prevention management plan will be defined. These plans will organise a systematic, rapid and effective response to any type of emergency, fire, explosion, accident or spill of water contaminated by hydrocarbons or any other hazardous product. These management plan will define how to contain as fast as possible any spill and rapid clean-up of any contaminated area. The plan will define the roles and responsibilities of employees and contractors that are part of emergency response team and will help in conducting drills to increase the team's readiness to handle emergency situations. Part of the management plan will include personnel training and awareness, and will specify the requirements in terms of periodic exercises, as well as frequency of checks and maintenance activities of emergency response team.	In progress
Emergency Prevention and Response	The emergency prevention and response plan will define the response and communication procedures to be followed in the event of an emergency or natural disaster. It will outline the on-site intervention process related to both construction and operating activities (road accident, explosion, fires, medical emergencies, etc.). It will set planned drills to increase the team's readiness to handle emergency situations. It is designed to reduce employee exposure to risks and injuries and limit potential impacts on the environment and the community in emergency situations. The prevention plan will include all possible emergency situations such as fires or explosions, medical emergencies, the transport of dangerous products, climatic phenomena, natural disasters, social and political tensions. This plan will also address community emergency preparedness and will be disclosed in a culturally appropriate manner to all communities in the social area of influence of the Project.	In progress
Social	The specific social management plans proposed for the Project are the following: • Livelihood restoration plan. • Working conditions and information management plan. • Local hiring and procurement plan. • Stakeholder engagement plan. • Grievance handling management plan. • Chance finds and cultural heritage management plan • Health and safety management plan. • Local assets management plan.	Approved
Livelihoods Restoration	The Project is likely to generate some economic displacement rather than physical displacement of camps or villages. Some lands are used as a means of subsistence by local communities as agricultural land, pastures or for hunting and gathering of forest products. The risks associated with economic displacement are mainly loss of revenue and impoverishment, social disruption due to loss of cultural identity and changes in family structure.	Approved

Table 20-23: SML Environmental and Social Management Plans

Management Plan	Description	Status
	In this context, a Livelihood Restoration Plan will be developed to ensure the restoration of the livelihoods of economically displaced households. The plan will include land and crop compensation and additional measures necessary to enable the restoration of the lost economic activity. Endeavour will follow guidance from IFC Performance Standard 5 for the development and implementation of the livelihood restoration programme. The main objective is to empower the impacted communities in a sustainable manner.	
Working Conditions and Information Management	This plan will detail the measures put in place to ensure the working conditions are in accordance with local regulations and international standards, through the Responsible Gold Mine Principles (RGMP) in order to show to all relevant stakeholders, that the gold is responsibly mined and sourced. The objective of this plan is to; ensure fair treatment of workers; fight against discrimination, protect the workers against the use of forced and child labour; and, promote healthy and safe working and housing conditions for workers. The plan guarantees good working conditions for the employees and the subcontracting companies by; including in the terms and conditions the standards to be respected and by providing for corrective measures in the event of non-compliance with the standards in place.	Not defined
Local Hiring and Procurement	This plan refers to the principles of social performance to which Endeavour adheres and intends to contribute to local sustainable development through its activities. The objectives of this plan are to optimize the positive social impacts of the Project through the implementation of a local procurement policy, a preferential hiring procedure in local communities for unskilled jobs or provide training to make people more marketable and a preference for the use of local businesses. Endeavour's hiring policy will promote local employment and local or national procurement. Monitoring indicators will be developed in this regard.	Approved
Stakeholder Management	The stakeholder engagement plan was developed as part of the Project by ENVAL and by Trust International, in strict compliance with Endeavour's policy on community engagement. This plan describes the consultations and the results of the consultations held as part of the Project's ESIA. It is used to structure the communication and consultation activities carried out and to plan the consultations to be carried out within the framework of the Project. Its objective is to ensure a continuous and transparent dialogue with the stakeholders of the Project during its various phases of development. The main objectives are to promote and maintain an open and respectful dialogue between the stakeholders and the Project, to identify the stakeholders, their interests, concerns and influences in relation to Project activities. The goal of the plan is also to provide to the stakeholders the information on the development of the Project and access to information and depending on the potential impacts of the Project, to give stakeholders the opportunity to communicate their opinions and fears through consultations and then preventing conflicts and developing a good relationship with the various stakeholders.	Approved
Grievance Management	The grievance management and resolution plan is a good international practise in order to ensure that that the Project and the operation, consider the population's complaints, and that the population has easy and free access to file a complaint. This plan includes information on the means provided to the population to file their complaints and on the procedures and deadlines for their follow-up and resolution. Good communication to the population, on a regular basis, will ensure its functionality. The plan will also be guaranteeing possibilities to elevate grievances for which any agreement or common understanding was not found between the parties, such as arbitration chamber or court. The goal is to have in place a framework for transparent communication and peaceful prevention or management of conflicts.	Approved
Health and Safety	This plan will include the procedures, infrastructure and means implemented to ensure the hygiene, health and safety of workers and the local population during the construction and operation phase. The health & safety management system ensures the identification and elimination of hazards, the prevention of accidents and incidents, the assessment of the various health and safety risks to which workers are exposed according to the type of activity.	Approved

Table 20-23: SML Environmental and Social Management Plans

Management Plan	Description	Status
	This plan will also deal with the impact on local communities of the risks associated with the addition of a relatively important construction workforce over a short period of time (construction activities, exposure to diseases from construction workers, increased traffic). The health and safety management plan will include an analysis of these different risks and detail the mitigation and management measures planned. The Project is also committed to promoting the prevention of communicable diseases regarding the prevention of HIV and malaria.	
Compensation and Resettlement	This plan encompasses the entire process of land acquisition and includes both physical and economic displacement. It is often the biggest impact on local communities and completing a successful resettlement is very difficult. It can pose a great risk to the company. For these reasons, resettlement can be considered one of the most important social issues and there is in place a policy regulating this matter within Endeavour. The main objectives are to reduce the risks to Endeavour Mining Corporation (Endeavour, EDV) and the impacts to other parties. In order to achieve these objectives, EDV will: • Avoid, and wherever possible limit, involuntary resettlement by considering alternative project designs. • Avoid forced eviction and prefer a consensual approach by obtaining stakeholder approval. • Anticipate and avoid, or where avoidance is not possible, mitigate adverse social and economic impacts resulting from land acquisition or restrictions on land use by: (i) providing compensation for loss of assets at full replacement cost and by (ii) ensuring that resettlement activities are accompanied by appropriate and timely disclosure of information and informed consultation and participation of those affected. • Improve or at least restore the livelihoods and living standards of displaced persons. • Improve the living conditions of physically displaced persons through the provision of adequate housing with security of tenure in resettlement sites. As of March 2022, full inventories for all economically displaced people impacted by the project footprint at the time have been prepared. The initial focus was on those affected by the closure of land within the perimeter fence and water harvest dam. These inventories were audited by a third party and endorsed by the local authorities Dabakala's Prefect and Sub-prefect. Initial compensation payments for all affected people were paid between November 2021 and February 2022. Some recipients have elected to receive their payments in instalments so these will continue until 2023. In December 2021, additional impacts were identified associated with the extension of the fence around the TSF and waste dumps to the south, as well as the Lafigué bypass on the access road.	Approved
Land disturbance	This procedure will aim at establishing a process for the management of land disturbance in order to identify and manage any impacts in line with corporate and site requirements and plans, and regulatory requirements. It will focus on the following situations: • Ground disturbance: any activity that will break the ground or surface rock layer including, but not limited to, tracked vehicle movements (bulldozer, excavator), post hole digging, excavation works, stockpiling activity (e.g. of soil or vegetation), grading, ripping, ploughing, dredging, blasting, drilling, placing of fill etc. Includes movement of soil from one location to another location. Excludes rubber tire vehicle movements. • Vegetation disturbance: pruning, thinning, trimming, clearing, burning etc. of vegetation. • Work in the beds or banks of a watercourse: a watercourse bed or bank refers to the area within the outer limits of the defined channel of the watercourse (i.e. following the highest points of land in the channel that are covered by the watercourse water, whether permanently or intermittently).	Approved

Table 20-23: SML Environmental and Social Management Plans

Management Plan	Description	Status
Cultural Heritage and Chance Finds	The purpose of this procedure is to ensure the protection of sacred sites and cultural heritage including potential archaeological finds in the areas of our operations and to define the procedures and processes that will be adopted in case of impact of the project and operations on them. During the exploration, construction of project infrastructure or mining phases of the gold deposit that involve excavation, discoveries may be made. These may include the following: • Archaeological heritage whose existence was unknown or unrecognized. • Graves containing human remains that were not mentioned in the preliminary studies (ESIA, RAP). • Sacred sites whose existence was not mentioned in the preliminary studies. • The requirements of this Procedure apply to cultural heritage whether or not it has been legally protected or previously disturbed. An integrated approach to cultural heritage management involves a wide range of stakeholders, including Endeavour, contractors, local and administrative authorities, regulators, museums and the general public.	Approved

20.8 Governance

20.8.1 Sustainability Reporting

Endeavour publishes an annual Sustainability Report, available on its website, which is aligned to several Environmental, Social, and Governance (ESG) frameworks, including;

- the Global Reporting Initiative (GRI);
- the World Gold Council's Responsible Gold Mining Principles (RGMPs);
- the Task Force for Climate-related Financial Disclosure (TCFD);
- the Sustainability Accounting Standards Board (SASB); and
- the Local Procurement Reporting Mechanism (LPRM).

The Company is also a participant of the UN Global Compact.

The Sustainability Report includes Endeavour's ESG strategy, its annual goals and progress in meeting these goals, as well as its ESG performance across a range of key indicators.

Since 2016, the company has disclosed details of its tax, royalty, dividend, and other payments to governments as required under Extractive Sector Transparency Measures Act ('ESTMA'). It also reports in line with the Extractive Industries Transparency Initiative ('EITI').[134] The ESTMA reports include information on certain tax, royalty, dividend, and other payments to governments on a country-by-country, as well as on a project-level basis, and are published annually on Endeavour's website.

[134] A requirement of the 2014 CI Mining Code.

Further, Endeavour also discloses its ESG management, processes, and performance, across a range of Rating Agencies, including MSCI[135] (AA rated in 2022), Sustainalytics[136] (Medium Risk rated in 2022, ranked 10th/177 precious metals companies) and CDP[137] (rated C for both climate change and water in 2021, which places Endeavour in the top 60% of respondents for climate change work).

20.8.2 Management Systems

SML has developed an Environmental and Social Management Plan (ESMP) to address the requirements of all Environmental and Social Impact Assessments. The ESMP is prepared in accordance with leading practises (ISO 14001:2015), but also all national regulatory requirements. The ESMP and its related documents apply to all aspects of Project development, operations, and closure. It is noted that the existing ESMP will need to be updated based on the outcomes of the recommended forward work programme included in Section 20.12.

The purpose of this strategic document is for the facilitation of environmental management measures to minimise the environmental risks associated with the Project and the operation, in order to protect the environment and have a positive sustainable impact on the community.

More specifically the ESMP:

- details environmental management operating practises to meet environmental objectives and outcomes;
- provides a framework for effective implementation of environmental and social management objectives;
- defines roles and responsibilities for environmental and social management issues and compliance; and
- ensures ongoing review of site-specific environmental and social management and mitigation measures to affect continuous improvement in environmental and community management.

The ESMP is reviewed every year or as required (if more frequent), e.g., if there were significant changes to activities and management plans.

20.8.3 Environmental and Social Monitoring

Environmental and social (E&S) monitoring activities will be conducted on a monthly basis, by following the relevant Standard Operating Procedures (SOPs). Monitoring results are compared against trigger values to determine if any contingency measures are necessary. The programmes are conducted by SML's Environmental and Social Performance Departments, which are responsible for ensuring compliance with the commitments agreed to as part of the various ESIA and environmental and social obligations.

SML's environmental inspection and monitoring programmes/plans also apply to all contractors/sub-contractors employed by SML. As such, the requirements of the programmes including associated reporting requirements, are built into all commercial contracts.

The key environmental and social aspects which form the basis for SML's E&S monitoring programmes are listed in Table 20-24, following:

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135 www.msci.com

136 www.sustainalytics.com

137 www.cdp.net/en

Table 20-24: Mine Environmental and Social Monitoring Programmes

Environmental	Occupational Health and Social
• Surface water • Groundwater • Air quality and GHG • Ambient noise • Biodiversity • Soil • Rehabilitation	• Worker Health • Noise • Socio-economic • Gender equality • Local hiring and procurement • Social cohesion

20.8.4 Carbon Emissions Reporting

Energy is a critical input for mining operations. It is also a significant business cost and a major source of Endeavour's GHG emissions. Working to improve the efficiency of its operations, reducing energy use, and lowering emissions are key drivers for the long-term sustainability of Endeavour's business.

Endeavour recognises its responsibility to contribute to the realisation of the Paris Agreement. In 2021, the company announced its climate change targets: Net Zero by 2050 and a 30% reduction in emissions intensity by 2030. Climate-related targets have also been included in Endeavour's long-term employee incentive plans. In 2022, the company announced a short-term target 670 $kgCO_2$ e/oz of gold produced, which is tied to the Group's annual 2022 employee bonus scheme.

The assessment and management of climate-related matters is embedded across Endeavour's operations and delegated authority flows down from the Board. At a management-level the ESG Steering Committee, which includes Endeavour's CEO and COO, provides internal oversight of strategy and progress, including climate-related aspects and targets.

In Q4 2021, Endeavour appointed a dedicated senior executive, reporting directly to the COO, who is responsible for managing and coordinating the Group's decarbonisation pathway, energy transition projects and hydrocarbon management policy across the Group.

During 2021, Endeavour undertook a GHG emissions abatement study across all its operations. The purpose of the study was to investigate opportunities to reduce emissions. It considered measures such as fleet improvements and/or fleet replacement (more fuel-efficient engines), increasing process efficiencies, and using cleaner fuels and renewable energy.

Thus far, switching to renewable power was identified as having the most potential. Solar power has been already identified as a key option, at Endeavour's Houndé and Sabodala-Massawa assets, and Endeavour expects it to be a core part of the Group's energy mix going forward. The detailed emissions abatement study is expected to be completed in 2022, which will feed into the outcomes of the scenario analyses; and ultimately, form part of Endeavour's decarbonisation strategy towards reaching its climate change targets.

In 2021, the company also undertook an assessment of climate risk management in accordance with the Task Force on Climate-Related Financial Disclosures (TCFD) framework, to ensure that risks are appropriately identified, managed, and monitored. The materiality of risks, as well as opportunities, are evaluated based on their financial or operational impact over the short, medium, and long-term using both qualitative and quantitative judgements. Ahead of a climate change scenario analysis, which is expected in late 2022, Endeavour has a high-level screening of climate risks, and potential impacts of climate change under a business-as-usual scenario and low carbon transition scenario.

20.8.5 Tailings Storage Facility (TSF) Audits

In line with Endeavour's 'zero harm' philosophy, the management of tailings is a critical thematic area within Endeavour's corporate risk management and reporting system. As such, there is a strong, structured, and robust approach, to the risk classification of existing and planned TSFs.

Endeavour and by association SML, will evaluate the consequence to human and environmental health in line with the classification systems of the Australian National Committee on Large Dams ('ANCOLD'); the Canadian Dam Association ('CDA') and the 'Global Industry Standard on Tailings Management' (GISTM). Accordingly, Endeavour conducts regular internal and external audits to monitor, measure, and evaluate the effectiveness and safety of the TSFs, across all its operations. The results of these audits are reported back to site, senior management, and the Board on a regular basis.

On the Endeavour website, as part of the 'Investor Mining and Tailings Safety Initiative', Endeavour publishes pertinent information on its TSFs annually. In 2021, Endeavour employed independent external reviewers to evaluate all of its tailing facilities, and no serious issues were identified.

For the tailings storage facility at SML, it may be noted that whilst ANCOLD standards were adopted for the DFS, the TSF will be designed in accordance with GISTM standards. This change is unlikely to result in any material change to the TSF design and/or costs.

20.8.6 Cyanide Management

Endeavour's, and by association SML's, approach to cyanide management is aligned to the International Cyanide Management Code ('ICMC'). The metallurgical plant will be built with International Cyanide Management Code (ICMC) compliance in mind. As with the other Endeavour sites, it will be audited both internally and externally (by an ICMC accredited auditor) against the ICMC code. Endeavour to date has not signed up for ICMC accreditation.

Endeavour/SML will provide training to employees and contractors on safe cyanide handling and management. On-site emergency response teams will receive special training to manage incidents involving cyanide.

20.8.7 Stakeholder Management

Endeavour's/SML's stakeholder management plans are in line with the World Gold Council's Responsible Gold Mining Principles (RGMPs).

The principles that guide stakeholders engagement and management are outlined in Section 20.6.

20.8.8 Health and Safety

Endeavour's Health, Safety and Environmental (HSE) Polices, Management System and Standards, align with international best practise and are audited internally and externally on an annual basis against the ISO 45001 standards. Polices and standards are managed at a group level and rolled out to Endeavour's various business units, including SML.

Relevant policies, procedures/standards and reporting requirements are incorporated into contracts between the 'Employer' and the contractors/subcontractors and as such, HSE is seen holistically across the group and in the areas impacted by Endeavour's operations.

Group wide and site specific; safety indicators are a metric in annual employee compensation and there are regular communications via toolbox meetings, safety briefings and visual communications between all stakeholders.

20.8.9 Local Procurement and Employment

Employment opportunities and sustainable development initiatives supported by SML through the Corporate Social Responsibility (CSR) programme will provide significant support for broad-based socio-economic growth in the area.

SMLs procurement and supply chain multiplies the positive impact on the local, regional, and national economy, strengthening local businesses and creating indirect and induced employment. SML will prioritise national and local suppliers of goods and services as well as the development of in-country manufacturing and supply chains.

Endeavour's/SML's supplier database uses the IFC ownership categories, and priority is given to local and national suppliers from this list, provided that they are competitive and meet standards.

Endeavour/SML has a 'Supplier Code of Conduct' (SCC) policy that sets out requirements for the supply of goods and services, which alongside the vendor due diligence programme, enables the Company to support national and local businesses, whilst maintaining standards and ensuring issues such as child labour or forced labour and bribery and corruption do not occur within the supply chain.

To assist with building strong community relations, as part of an onboarding process of new local suppliers, the supply chain team organises an introductory meeting with all stakeholders to sensitise them to the local operating environment.

Each year, the supply team meet with their key contractors and suppliers regularly to review performance and at the group level, senior management meets with strategic suppliers to discuss performance and broader topics, such as ESG efforts.

20.8.10 Closure Review and Commitments

As mine activities progress, the Mine Closure and Rehabilitation Plan (MCRP) is regularly reviewed and updated to reflect any changes that have occurred. Where there is a substantial or material change to the closure strategy, SML will prepare a revision. However, as a minimum, the MCRP will be updated every three years.

Closure payments and commitments for SML in relationship to PE 58 are discussed in Section 4 of this Report.

20.9 Data Verification

The approach to the verification and use of data/information presented in Section 20 is discussed in Section 12.

20.10 Comments on Section 20

The information provided for the Property (PE 58) and the surrounding potentially impacted natural and human environment is of sufficient detail to ensure that major issues have been dealt with and major risks have been identified and mitigation measures incorporated into the project design and management plans. The communication methods adopted by the mine must be of such a quality, that issues are dealt with as they arise and such that, issues not previously noted are dealt with. The management plans for social and environmental impacts must also be of such a quality that any adverse trends can be picked up and remedial measures applied where necessary.

The large number of artisanal miners who were active at the site have been moved off and this is not uncommon at gold mining sites in West Africa. From the information reviewed it appears that this was done peacefully in the vast majority of cases. It will mean that the impacts of ASM on the operation will need to be continually managed as the pressures of having people working around a formal mine will always be there, and social programmes will need to be put in place to ensure that alternative livelihoods are created and conflicts are managed when they do occur. The implementation of an audit to see where the miners who were on Site have moved to, and to record and deal with any grievances which may exist, will help prevent or reduce areas of conflict if they do exist.

The public consultation was conducted in line with CI standards and legislation. The process followed is thus different to a normal IFC/World Bank approach of gathering issues after the initial project plan is presented, then compiling the assessment and management plans and asking for comments and feedback from a range of stakeholders on the management measures proposed. The process conducted did get feedback and was officiated on by various government departments and committees. DWE have not seen anything to suggest that a major issue which was raised, or could have been raised, has not been dealt with.

20.11 Interpretations and Conclusions

Interpretations, conclusions and risks pertaining to Section 20 are discussed in Section 25 of this Report.

20.12 Recommendations

Recommendations/forward work programme activities to be carried out; Prior to the FEED Phase (PFP); FEED Phase (FP); and in the Operational Phase (OP) are documented in Section 26 of this Report.

20.13 References

References cited in the preparation of Section 20, are presented in Section 27 of this Report.

21. CAPITAL AND OPERATING COSTS

21.1 Study Contributors

The capital cost estimate for the Lafigue Project has been compiled by Lycopodium based on inputs from:

- KP for quantities on the tailings storage facility, water and drainage infrastructure, site access roads, and airstrip. Pricing with rates/unit costs was developed by Lycopodium.
- Endeavour, supported by SRK and ECG, provided area specific estimates for; mine establishment and facilities, infrastructure facilities, high voltage power supply and Owner's costs.
- LoM Sustaining capital costs were developed by Endeavour and KP, and whilst reported herein, they are applied in Section 22 (Economic Analysis).

Operating cost estimates have been compiled by Lycopodium based on costs developed by:

- SRK/Endeavour - Mining contractor and mine management costs.
- Lycopodium - Processing costs.
- Endeavour/Lycopodium - Site General and Administration (G&A) costs.
- Endeavour - Labour organisational charts, project manning, labour rates and manning build-up.

21.2 Basis of Estimate

21.2.1 Base Date & Accuracy

The estimate is expressed in USD based on prices and market conditions current as of the second quarter of 2022 (2Q22).

The capital cost estimate has been developed in accordance with Lycopodium's capital cost estimating procedures and has an associated accuracy provision of (-5 to +15)%.

The operating cost estimate is based on the LoM plan, feasibility study testwork, plant and mine designs, labour schedules and costs; General and Administration (G&A) costs, and base date prices and market conditions, and is considered to have an accuracy provision of ±15%.

21.2.2 Foreign Exchange Rates

The foreign exchange (FOREX) rates adopted for the estimates and the foreign currency capital exposure is shown in Table 21-1.

Table 21-1: Foreign Currency Exposure (Q2 2022)

Currency	Exchange Rates	USD Portion (M)	Percentage of Capital Estimate
USD	1.00	404.61	85.5
AUD	0.77	46.43	11.2
EUR	1.15	8.55	2.0
GBP	1.43	2.19	0.5
ZAR	0.07	2.43	0.6
CAD	0.81	1.02	0.2

21.2.3 Basis for the Capitalisation of Pre-production Costs

Project costs relating to exploration and feasibility studies have been considered as sunk costs (Section 21.2.6.1) and these were cut off at December 2021. Board approval for the Project set this as the capital expenditure (CAPEX) commencement date (January 2022) with project implementation and pre-production costs being incurred up to the date of ore through the mill for the first gold pour (Q2 2024).

The Sunk cost period was subsequently extended from January 2022 to 1 June 2022, the 'Effective Date' of this Report.

The progressive ramp up of owner's team manning costs and expenditures during the project implementation phase is shown in Figure 21-1. This period is followed by operations pre-production labour build-up and mining pre-strip, transitioning to commercial production operating costs (OPEX).

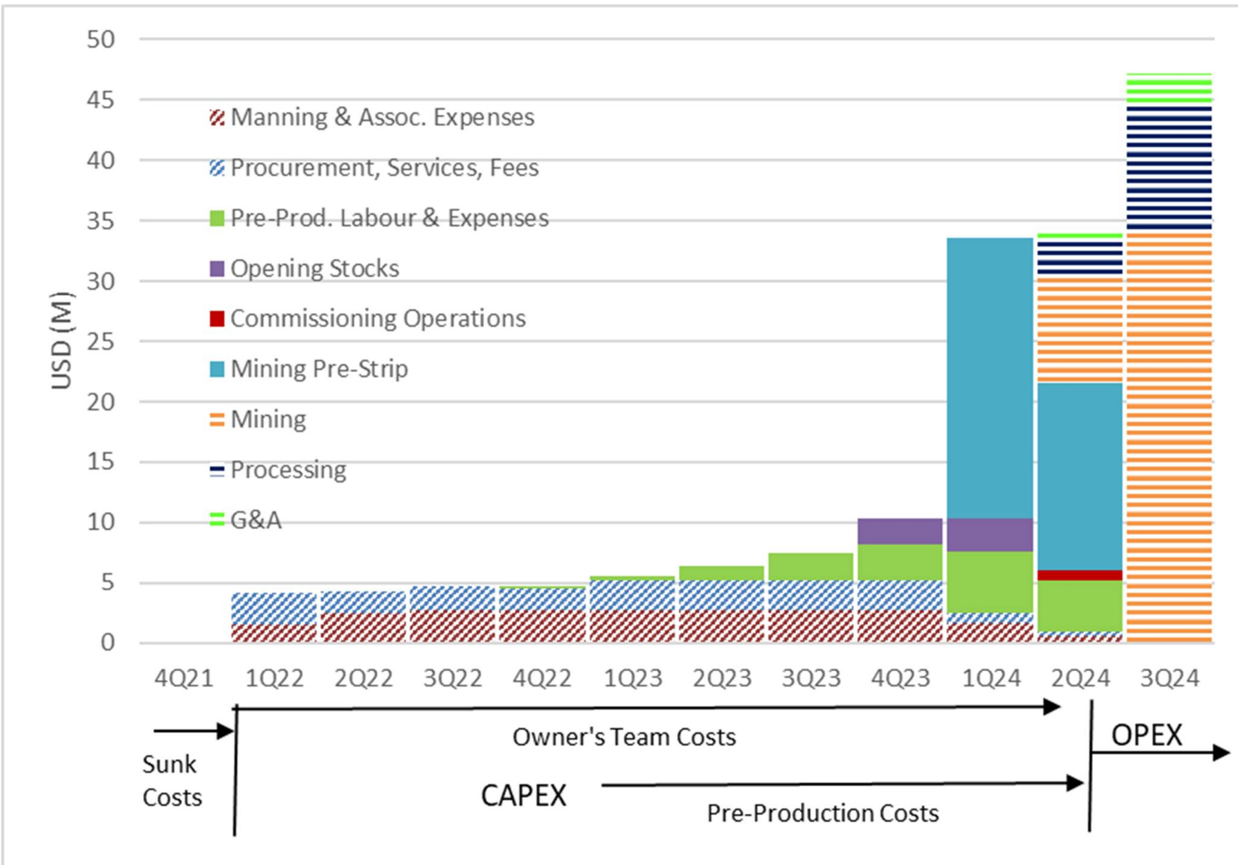

Figure 21-1: Lafigué Owner's Team, Pre-production and Operating Cost Ramp-up

21.2.4 Duties and Taxes

Based on guidance from Endeavour, the following government duties and taxes have been applied to components of the capital and operating cost estimate. Further detail is provided in Section 4 (Endeavour, 2022):

- For construction, a 2.5% regional levy has been applied to goods imported from outside of the Economic Community of West African States (ECOWAS).
- No VAT is payable over the construction period (not considered in the financial model)

- A 1.6%[138] banking fee (central and commercial banks) has been applied to items not purchased in EUR or XOF.
- The applicable company withholding tax (WHT) for Project foreign service providers not registered in a Double Tax Treaty country have been applied.
- With the exception of chemical products which are subject only to the regional levy of 2.5%, all other materials and consumables used in operations are subject to full duties (the basis for the DFS).

Whilst contrary to the tax and duty basis outlined in Section 4, the mining contractor has assumed that;

- they like the permit holder will be VAT exempt during construction; and,
- after the three-year temporary admission period has ended on the mobile equipment imported, they will pay the full duty on the amortised amount, not the CIF value held by the Customs Authority.

21.2.5 Owner's Costs

As part of Project development, Endeavour will be required to provide the Owner's project management and operations team.

The Owner's construction team will interact closely with operations management personnel recruited during the construction phase of the Project.

In addition to the above, the following allowances have been made in the capital cost estimate:

- Owner's project expenses (provided by Endeavour).
- Pre-production costs.
- First fills (lubricants, fuel, and reagents).
- Opening stocks.
- Plant mobile equipment.
- Project spares (operational and strategic spares).
- Vendor representative and training costs for the process plant.

21.2.6 Cost Definitions

21.2.6.1 Sunk Costs

Costs relating to geological exploration, resource definition, tenement acquisition, permitting and feasibility studies prior to project approval are considered sunk costs. Further, the basis for either the exclusion or inclusions of operating/capital costs incurred prior to the 'Effective Date' of this Report (1 June 2022), is as described below.

- Historical exploration work and study/Project development and permit costs (i.e., PFS and DFS and supporting studies) are considered 'Sunk' and excluded from the CAPEX estimate.
- Payments (USD 5.2 M) for land compensation made prior to 31 December 2021, are excluded from the CAPEX estimate presented.

[138] Estimate only

- Early front-end engineering design (FEED) and Site development activities/costs (USD 6.2 M) incurred from 31 December 2021 to 1 June 2022 whilst included in the CAPEX estimate (Table 21-19), are excluded from the CAPEX estimate in the financial model[139]. Early works activities include, but are not limited to; fencing, airstrip, roads and water dams etc.

21.2.6.2 Project Development Costs (Capital Costs)

Project development costs relate to the deployment of capital to finance the design and construction of the mine, processing facilities and associated support infrastructure to achieve the Project objectives. These functions are typically contracted out to companies with expertise in the various project areas.

21.2.6.3 Pre-Production Costs

Pre-production costs are all on-going costs from the point at which costs are considered sunk, through to ore entering the mill for the first gold pour. These include for example: early mining/pre-strip, operations manning build-up and training, reagents, first fill and opening stocks and costs for equipment vendor representatives to attend site to sign off on the equipment installation and provide commissioning assistance as required. Further detail is provided in Section 21.2.3.

21.2.6.4 Start-up Working Capital

Project, not operational working capital, is the capital typically required to fund initial operations until the project becomes self-sustaining through revenue generation.

21.2.6.5 Sustaining Capital

Sustaining capital is the ongoing capital expenditure post start-up of operations required to facilitate ongoing operations, for example, the costs for progressive tailings dam lifts, replacement of mobile equipment and plant items, outside of normal maintenance. Whilst Section 21 includes the basis of the sustaining cost calculations, the actual quantum by year, is used and presented in the financial model (Section 22).

21.2.6.6 Closure Costs and Annual Bond Payments

Based on Article 144 of the 2014 Mining Code and the Decree, a rehabilitation bond is to be provided by each operating entity. The Closure cost estimate is detailed in Section 20, whilst the basis of the annual bond payments is detailed in Sections 4 and 21.

21.2.6.7 Direct Costs

Direct Capital Costs

Direct capital costs are generally quantity based and include all of the permanent equipment materials and labour associated with the physical construction of the mining and processing facility, including but not limited to: general infrastructure; buildings; fixed and mobile equipment; and contractor's indirect costs (P&G's) such as: mobilisation; support, and non-productive labour items (mobile equipment, craneage, logistics scaffolding and PPE).

[139] See Table 21-21 for costs considered sunk, and removed from the CAPEX estimate in the financial model

Direct Operating Costs

Direct operating costs are those costs directly related to the on-site production of doré gold bars, and include; power and fuel usage, operating consumables, direct labour, maintenance, insurance, etc.

21.2.6.8 Indirect Costs

Indirect Capital Costs

Indirect capital costs are those costs incurred in the development of the mine that are not directly assignable to 'Direct Costs'. Indirect costs include, but are not limited to: EPCM costs; community; testing (non-destructive testing and geotechnical); health & safety; temporary construction facilities; waste services, commissioning costs, pre-production operating costs, working capital (insurance spares; cash flow delays; stores and stocks); owner's team costs; service level agreements (SLAs); travel, accommodation, recruitment and relocation; construction insurances and bonds and guarantees (engineering, construction and warranty periods).

Indirect Operating Costs

These are typically off-site charges to the project, for example head office charges, government taxes and royalties (apart from direct import duties on consumable reagents), gold refining and bullion transport costs and capitalised costs.

21.2.6.9 Fixed Operating Costs

Fixed costs are the operating costs that do not vary in the short-term and, do not change with the mining or processing tonnage rates, for example, labour and administration costs.

21.2.6.10 Variable Operating Costs

Variable costs are the operating costs that vary with the mining or processing tonnage rates, and include; the bulk of the fuel and power, and reagent and consumable costs.

21.2.6.11 Contractors Distributables/Preliminary & Generals

Contractor distributables/Preliminary and General (P&G) costs are defined as those expenses which are incurred before work in producing the project deliverable, together with those costs that are non-specific to a particular Bill or Activity list item. P&G's will typically include: mobilisation costs; insurance cost; finance charges; site establishment; site disestablishment; project management; site management; planning; material control; ancillary equipment (cranes) site offices and services (secretarial, data processing, QA, QC, contract admin, planning and survey). Valuation of Preliminary and General is based on fixed, time and value related cost elements which are, directly dependent on the level of Direct Field Labour (DFL) effort required for a given scope of work and time duration.

21.2.7 Key Operating Cost Inputs and Basis

21.2.7.1 Fuel Basis

A diesel price, delivered to site, of USD 0.91/L for production and USD 1.00/L for construction has been used as advised by Endeavour based on forward looking market pricing analysis (USD 73/bbl). The majority of the diesel will be utilised for operating the contract mining fleet and will be free issued to the mining contractor.

Diesel will be supplied by a yet to be appointed third party service provider, who will dispense fuel on Site on a consignment basis.

Diesel will be used in the processing plant for carbon elution heating, plant mobile equipment and miscellaneous minor uses such as emergency power generation, waste incineration, etc. Diesel will also be used for mobile equipment servicing Endeavour administrative and mining management staff.

All non-mining contractor diesel consumption is accounted for as a processing consumable in the cost estimate presentation.

21.2.7.2 Power Basis

Electricity to power the Project will be sourced from the Côte d'Ivoire (CI) national grid service provider CI Energies. ECG has been contracted to design and manage the transmission line and associated high voltage infrastructure to supply the site power. Based on negotiations with CI-Energies, ECG advises that the power cost estimate should be based on an average unit rate of USD 0.112/kWh.

Estimated power costs are presented under the respective process areas, given the majority of the power being consumed is by processing and processing support services.

21.2.7.3 Labour Basis

Labour Rates

Labour rates were provided by Endeavour for three employment categories; Senior Management (Cadre(Fr.), Supervisors and Administrators (Agent de Maitrise (Fr.)) and tradesmen/workers (Ouvrier (Fr,)). Expatriate employees are classified separately as they are not subject to the same local statutory provisions. The base annual salary and overhead make up costs for each category within the broader classes was provided, along with the associated work roster. The labour rates presented include overhead allowances.

Organisational charts with employee roles and manning levels were provided by Endeavour and were used to determine the labour operating cost estimate. Structures and position categories were based on other Endeavour operations and benchmarking within CI.

National Staff

National staff overhead allowances include provisions for overtime pay, bonuses earned, redundancy funding, life and health insurance, pension fund contributions as well as income and social taxes. The proposed roster system is as defined in Section 20.

Senior management and supervisory level personnel will be housed in the camp, but workers will find accommodation in local villages, with a bus service providing transport to site. All non-camp resident staff will be provided with a daily meal when rostered on.

Expatriate Staff

The expatriate overhead costs include allowances for visas and working permits, income tax, rest and relaxation (R&R) flights and transit accommodation costs and health and life insurance. Provision for leave cover is also allowed.

Expatriate personnel will be housed in the camp when on site. Catering and accommodation costs have been provided for in the G&A costs.

Labour numbers and costs are detailed in the estimate presentation.

21.2.7.4 General and Administration (G&A) Costs

The General and Administration (G&A) costs prepared for the Project/Mine, are based on operational data from another in-country Endeavour mine, adjusted for planned differences in operating and administration approaches.

Selected G&A costs are attributable to specialist service providers (e.g., site security, camp catering and cleaning, medical services and staff transport). Proposed SLAs and associated costs were advised by Endeavour

Further detail is provided in Section 21.3.2.4.

21.2.7.5 Reagents and Consumables

The usage rates of reagents and other consumables have been calculated from laboratory testwork and comminution circuit modelling based on average ore properties for the three weathered ore states (fresh, transition and oxide). Where these rates could not be sourced from testwork, consumption was calculated from first principles, or assumed based on experience with other operations. No additional allowances for process upset conditions and wastage of reagents have been made.

Consumable supply costs were sourced from purchase contract pricing advised by Endeavour, budget quotations from reputable suppliers, or in house data relating to similar projects in the region. Transport and freight costs to site along with import duties and direct taxes have been added.

Costs for the processing operating consumables along with expected consumption rates for key consumables are summarised in Table 21-11.

21.2.7.6 Mobile Equipment

Plant and general site mobile equipment requirements were agreed with Endeavour. Mobile equipment costs provide for the fuel and maintenance of the mobile equipment fleet (excluding the mining fleet and mining contractor light vehicles). The purchase cost of this mobile equipment has been included in the capital cost estimate with ongoing replacement costs in sustaining capital.

The fuel and maintenance costs for the mobile equipment are included in the consumables and maintenance cost centres respectively and allocated to the processing cost centre as the dominant non-mining user.

21.2.7.7 Water Supply and Services

Water supply costs are based on operation (power usage) and maintenance of the water harvest dam (WHD) supply pumps and pipeline and the water storage dam (WSD) pumps and pipeline.

Water supply costs have not been presented separately, as they have been included in the other cost centres:

- Water supply pumping power is included in the power cost.
- Maintenance costs associated with water supply are included in the maintenance cost.
- No additional labour is required for the WHD and WSD pumping stations, however plant personnel will routinely check the pumps and pump lines (labour included in the plant labour cost) and security personnel will check the remote WHD area (labour included in the G&A cost).

Raw, fire and potable water will be supplied to the MSA area from the plant.

The consumable cost estimate includes allowances for the treatment of filtered and potable water and for the addition of antiscalant to both the decant return and the elution water.

21.2.7.8 Service Level Agreements

A number of operations functions will be contracted to third-party providers, and these will be significant drivers of the Project's operating costs. Key SLAs are:

- The mining contract:

  Expected cost details and estimate build-up from the mining cost estimate, with summary level costs presented in this section.

- The diesel fuel supply service agreement with a CI Fuel supplier.

  Fuel costs are essentially all included in the mining costs, with only minor fuel usage outside the mining fleet requirements.

- The grid power supply agreement with CI Energies:

  The processing plant will be the major site power user. Power draws and costs are detailed in Table 21-9, whilst annual power consumption by area is presented in Table 21-7.

- The site assay laboratory operations will be contracted out to take advantage of the third-party quality control and expertise in this critical production accounting function.

Additional service contracts falling under G&A costs are:

- Site security.
- Worker transport to site.
- Camp management, catering, and cleaning.
- Medical services.

A further non-specific provision for external consultants/area specialists has been made in the G&A costs.

Process plant maintenance costs allow for contracting out specific functions such as the ball mill relining and assistance with HPGR roll changes. A non-specific allowance for contractor assistance during planned maintenance shutdowns is also included in the plant maintenance costs.

21.2.8 Operating Cost Estimating Methodology

21.2.8.1 Battery Limits, Inclusions & Exclusions

Battery Limits

The battery limits for the processing operating costs are as follows:

- Ore haul truck direct tipping/loader feed to the plant RoM bin (excludes operations on the RoM pad, rehandle and reclaim of the plant stockpile material). Note however that the RoM rehandling costs estimated by Endeavour were provided to Lycopodium for inclusion as a separate processing cost component to align with Endeavour's financial accounting classifications.
- Gold bullion in plant goldroom safe.

- All tailings management operating costs are included (excludes future lifts which are covered by sustaining capital).
- All raw water supply management is included from the water harvest and storage dams into the raw water storage tank.
- Receipt of bore water to the plant bore water storage tank.
- Power supply from the HV switchyard (maintenance and operation of the emergency power generation is in the processing scope).
- Offloading of diesel fuel to the plant day tank and emergency power generator fuel storage.
- Delivery of reagents, grinding media and other consumables to the point of offloading inside the plant fence.

The basis of the OPEX estimate is as further defined in the exclusions and inclusions following.

Inclusions

The process plant operating cost estimate presented in this section includes:

- Duties, tariffs and regional levies for goods imported from outside the ECOWAS Region.
- Central and commercial bank payments on FOREX transactions that are not in EUR or XOF (1.6% of FOREX value).
- Company withholding taxes where applicable.
- Other:
  - Employer payroll contributions, including payroll tax and social security contributions (retirement, family and worker compensation) are reported in the labour costs.
  - First fill and opening stock costs (included in the capital cost estimate).
  - Costs for mining services area (MSA) power; and raw, fire and potable water supply (in the power and consumables costs).
  - Costs for the preparation and assaying of 200 mine grade control samples per day in the plant laboratory in addition to the normal process and environmental samples (in the laboratory cost).

Exclusions

The process plant operating cost estimate presented in this section excludes:

- All mining operating costs associated with the mining contractor, including:
  - Ore feed to the RoM bin and RoM stockpile management.
  - Haul road construction and maintenance.
  - ANFO supply and preparation.
  - Pit dewatering.
- Tailings storage facility maintenance and lifts, rehabilitation and/or closure costs (included in the KP and deferred/sustaining capital cost estimate).
- Water harvest dam and water storage dam maintenance (included in sustaining capital).
- Overall site water management (included in the KP scope).

- Any impact of foreign exchange rate fluctuations.
- Any escalation from the date of the estimate.
- Any contingency allowance.
- Any head office charges or project funding/financing costs.

The following cost elements are included in the financial analysis and are thus excluded from the operating costs presented herein.

- Corporate income tax.
- Ad valorem taxes based on gross revenue, with deductions added to the cost of gold sales. These taxes include:
  - Royalties.
  - Community levies.
- Cost of gold sales, including; transport, vaulting, refining and payability charges and insurance.
- Rehabilitation/closure bonds.
- Other exclusions include, but are not necessarily limited to:
  - Any form of carbon taxes (not currently applicable).
  - Value added taxes (VAT).
  - JV fees.

21.2.8.2 Mining Cost Basis

The mining schedule and contractor operating cost estimate have been used as the basis for the annualised production and cost model. Annual waste and ore tonnages, strip ratio and head grade, are based on the Scenario 13 mining schedule (Section 16).

The Project/Mine will make use of conventional open pit truck and excavator operation with the production unit operations (drilling, blasting, loading, hauling, and dumping) carried out by contractor mining personnel and equipment.

Mining operating costs exclude contractor mobilisation and site establishment costs, as these are considered capital costs. Pre-strip costs are included in the operating costs, but are capitalised prior to the first gold being shipped, on the basis that as per the 2014 CI Mining Code, this date is considered as the commencement of 'Production'.

The mining contractor will be responsible for delivery of ore to the plant RoM pad and maintaining sufficient ore stockpiles to ensure 4.0 Mt/a (db) crusher feed equivalent. The contractor will also be responsible for building waste stockpiles.

Endeavour has estimated the owner mining management labour required to cover geology, mine planning, grade control sampling, emulsions and explosives, and contractor management functions.

21.2.8.3 Processing Cost Basis

Processing operating costs have been developed for fresh and blended (fresh/oxide/transition) ores. The average life of mine (LoM) blend based on reserve tonnes, would be 94% fresh and 6% oxide/transition material. The LoM processing costs are a weighted average cost based on the LoM blend.

Based on the process feed schedule derived from the available mined ore, it is expected that the plant will initially operate on up to 50% oxide/transition ore (maximum 20% oxide in the feed) blended with the fresh feed, but after Q3 Year 3, the feed will be 100% fresh ore (unless additional oxide resources are discovered in the vicinity).

Processing operating costs have been determined for a plant with an annual throughput of 4.0 Mt/a (db) plant feed ore at a P_{80} grind size of 106 µm, based on operating 24 hours per day, 365 days per year. The milling and downstream plant design operating availability is 91.3%.

21.2.9 Capital Cost Estimating Methodology

21.2.9.1 Assumptions and Exclusions

The capital estimate is qualified by the following assumptions:

- Contingency has been allowed based on the quality of the information and level of technical development, however no allowance for escalation has been included.
- Prices of materials and equipment with an imported content have been converted to USD at the rates of exchange stated in Table 21-1. All pricing received has been entered into the estimate using the supplier's native currency.
- Contractor rates and distributables/P&Gs include; mobilisation/demobilisation, recurring costs, direct and indirect labour, construction equipment, construction crane (up to 100 t), materials, materials handling and offloading, temporary storage, construction facilities, off site costs, insurances, flights, construction fuel, tools, consumables, meals and PPE.
- Potable water and raw water supply will be provided by the Owner and available at site for the use by contractors. The costs for establishing the potable water and raw water supply facilities are included in the capital cost estimate, whilst the contractor's costs for transport and handling from the water supply battery limits are in the contractor P&Gs.
- Site construction offices will be containerised units only, with the intention that the permanent administration building construction schedule will be accelerated for early use by the EPCM and Owner's team. Early establishment of permanent site buildings is achievable, given Endeavour's plan to purchase RA International prefabricated flatpack buildings that are currently in storage and do not require fabrication.
- The bulk commodities for earthworks that include imported materials, assume that suitable construction/fill materials will generally be available from borrow pits within 2 km of the work fronts, other than roads which will likely have longer haulage distances. Imported materials for concrete, have been included in the concrete installation rates by the contractor.
- Engineering quantities for the; tailings storage facility (TSF), WHD, WSD, haul roads, site access roads, surface water management and sediment control structures were provided by KP, and appropriate rates were applied to complete the capital cost these items. KP subconsultant costs for design and construction supervision are included.
- The estimate allows for aggregate and sand for concrete batching to be provided by the concrete contractor and locally imported from an existing commercial quarry in the Bouake region ((80 to 90) km from the Site).
- The estimate allows for all reinforcing steel bars and mesh for construction to be provided by the concrete contractor. Free issue of materials would be a project capital savings opportunity, particularly given savings on duties.

- There is no allowance for blasting in the bulk earthworks, which is consistent with preliminary geotechnical information.
- The estimate allows for the supply of structural steel and platework from Southeast Asia.
- Meals and accommodation for the Owner, EPCM teams and senior contractor management have been allowed for in the estimate.
- Domestic flights for the EPCM team are to be provided by the Owner.
- Project spares are a percentage allowance of the mechanical supply cost, based on similar size projects.
- A commissioning assistance crew is included in the estimate as part of the contractor P&Gs.
- PLC programming for the process plant has been allowed for in the EPCM allowance.
- Communications network and data for construction facilities to be free issued by the Owner.
- No allowance for landscaping in the permanent village or at other Site infrastructure features has been included in the capital estimate, however, allowances have been made for civil finishing works including roads, paths, drainage and slope stabilisation (grouted rock pitching).
- The full internal fit out and furnishing of architectural buildings, stores and workshops have been included.
- Owner's mobile equipment to be purchased early and made available for construction and operations for early use by the EPCM and Owner's team.
- The capital estimate includes an allowance for mill installation supervision by the vendor.
- Permits and licences costs up to first gold pour, are included in the capital estimate as provided by Endeavour.

The following items are specifically excluded from the capital cost estimate:

- Working capital (included directly in the financial model if required).
- Exchange rate variations.
- Escalation.
- Sustaining Capital Costs (included directly in the financial model).
- Closure Costs (included directly in the financial model).

21.2.9.2 Overview

The mine, plant, and infrastructure capital cost estimates were prepared in accordance with Lycopodium's standard estimating procedures and practises. The estimate basis and methodology is summarised in Table 21-2 and Table 21-3 following.

General arrangement drawings and a 3D model have been produced with sufficient detail to permit the assessment of the engineering quantities for earthworks, concrete, steelwork, mechanical and electrical for the crushing plant, processing plant, conveying systems and infrastructure.

Unit rates that reflect the current market conditions were established for bulk materials, capital equipment and labour, from region specific budget quotation requests (BQRs). These rates were then benchmarked against projects that are either currently under construction or were recently completed.

Budget pricing for equipment and infrastructure facilities was obtained from suitable suppliers and contractors.

Table 21-2: Capital Cost Estimate Basis

Description	Basis
Site	
Geographical Location	Site Plan
Maps and Surveys	Topo and surveys provided by Endeavour (1 m contour increments based on an amalgamation of 2 m contour data, LiDAR data and site survey data)
Geotechnical Data	Initially assumed as competent ground conditions based on previous projects and informal preliminary advice from Geotech Consultant (KP). Preliminary Geotechnical Report completed at end of DFS was reviewed and is considered to be generally consistent with early DFS assumptions/allowances.
Process Plant Definition	
• Process Selection	Based on Flowsheets
• Design Criteria	Lycopodium DFS Standard (Based on PDC)
• Plant Capacity	4 Mt/a (db) (DFS)
• Flowsheets	Lycopodium DFS Standard
• P&IDs	Not Required – but preliminary P&IDs developed for DFS
• Mass Balances	Lycopodium DFS Standard
• Equipment List	Lycopodium DFS Standard
• Equipment Selection	Lycopodium DFS Standard
• General Arrangement Drawings	Lycopodium DFS Standard
• 3D Model	Semi Detailed
• Piping Drawings	Not Required
• Electrical Drawings	Single Line Diagrams, electrical layouts
• Specifications/Data Sheets	Used for equipment pricing
Mining Preproduction Definition	
• Mining mobilisation, site establishment & pre-strip	Based on mining schedule and tendered mining costs (By Endeavour)
• Haul Roads	Combination of preliminary design & quantities (KP), and tendered mining costs (By Endeavour)
Infrastructure Definition	
• Existing Services	None
• Power Supply	From existing grid (By Endeavour/ECG). Battery limit is connection of outgoing 11 kV feeder.
• Water Supply	Natural runoff collection – KP preliminary design & quantities; Groundwater bores allowances
• Accommodation	341-person permanent camp (including construction camp facilities), 324-person construction camp, and 48 person gendarmes barracks
• Infrastructure Buildings	Based on building list and layout drawings
• Access and site roads	Site access road preliminary design and quantities (KP); and public access road upgrades under construction by Endeavour
• Surface water management and sediment control structures	Preliminary design & quantities (KP)
• TSF	Preliminary design & quantities (KP)
• Mine Services Facilities	Included in mining costs provided by Endeavour
• Security/Fencing	Plant high security fencing, perimeter fence of mine site boundary and local fencing of accommodation areas. Security services and security infrastructure facilities as provided by Endeavour.
• Design Basis	Preliminary
• Layout	Defined

Table 21-3: Capital Cost Estimate Methodology

Description	Methodology/Basis
Bulk Earthworks	Volume for bulk earthworks provided by the preliminary project model.
Detailed Excavation	Allowances for under pad excavation and backfill to prepare site for concrete works.
Concrete Installation	Quantities based on study engineering, reference projects and estimated structures.
Structural Steel	Quantities based on study engineering and reference projects.
Platework & Small Tanks	Platework items as per the mechanical equipment list.
Tankage Field Erect	Tanks as per the mechanical equipment list.
Mechanical Equipment	Items as per the mechanical equipment list. Formal budget enquiries with datasheets and/or specifications for the major mechanical items. Costs for minor items taken from the Lycopodium (recent) database.
Plant Piping General	Factored from mechanical costs.
Overland Piping	Size and specification based on engineering selection. Quantity based on site layout. Rates taken from the Lycopodium database.
Electrical General	Quantities derived from engineering design and site layout. Electrical equipment priced for the project. Bulks and installation costs drawn from a combination of recent database and budget pricing.
Electrical High Voltage Power Distribution	Quantities derived from engineering design and site layout. Electrical equipment priced for the project. Bulks and installation costs drawn from a combination of recent database and budget pricing.
Commodity Rates - General	Based on specific contractor enquiries with indicative drawings.
Installation Rates - General	Based on specific contractor enquiries with indicative drawings.
Large Cranage	Hire of a 250-t crawler crane for major lifts
Freight General	Combination of unit rates for estimated freight tonnes and factoring from supply costs.
Contractor Mobilisation/Demobilisation	Based on Contractor pricing from budget quotation requests (BQRs).
EPCM Fees	Resource based estimate for the EPCM controlled scope.
Owner's Costs	
• Owner's Project Costs	Endeavour estimate.
• Construction Accommodation	EPCM personnel and contractors expats at USD 25/d (Owner's team included in Owner's Costs)
• Project Insurances and Permits	Part of Owner's Project Costs
• Mine, Administration and Plant preproduction expenses	Estimated as part of operating cost estimate for inclusion in the capital cost estimate.
• Opening Stocks, First Fill Reagents and Consumables	Estimated from consumption rates and costs as part of operating cost estimate.
• Spares	% Allowance for commissioning and operational spares. Strategic spares based on budget pricing.
• Duties and Taxes	2.5% regional import levy, 1.6% Banking Fee and withholding tax (WHT) added where applicable.
• Escalation	Excluded.

21.2.9.3 Engineering Status

The level of engineering development varies on a facility-by-facility basis; from recently completed designs, modified construction, and as-built drawings of current and past project facilities, to initial concept drawings.

The key process and engineering design criteria used for equipment selection in the development of the capital cost estimate is described in Section 17. Similarly, refer to Section 18 for a description of site infrastructure and mining facilities.

21.2.9.4 Quantity Development

The Project works were quantified to represent the defined scope of work and to enable the application of rates to determine costs. Allowances for compaction, waste, rolling margin and the like are included in the build-up of unit costs.

Quantity information was derived from a combination of sources and categorised to reflect the maturity of design information as follows:

- Study engineering that includes quantities derived from project specific engineering for the purpose of the study. Includes equipment lists and 3D modelled facilities.
- Reference projects that include quantities drawn from previously constructed projects or detailed designs, and by exception have been adjusted to suit the equipment sizing and layout specific to this Project.
- Estimate that includes quantities derived from sketches or redline mark-ups of previous project drawings and data, compiled by estimating.
- Factored quantities derived from previous estimates or projects.

The derivation of quantities within these categories by percentage is provided in Table 21-4 weighted by bulk quantity.

Table 21-4: Derivation of Quantities

Classification	Quantity	Unit	Study Engineering List/ Model %	Reference Projects (Adjusted) %	Estimated %	Factored %
Earthworks	4 103 761	m³	90	-	10	-
Plant Concrete	13 423	m³	20	70	10	-
Structural Steel	1918	t	3	97	-	-
Platework	767	t	-	90	10	-
Field Erected Tanks	824	t	-	100	-	-
Mechanical Equipment	448	ea.	100	-	-	-
Conveyors	1137	m	100	-	-	-
Piping – Plant	1	lot	-	-	-	100
Piping – Overland	23	km	100	-	-	-
E&I Plant & Infrastructure	1	lot	80	-	20	-
Blockwork Buildings	3228	m²	90	-	10	-
Prefabricated Buildings	14 413	m²	80	15	5	-
Steel Framed Buildings	5879	m²	85	15	-	-

21.2.9.5 Pricing Basis

Estimate pricing was derived from a combination of the following sources:

- Budget Pricing - Market pricing solicited specifically for the project estimate.
- Database - Actual costs from similar projects that have recently been constructed or were under construction at the time of the estimate and are less than six months old.
- Estimated - Historical database pricing older than six months, escalated to the current estimate base date.
- Factored - Factors derived from previous estimates or projects.

Table 21-5 summarises the source of pricing by major commodity, weighted by value of the direct permanent works (excluding temporary works, construction services, commissioning assistance, EPCM costs and contingency), including supply and installation. The 'pricing derivation' basis is presented graphically in Figure 21-2.

Table 21-5: Sources of Pricing

Classification	Subtotal USD (M)	Budget Pricing %	Database %	Estimated %	Factored %
A Architectural	19.67	49	51	-	-
B Earthworks	40.73	90	5	5	-
C Concrete	17.39	90	10	-	-
E Electrical	27.37	75	25	-	-
L Platework	14.28	95	5	-	-
M Mechanical	55.21	70	25	5	-
P Piping	17.23	56	6	-	38
S Steelwork	12.89	95	5	-	-
S1 SMP Indirects	10.83	94	6	-	-
U Owners Costs	163.21	100	-	-	-
V EPCM	30.76	100	-	-	-
W Consultants	1.29	100	-	-	-
Z General	5.63	-	-	100	-
Pricing Derivation	**416.56**	**83%**	**12%**	**3%**	**2%**

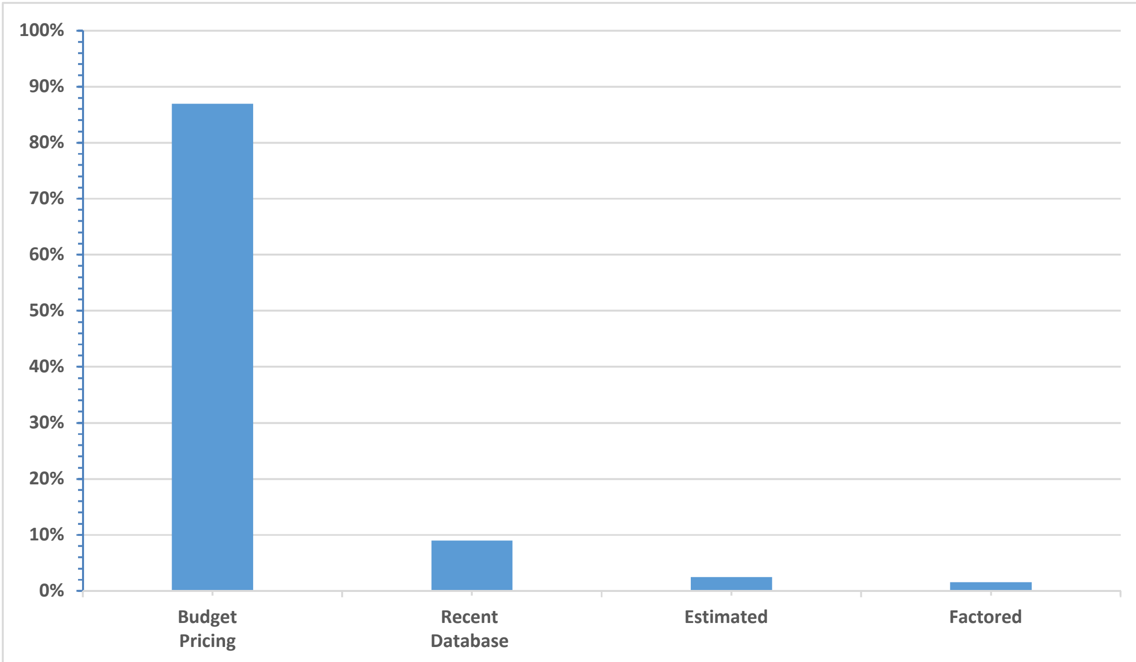

Figure 21-2: Sources of Capital Pricing

21.2.9.6 Bulk Materials

The bulk materials component covers all materials normally purchased in bulk form, for installation on the Project. Costs include; the purchase price ex-works, shop detailing and off-site fabrication (where applicable), and over-supply for anticipated wastage.

21.2.9.7 Plant Equipment

The plant equipment component represents prefabricated, pre-assembled, off-the-shelf mechanical, electrical and instrumentation equipment items. Pricing is inclusive of all costs necessary to purchase the goods ex works, generally excluding delivery to site (unless otherwise stated) but including operating and maintenance manuals (French and English). Vendor representation and commissioning spares have been allowed for separately in the estimate.

21.2.9.8 Installation

The installation component represents the cost to install the plant equipment and bulk materials on site or to perform site activities. Installation costs are further divided between; trade labour, equipment and contractors' distributables.

The labour component reflects the cost of the trade labour workforce (excluding management, supervision and other onsite support staff) required to construct the Project. The labour cost is the product of the estimated work hours spent on site, multiplied by the cost of labour to the contractor inclusive of overtime premiums, statutory overheads, payroll burden, and contractor's margin.

Direct site construction manhours have been estimated for the Project scope, with productivity factors applied based on data from previous projects in a similar geographical area to the Project.

The equipment component reflects the cost of the construction equipment and running costs required to construct the Project. The equipment cost includes small tools, consumables, PPE and the applicable contractor's margin.

Contractors' distributable costs encompass the remaining cost of installation and include items such as off-site management, onsite staff and supervision above trade level, site facilities, cranes (up to 100 t) and crane drivers, mobile equipment, scaffold, mobilisation and demobilisation, rest and relaxation periods (R&Rs), meals and accommodation costs, durations costs general and the applicable contractors' margin. These contractor distributable costs are also referred to as Preliminaries and General Costs (P&Gs).

21.2.9.9 Construction Infrastructure

Project construction offices and establishment, construction services, power, water, PPE, communications, computers, IT services, servers and telephones are all included in the capital estimate.

A heavy lift crane of 250 t capacity has been included in the estimate for a six-month duration, based on the heavy lift requirements from the preliminary construction schedule.

21.2.9.10 EPCM Services

The Project will be implemented using an EPCM approach, whereby the EPCM Engineer will provide design, procurement and construction management services on behalf of the Owner.

The EPCM services cost estimate includes head office support and site staffing, sub-consultants, office consumables, equipment and associated project travel. The cost of a fully equipped home design office and all project computing requirements are included under management costs.

The estimate for EPCM services costs has been based on a preliminary manning schedule for the anticipated Project deliverables and schedule. The resulting EPCM cost estimate is consistent with other projects of this nature in terms of the percentage of the plant capital cost.

The engineering and design component of the EPCM estimate for the home office is based on a calculation of required manning levels to complete the Project and benchmarked against Lycopodium's experience on similar projects.

Engineering, design and procurement is assumed to commence shortly after Endeavour's Board approves the Project, unless initiated early through pre-commitment funding approvals.

21.2.9.11 Spares

A capital value for the Project's operational and strategic spares has been estimated by Lycopodium and included in the estimate. Estimates for wear consumption rates for major equipment are based on vendor data, Lycopodium and OMC databases, and where applicable, are factored to account for differences in process variables.

For the financial model USD 2.2 M of spares has been removed from the CAPEX presented and incorporated into working capital.

21.2.9.12 Escalation

There is no allowance for project escalation in the capital estimate. Escalation should be calculated from the estimated base date of 2nd Quarter of 2022 and included directly in the financial model.

21.3 Operating Cost Presentation

21.3.1 Summary

Operating costs for the Lafigué DFS have been built up from individual cost elements within each business cost centre and reported by year. The basis for the operating cost estimate is the 'Scenario 13' mining schedule presented in Section 16, the 4 Mt/a (db) Plant', TSF, and other supporting infrastructure on the Site. Current reserves indicate an operating mine life of 13 years.

Operating costs are presented in US dollars (USD) based on input pricing from the second quarter of 2022 (2Q22) and have an accuracy provision of ±15%. No contingency has been allowed for operating costs.

The operating costs presented herein, reflect direct production costs for doré bars in the goldroom safe and apply from ore through the mill for the first gold pour. Operating costs prior to this date are capitalised and reported separately as pre-production costs. Additionally, the following cost elements are reported in the financial analysis and not discussed further.

- All operating costs/government payments associated with gold sales/revenue, including gold transporting, vaulting, refining and sale; royalties and community levies.
- Ongoing sustaining capital and closure costs.
- Financing, Joint Venture charges/payments and taxes.

All reagent and consumable costing is based on a Delivery Duty Paid (DDP) basis (Incoterms® 2010) and includes the statutory 2.5% regional/ECOWAS levy. As per the 2014 Cote 'd'Ivoire Mining Code, reagents and fuel are duty exempt from the first commercial production (not exempt from the 2.5% levy). Full duties do apply in the 'Construction Phase' (10% duty for fuel), and for consumables (i.e., grinding media, tyres) in the 'Production Phase.

Corporate costs, including costs associated with regional and head offices and exploration, are not assigned/apportioned to the mine or Project.

The following major cost areas have contributed to the overall operating costs summarised in Table 21-6:

- Mining contractor costs built up from equipment fleet operating hours and fuel usage rates.
- Labour pay rates and manning as advised by Endeavour.
- Diesel cost as advised by Endeavour.
- Grid power cost as advised by ECG Engineering Pty Ltd (ECG) based on CI Energies supply.
- Processing consumable prices as advised by Endeavour (Incoterms® 2010 DDP basis).
- Plant maintenance costs factored from the capital equipment supply cost, using factors from the Lycopodium database.
- Quoted site laboratory operating costs.
- Processing consumable usage and gold recoveries based on metallurgical testwork results.
- General and Administration (G&A) costs as advised by Endeavour based on costs from a similar in-country mine site.
- Constant average gold recoveries over the life of mine given the narrow average head grade range.
- Silver production is assumed to be 5.4% of the recovered gold oz. No silver resource is quoted, so any silver revenue received is an unaccounted project upside.

Table 21-6: Operating Cost Estimate and Production Summary by Year (USD, 2Q22, ±15%)

Ore Weathering/Grade	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	LoM
Fresh (kt)	1633	2901	3891	4011	4014	4000	4000	4003	4008	4000	4000	3851	2554	46 866
Transition (kt)	1192	699	180	0	2	6	0	0	0	0	0	0	0	2 079
Oxide (kt)	441	400	27	0	0	0	0	0	0	0	0	0	0	867
Total kt ore feed	3266	4000	4098	4011	4016	4006	4000	4003	4008	4000	4000	3851	2554	49 813
Avg. Grade (Au g/t)	2.01	1.64	1.52	2.05	1.75	1.85	1.76	1.80	1.91	1.81	1.74	1.25	0.33	1.68
Mining Cost (USD M)	106.4	129.1	131.8	141.1	143.6	142.0	135.4	123.0	94.4	60.2	33.3	22.5	0.1	1262.9
Process Cost (Incl. Rehandle) (USD M)	36.5	45.2	47.2	46.8	46.1	46.7	46.6	46.7	46.7	46.7	46.6	45.2	31.0	577.8
G&A Cost (USD M)	15.6	18.7	18.7	18.7	18.7	18.7	18.7	18.7	18.7	18.7	18.7	18.7	14.0	235.4
Total Cost (USD M)	**158.5**	**192.9**	**197.8**	**206.5**	**208.4**	**207.4**	**200.7**	**188.4**	**159.8**	**125.6**	**98.6**	**86.4**	**45.1**	**2076.1**
Gold Produced (koz)	201	201	190	251	215	226	215	220	234	222	213	147	26	2560
Silver Produced (koz)*	11	11	10	13	12	12	12	12	13	12	11	8	1	138

Table 21-6 notes:

- *Assumed
- Project financial year for this presentation is from start Q2 to end Q1 in the following year.
- Per study schedule, Year 1 is 2024.
- Year 1 tonnes reflect a typically short ramp up to nameplate production, but also a reduced number of operating months.
- Based on reduced tonnes in the final year of operations, year 13 labour and G&A costs were calculated as ¾ of a full year, as advised by Endeavour.

21.3.2 Operating Cost Estimate Breakdown/Derivation

The mine operating costs presented in Table 21-6 by year, are further broken down to a level three cost breakdown structure in Table 21-7 following. Further;

- the overall mine operating costs at a level two cost breakdown structure are presented graphically in Figure 21-3;
- level two/business area costs, namely; mining, processing, tailings disposal, and G&A costs are discussed further in Sections 21.3.2.1, to 21.3.2.4 respectively;
- the build-up/basis for the process costs presented in Table 21-7 are presented in Section 21.3.2.3;
- the build-up/basis for the G&A costs presented in Table 21-7 are detailed in Section 21.3.2.4; and,
- pre-production operating costs incurred up to first gold pour, and subsequently capitalised, are discussed in 21.3.3.

Table 21-7: Operating Cost Estimate Area Summary by Year (USD (M), 2Q22, ±15%)

Business Area	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	LoM
Fixed Contractor (Management Fee)	9.6	12.2	12.1	12.3	12.2	11.8	10.1	9.5	7.2	4.5	1.9	1.3	0.0	104.6
Drill & Blast	18.1	27.0	28.1	27.3	28.8	26.5	24.0	22.6	15.9	10.7	4.7	3.1	0.0	236.8
Load & Haul	73.4	83.7	86.7	96.0	98.4	98.3	95.4	84.5	67.4	40.9	22.1	15.0	0.1	861.8
Contractor Labour, Mine Management Labour, Dewatering, Grade Control Sampling, etc.*+	5.4	6.2	5.0	5.5	4.2	5.4	5.8	6.4	3.9	4.1	4.5	3.2	0.0	59.7
Total Mining	**106.4**	**129.1**	**131.8**	**141.1**	**143.6**	**142.0**	**135.4**	**123.0**	**94.4**	**60.2**	**33.3**	**22.5**	**0.1**	**1262.9**
Power	11.7	14.6	15.5	15.4	15.4	15.4	15.4	15.4	15.4	15.4	15.4	14.8	9.8	189.6
Operating Consumables	12.2	15.3	16.2	15.9	15.9	15.9	15.9	15.9	15.9	15.9	15.9	15.3	10.1	196.3
Maintenance	3.8	4.6	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	4.6	3.0	59.1
Laboratory	2.0	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.3	1.5	29.35
Process & Maintenance Labour	4.5	5.4	5.4	5.4	5.4	5.4	5.4	5.4	5.4	5.4	5.4	5.4	4.0	67.6
Ore Rehandle	2.34	2.86	2.95	2.91	2.19	2.88	2.88	2.89	2.89	2.89	2.86	2.81	2.44	35.78
Total Processing	**36.5**	**45.2**	**47.2**	**46.8**	**46.1**	**46.7**	**46.6**	**46.7**	**46.7**	**46.7**	**46.6**	**45.2**	**31.0**	**577.8**
Administration Labour	4.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	4.4	73.7
General & Administration Costs	10.7	12.8	12.8	12.8	12.8	12.8	12.8	12.8	12.8	12.8	12.8	12.8	9.6	161.6
Total G&A	**15.6**	**18.7**	**18.7**	**18.7**	**18.7**	**18.7**	**18.7**	**18.7**	**18.7**	**18.7**	**18.7**	**18.7**	**14.0**	**235.4**
Total	**158.5**	**192.9**	**197.8**	**206.5**	**208.4**	**207.4**	**200.7**	**188.4**	**159.8**	**125.6**	**98.6**	**86.4**	**45.1**	**2076.1**
Unit Costs USD/t of ore (db)	48.53	48.23	48.26	51.49	51.89	51.76	50.18	47.06	39.88	31.40	24.66	22.43	17.66	41.68

Table 21-7 notes:

- *Allowance for grade control sampling only. Mine sample preparation and assay costs are included with the plant laboratory costs.
- +Costs include; geology, mine planning and contractor management by Endeavour personnel.
- Project year for this presentation is aligned with the financial modelling from start Q2 to end Q1 the following year. This suits a mill production commencement in Q2 of 2024.

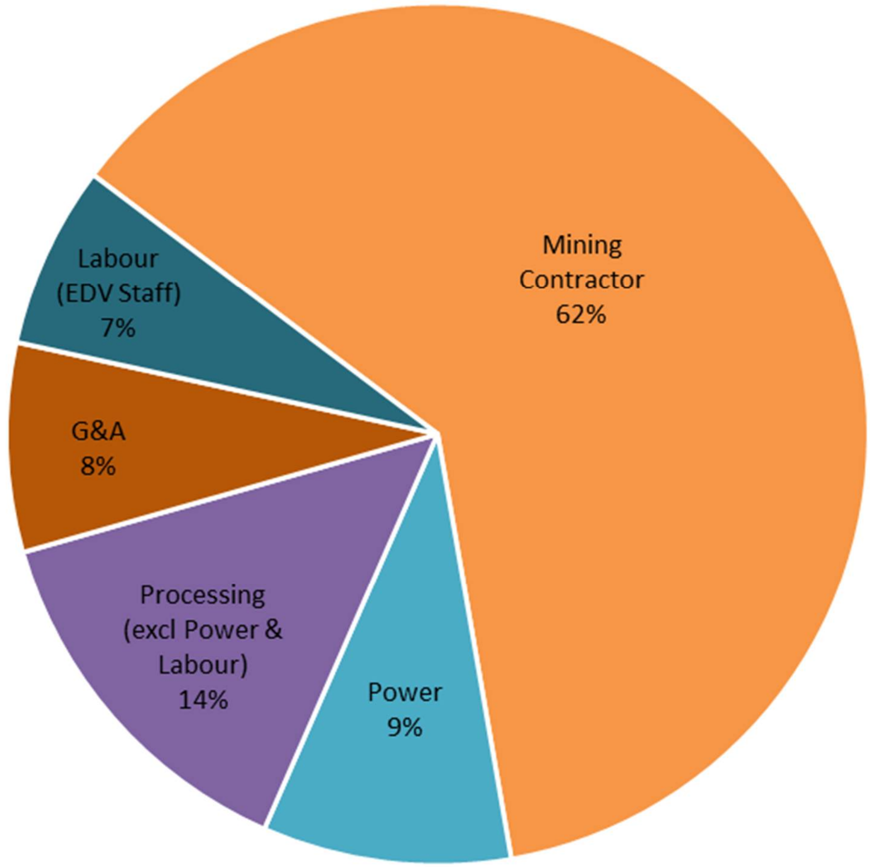

Figure 21-3: Lafigué Average LoM Overall Project/Site Operating Cost Breakdown

21.3.2.1 Mine Operating Costs

Mining contractor costs are estimated for material moved per bench, with drill and blast costs added per pit stage.

The contractor margin is included in all mining costs. Free issue; fuel, flights and accommodation were estimated by SRK/Endeavour, based on contractor advice of required inputs, and are added to the contractor costs.

Crusher feed costs and stockpile rehandle costs are averaged and applied to a percentage of ore tonnes. No allowance has been made for mining equipment to be used to reclaim stockpiled HPGR product and feed this into the milling circuit when the HPGR is offline. The plant will be off RoM feed during these times, so plant mobile equipment can be used inside the plant, for the expected infrequent rehandle duties required. These costs are included under mining contractor costs.

Endeavour mining department staff will manage the contractor and perform key functions such as geology, planning, and grade control sampling. These Owner mine management personnel costs are included with the mining costs.

21.3.2.2 Process Plant Operating Costs

Processing operating costs have been developed by Lycopodium for fresh and blended (fresh/oxide/transition) ores; the average life of mine (LoM) blend based on reserve tonnes would be 94% fresh and 6% oxide/transition material. The LoM processing costs are a weighted average cost based on the LoM blend.

The operating costs have been compiled from a variety of sources, including the following:

- Labour pay rates and manning as advised by Endeavour.
- Power cost as advised by ECG based on grid supply by CI-Energies.
- Consumable prices based on supplier budget quotations from Endeavour advice (DDP pricing basis), or the Lycopodium database.
- Modelling by Orway Mineral Consultants Pty Ltd (OMC) for crushing and grinding energy and consumables, based on Lafigué ore characteristics determined in the metallurgical testwork programme, or benchmarked ore characteristics from the OMC database.
- Reagent consumption and gold extraction based on metallurgical testwork results.
- First principal estimates based on typical operating data/standard industry practise.

The process plant operating cost estimate by cost centre are summarised in Table 21-8, with the relative contribution of each operating cost centre shown in Figure 21-4.

The process operating costs have been split into their respective fixed and variable components, to derive annual costs for changing plant feed blends and/or throughput over the life of mine. The fixed and variable costs are considered valid for throughput variations within ±25 % of the design plant feed throughput.

Table 21-8: Process Plant Operating Cost Estimate Summary

Ore Type **Proportion of LoM** **Plant Feed Mt/a**	**Fresh** **94.2%** **4.0 (db)**		**Transition** **4.1%** **4.0 (db)**		**Oxide** **1.7%** **4.0 (db)**	
Cost Centre	**USD (M)/a**	**USD/t**	**USD (M)/a**	**USD/t**	**USD (M)/a**	**USD/t**
Power	15.4	3.85	14.3	3.57	9.8	2.45
Operating Consumables	15.9	3.97	14.6	3.65	12.3	3.06
Maintenance	4.8	1.19	4.7	1.18	3.6	0.89
Contract Laboratory	2.4	0.59	2.4	0.59	2.4	0.59
Process & Maintenance Labour	5.4	1.34	5.4	1.34	5.4	1.34
Ore Rehandling Cost (from EDV)	2.8	0.69	2.8	0.69	2.8	0.69
Total Processing	**46.5**	**11.63**	**44.1**	**11.02**	**36.1**	**9.02**
• Fixed Component USD M/a	16.5		16.4		14.6	
• Variable Component USD/t		7.51		6.93		5.38

Table 21-8 notes: MSA power and water and grade control assay costs included above.

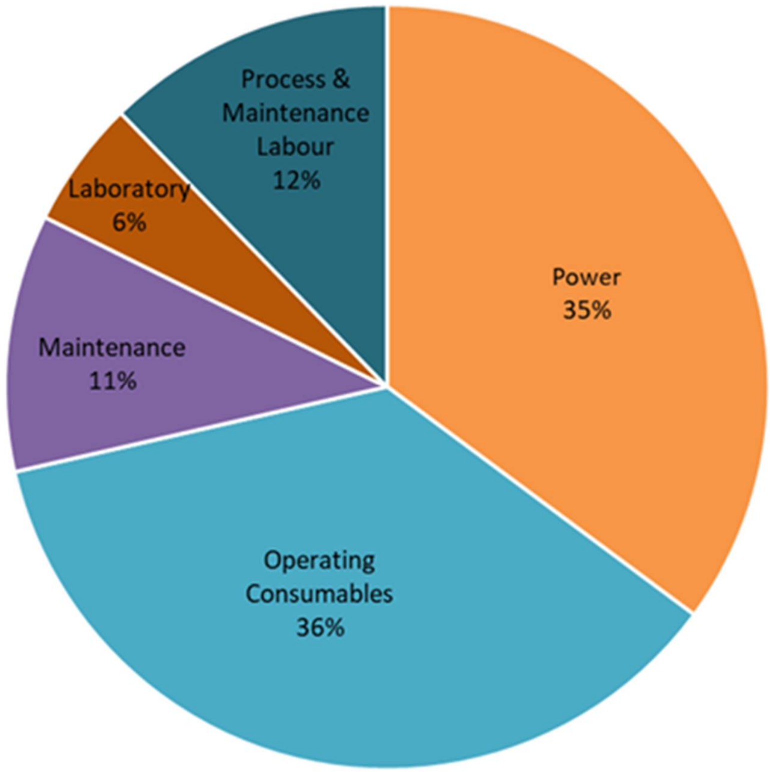

Figure 21-4: Lafigué Average LoM Processing Operating Cost Breakdown

21.3.2.3 Processing Cost Build-Up

Power

The majority of the site power is consumed by the processing plant and supporting process infrastructure, with only 4.2% being used for the camps, miscellaneous site infrastructure and mining support services.

The installed power, average continuous power draw and power cost for each ore type by plant area is summarised in Table 21-9. Further, due to the relatively small balance of facility/mine power usage and costs, these elements have also been included in Table 21-9.

Table 21-9: Process/Mine Power Summary

Plant Area	Installed Power	Fresh Ore			Oxide/Transition Ore		
		Avg. Cont. Draw	Total Power Cost		Avg. Cont. Draw	Total Power Cost	
	kW	kW	USD M/a	USD/t	kW	USD M/a	USD/t
Process Plant							
Crushing and Screening	2 374	1 280	1.25	0.31	1 166	1.14	0.29
Milling & Gravity (Excl. Grinding)	3 867	1 773	1.73	0.43	1 773	1.73	0.43
Grinding Power	10 400	7 743	7.58	1.89	6 712	6.57	1.64
Trash Screens & Pre-Leach Thickener	288	134	0.13	0.03	134	0.13	0.03
Leach/CIP	1 245	871	0.85	0.21	871	0.85	0.21
Elution & Gold Room	1 842	985	0.96	0.24	985	0.96	0.24
Tails Thickening and Pumping	573	232	0.23	0.06	232	0.23	0.06
Reagents	145	37	0.04	0.01	37	0.04	0.01
Water Systems	2 240	813	0.80	0.20	813	0.80	0.20
Air Systems	702	468	0.46	0.11	468	0.46	0.11
Balance of Mine Facilities							
Non-Process Plant/G&A Facilities	2 265	714	0.70	0.17	1 150	1.13	0.28
MSA & Camp	1 131	670	0.66	0.16	234	0.23	0.06
Total	**27 072**	**15 720**	**15.38**	**3.85**	**14 574**	**14.26**	**3.57**
Fixed Power Cost, USD (M)/a			**5.84**			**5.72**	
Variable Power Cost, USD/t				**2.39**			**2.13**

Table 21-9 notes:

- Basis USD 0.112/kWh
- Some discrepancies exist between estimated power requirements in the OPEX Estimate and electrical loads outlined in Section 18 for site infrastructure, due to the different approaches used for calculating installed/drawn power for items within 'Non-Process Plant/G&A Facilities'.

OMC estimated the power consumption for the comminution equipment based on the Lafigué deposit ore characteristics, with additional benchmarking against the OMC database for the HPGR. The power consumption for the remainder of the Plant has been estimated based on the installed power from the mechanical equipment list. Allowances for typical MSA and camp power requirements have been made. Typical drive efficiency and utilisation factors were applied to the installed power to estimate the plant average continuous power draw.

Operating Consumables

Costs for process operating consumables, including; reagents, liners, fuels and process supplies have been estimated and are summarised for the fresh transition and oxide ores by Plant area in Table 21-10. Expected consumption rates for key consumables are summarised in Table 21-11.

Consumable supply pricing has been sourced from purchase contract pricing advised by Endeavour, budget quotations, or in house data relating to similar projects in the region. Transport and freight costs to site and import duties and taxes have been added where applicable.

Table 21-10: Process Consumables Operating Cost Summary

Area Annual Throughput Proportion of LoM	Fresh 4 Mt/a (db) 94.2%		Transition 4 Mt/a (db) 4.1%		Oxide 4 Mt/a (db) 1.7%	
	USD M/a	USD/t	USD M/a	USD/t	USD M/a	USD/t
Crushing	2.3	0.58	1.8	0.46	0.0	0.01
Milling	6.5	1.62	4.3	1.08	1.3	0.32
Leach/Adsorption	2.8	0.70	3.9	0.98	6.0	1.50
Refining	1.5	0.39	1.5	0.39	1.5	0.39
Thickening	1.0	0.24	1.2	0.30	1.6	0.40
Water	0.05	0.01	0.06	0.02	0.06	0.02
Other	0.02	0.01	0.02	0.01	0.02	0.01
Diesel	1.7	0.42	1.7	0.42	1.7	0.42
Total	**15.9**	**3.97**	**14.6**	**3.65**	**12.3**	**3.06**

Table 21-11: Expected Process Plant Consumption Rates

Consumable Usage		Fresh	Transition	Oxide
Jaw Crusher Liners	sets/a	2.5	1.8	1.2
Sec Crusher Liners	sets/a	7.4	5.9	0.0
HPGR Tyre Replacement	sets/a	0.9	0.8	0.0
Ball Mill Liners	sets/a	1.0	0.7	0.5
Ball Mill Media	t/a	795	889	1082
Cyanide	t/a	844	938	1131
Hydrated Lime	t/a	814	3347	8051
Lead Nitrate	t/a	120	85	20
Carbon	t/a	161	161	161
Flocculant	t/a	240	296	400
Hydrochloric Acid	t/a	677	677	677
Sodium Hydroxide	t/a	238	238	238
Diesel	kL/a	1856	1856	1856

Diesel consumption for plant mobile equipment is based on industry standard vehicle consumption rates and estimated equipment utilisation. The diesel usage for carbon elution strip solution heating, has been calculated from first principles.

Allowances have been made for water treatment reagents and operator supplies. Lubricants are excluded here, as they are covered under the maintenance cost centre.

Maintenance

The plant maintenance cost allowance has been factored from the capital supply cost for mechanical equipment, using factors from the Lycopodium database. Said costs for fresh ores and the LoM Blend are summarised in Table 21-12.

Table 21-12: Plant Maintenance Cost Summary

Ore type	**Fresh**		**LOM Blend***	
Annual Throughput	**4 M t/a (db)**		**4 M t/a (db)**	
Proportion of LoM	**94.2%**		**100%**	
Cost Centre	**USD M/a**	**USD/t**	**USD M/a**	**USD/t**
Crushing/Milling/Gravity	2.01	0.50	1.99	0.50
Preleach Thickener/Leach/CIP/Elution	0.65	0.16	0.65	0.16
Tails Thickener & Tails Pumping	0.14	0.03	0.14	0.03
Reagents	0.07	0.02	0.07	0.02
Plant Services	0.08	0.02	0.08	0.02
Water Supply	0.19	0.05	0.19	0.05
Buildings	0.51	0.13	0.51	0.13
Mobile Equipment	0.47	0.12	0.47	0.12
Maintenance General	0.28	0.07	0.28	0.07
Contract Labour	0.38	0.10	0.38	0.09
Total Cost	**4.77**	**1.19**	**4.75**	**1.19**
• Fixed Cost, USD M/a	3.39		3.24	
• Variable Cost, USD/t		0.34		0.38

Table 21-12 notes:* Weighted Average

The maintenance provisions cover mechanical spares and wear parts but exclude crushing and grinding wear components and grinding media which are allowed for in the consumables cost.

The maintenance cost presented excludes payroll maintenance labour, which is included in the labour cost. Contract labour has been allowed for ball mill liner changes and assistance with plant shutdowns. It has been assumed that primary and secondary crusher liner changes will be completed by site personnel. HPGR roll tyre replacement will be conducted off-site at the vendor maintenance facility, with costs for this included in the consumables and labour cost centres. Site personnel will complete the roll change-out with the spare set.

Allowances for mobile equipment servicing, building maintenance and general site infrastructure maintenance expenses have been made.

The mobile equipment servicing allowance has been based on unit costs for maintenance of all light vehicles, site cranes, plant trucks, forklifts and loaders, site generators and minor mobile equipment for the process plant.

The buildings maintenance allowance includes maintenance on plant and non-process buildings and general site infrastructure.

General maintenance expenses include specialist maintenance planning software, maintenance manuals and control system licence fees.

Labour

The labour rates, manning levels and rosters used to determine the labour operating cost estimate were agreed with Endeavour and are based on other Endeavour operations and benchmarking within CI.

The mine will employ some 1555 persons, of which 1266 will be employed on the mine site indirectly through SLAs (Table 21-14), whilst a further 289[1] persons will be employed directly as part of the owner's team. Owner's team labour costs are summarised in Table 21-15.

The owners team; administration, process plant and mining personnel project roles and numbers per position are based on the operations organisation structure provided by Endeavour.

With respect to the development and reporting of costs for the labour associated with SLAs, the following may be noted:

- Contract mining labour costs are included in the mining costs.
- Labour costs associated with the laboratory contract are built into the laboratory contract rate.
- Labour costs associated with the other SLA's noted in Table 21-14 are built into the contractual rate and are discussed more fully in Section 21.3.2.4.

Table 21-13: Lafigué Owner's Team Site Manning Numbers

Department	No. of Employees				
	Local Worker	Local Supervisor	Local Management	Expat Workers	Total EDV Personnel at Site
Administration	31	49	31	5	116[2]
Plant Operations	36	8	5	4	53
Plant/Infrastructure Maintenance	47	13	9	4	73
Mine Management & Mine Technical Services	9	13	22	3	47
Total EDV Personnel	**123**	**83**	**67**	**16**	**289**

[1] An additional seven shared positions have been allowed for in the operating cost estimate but will not be required in the overall site labour numbers, as these people will likely reside in Abidjan.

[2] An additional seven shared positions have been allowed for in the operating cost estimate but will not be required in the overall site labour numbers, as these people will likely reside in Abidjan.

Table 21-14: Lafigué Out-Sourced Service Providers, Manning Numbers for Site

Service Provider	Personnel
Mining Contractor	699[1]
Mining Emulsions & Explosives	37
People Transport Services	0[2]
Logistics Service Provider	0
Fuel Service Provider	6
Site Laboratory Contractor	27
Camp Management and Catering	83
Security Contractor	372
Gendarmes	36
Medical Services	6
Total	**1266**

Table 21-15: Lafigué Owner's Team Staff Labour Costs

Department	Annual Labour Cost (USD M)*				
	Annual Cost	Local Workers	Regional Supervisors	Local Management	Expat. Employees
Administration	5.86	0.52	1.26	2.42	1.65
Plant Operations	2.48	0.76	0.26	0.34	1.13
Plant Maintenance	2.89	1.12	0.30	0.55	0.92
Mine Management & Mine Technical Services	2.99	0.20	0.32	1.51	0.95
Total EDV Labour Cost	**14.22**	**2.61**	**2.13**	**4.82**	**4.65**

Table 21-15 notes*: * Excludes Contract Labour Costs*

For Table 21-15, the labour cost includes all Owner's staff labour costs associated with Site based administration (including some shared head office services), plant operations and maintenance personnel. The mine management team includes; the geology, survey and planning functions in addition to the direct mining contractor management. Apart from the head office resources directly allocated to the Project, these labour cost exclude head office staff costs (apart from what is apportioned to general labour overhead costs).

The site laboratory will be operated on a contract basis with the lab personnel included in the process labour count (for accommodation, catering, and transport provisions), but the labour costs are included under the plant laboratory contract cost.

[1] 35 expatriate personnel (all sourced from Portugal) – mining only

[2] Assumed that there will be no site personnel, to be confirmed in FEED phase

Camp management/catering/cleaning, site security and the medical clinic will be operated on a contract basis, with the appropriate personnel included in the labour count for determining, camp travel and food costs. Actual labour costs are included in the camp contract cost, the security contract cost, and the medical services contract cost.

Endeavour staff labour numbers have been based on a four-panel roster (two shifts working 12 hours per day, rotation shifts on R&R), to provide continuous coverage for Plant operation. Provision has been made in the manning numbers to cover, annual and sick leave requirements.

Camp and transportation costs for the workforce are excluded from the above labour cost, as they are included in the G&A cost.

Laboratory Costs

A SLA will be set up to provide contract laboratory services for the Site. Laboratory costs and the estimated number of samples to be processed by business area, are summarised in Table 21-16 following.

Laboratory costs have been based on a wet assay only contract laboratory using the PAL (accelerated cyanide leaching) gold assay system, but otherwise similar to operations at other Endeavour mine sites. The laboratory cost allows for the supply and maintenance of the laboratory equipment (amortised over the operating term), mobilisation and all ongoing costs (laboratory labour, equipment, and consumables) comprising a fixed monthly cost and a variable cost related to the number of samples being processed. The laboratory building will be provided for the contractor to fit out. The building cost is included in the capital cost estimate.

Table 21-16: Laboratory Cost Summary

Item			Monthly Samples	USD/month	USD (M)/a
Fixed Fee				146 625	1.76
Variable Fee (Based on the following samples)		Internal		50 001	0.60
• Mine Exploration & Grade Control		Internal	6 080		
• Plant Solids (Assay, Moisture, Sizing)	Dry	Internal	90		
•	Slurry	Internal	90		
•	Total	Internal	180		
• Plant Solutions (Assay)		Internal	90		
• Plant Carbon (Assay)		Internal	210		
• Bullion (Au & Ag)		Internal	30		
• Environmental (CN, WAD, Total CN)		External	30		
• Environmental (pH, TSS,TDS,E Coli, Assay)		External	4		
Total				**196 626**	**2.36**

Table 21-16 notes: 'External indicates that the samples will be done in an off-site laboratory

Tailings Costs

All tailings storage, decant water return and raw water supply operating costs are included in the process operating costs.

21.3.2.4 Site General and Administration

The build-up of the G&A expenses presented in Table 21-17 are summarised in the bullet points following.

- **Site office expenses:** Includes costs for communications and communication maintenance, office equipment and supplies, computer supplies and general office software licenses.
- **Insurance expenses**: Includes industrial special risks, third party liability, motor vehicles and operational shipping. Labour associated insurances (medical, death and disability) and workers liability insurances are included and reported in the labour total cost to company (TCTC).
- **Financial expenses:** Includes; banking charges, legal fees, auditing costs and accounting consultants and bullion selling. Bullion transport, vaulting, refining, and royalties are excluded, as they are included separately in the financial model.
- **Government charges:** Includes surficial fees associated with permits and environmental inspection fees. All charges associated with revenue, including royalties and community levies are covered in the financial modelling section.
- **Personnel expenses:** Includes first aid and medical costs, safety supplies, business travel and accommodation for meetings/training, international expat recruiting/relocation costs, training, recreational and local facilities costs, professional memberships and subscriptions, and entertainment allowances. The allowances for international expat recruiting/relocation and other per person costs cater for all personnel (mining as well as administration and processing staff).
- **Service Level Agreements (SLA):** Covers contract costs for personnel transport to site, camp management, catering and cleaning, site security, medical services, environmental compliance testing, OH&S and other consultants. The camp, catering, and cleaning contract cost includes all Endeavour staff personnel (mining as well as administration and processing).
- **Community relations expenses:** Includes general expenses, community projects and scholarships.
- **Other**: General miscellaneous expenses provision.

Table 21-17: Site General and Administration Cost Summary

Area	Annual Cost USD (M)/a
Site Office	**1.13**
Insurances	**1.42**
Financial	**1.45**
Government Charges	**0.55**
Personnel	**1.59**
Community Relations	**0.22**
Other	**0.48**
Contracts - SLA	**6.00**
Worker Transport (Bus) Contract	**1.56**
Camp, Catering and Cleaning Contract	**2.10**

Table 21-17: Site General and Administration Cost Summary

Area	Annual Cost USD (M)/a
Security Contract	**1.58**
Medical Services Contract	**0.54**
Other/Consultants	**0.22**
Total G & A	**12.85**

21.3.3 Process Plant Pre-Production and Working Capital Costs

21.3.3.1 Summary

The costs incurred by operations during the latter stages of construction and commissioning are included in the capital cost estimate but are derived from the operating costs. The pre-production cost estimate for site administration and processing is summarised in Table 21-18. Pre-production costs associated with the mining contactor are presented as the pre-strip allowance with details being presented in Section 16, along with the mining activity ramp up. These basis for the costs presented in Table 21-18 is discussed in Sections 21.3.3.2 to 21.3.3.8.

Table 21-18: Lafigué Pre-Production Cost Summary (excl. Owner Costs)

Cost Centre		USD (M)
Mine*	*Mining Pre-Strip*	*38.78*
	Labour (EDV Staff)	2.68
	Expenses	2.40
	Subtotal	**43.86**
Administration	Labour	3.42
	Expenses	5.61
	Subtotal	**9.02**
Process Plant	Labour	2.63
	First Fill	1.28
	Opening Stocks	2.76
	Vendor Representatives	0.76
	Training	0.17
	Commissioning Operations	0.85
	Working Capital*	0.00
	Subtotal	**8.45**
Total	**(excluding Mining Pre-Strip)**	**22.55**
	(including Mining Pre-Strip)	***61.33***

Table 21-18 notes: *Working capital requirements calculated in the financial model.

21.3.3.2 Pre-Production Labour

Pre-production site administration and processing labour costs reflect the need to recruit key operating personnel in time for them to set up and establish operating procedures and undergo training as required. It is envisaged that manning will build-up over the year preceding plant start-up, with key mining personnel in place still earlier to finalise the mining contract and commence planning ahead of the contractor commencing the pre-strip.

21.3.3.3 Pre-Production Site Administration Expenses

The pre-production site administration expenses cost covers the establishment of site operations during the year preceding start-up to support the early start personnel. This cost includes provision of regular operating supplies and services as well as; power consumed, mobile equipment, contractor costs and other expenses incurred during this period.

21.3.3.4 First Fill Reagents and Opening Stocks

Costs have been allowed to purchase the consumables and reagents required for the process plant first fill and opening stocks.

Sufficient first fill reagents and consumables have been estimated to fill the reagent tanks, charge the ball mill with media, and add the carbon required for the CIL circuit as well as other plant consumable requirements. Opening stocks refer to the purchase of the reagents and consumables required to sustain the operations for twelve weeks, which is the on-site start-up storage quantity nominated by Endeavour to mitigate against future supply disruptions.

Quantities allowed have been based on either consumption over the minimum period or minimum shipping quantities, considering package size.

21.3.3.5 Vendor Representatives

This cost allows for specialist vendor representatives to oversee commissioning of their processing equipment and includes provision for manhours, airfares, and expenses.

21.3.3.6 Training

The training allowance covers the cost of providing pre-production training for process plant operations and maintenance staff, but not the trainer's salaries, as these are covered in the pre-production labour costs. Equipment vendors can be requested to provide specific training on operating and maintaining their equipment while on site.

21.3.3.7 Commissioning Operations

Two months of commissioning operation and process ramp up to achieve first gold have been allowed for. Costs for this period include the full labour contingent with associated G&A costs with a relatively slow process ramp up averaging only 35% uptime for dry plant operations over the first four weeks, increasing over the second four weeks to full time operation, building the stockpile and filling the HPGR surge bin and product stockpile. Milling operations will commence in the second month, building up to 80% operating time to achieve the required gravity gold and CIL recovery for the first gold pour.

21.3.3.8 Working Capital

Working capital would typically cover the cost of operating the Project from commissioning until the sufficient revenue is generated from bullion sales. This capital provision ensures that cash flow is maintained until the Project becomes self-funding and indicates the amount of project financing required. Working capital is calculated and reported separately in the financial and economic analysis section.

21.4 Capital Cost Presentation

21.4.1 Summary

The capital cost estimate reflects the Project scope described in this study report and has been peer reviewed for acceptance by the study team. All costs are expressed in United States Dollars (USD) unless otherwise stated and are based on 2Q22 pricing.

The capital cost estimate is summarised by main area (WBS Level 2) in Table 21-19 and by sub-area (WBS Level 3) in Table 21-20.

The estimate presented for the capital cost is based on a 4 Mt/a (db) production throughput.

Table 21-19: Capital Estimate Summary by Main Area (WBS Level 2)

Main Area (WBS Level 2)	USD (M)
000 Construction Distributables	37.38
100 Treatment Plant Costs	96.61
200 Reagents and Plant Services	23.79
300 Infrastructure	84.52
400 Mining	60.36
500 Management Costs	33.97
600 Owner's Project Costs	79.93
700 Owner's Operation Costs (Working Capital)	Excl.
Subtotal	**416.56**
Contingency	43.03
Taxes & Duties	5.64
Escalation	Excl.
Estimated Total	**465.23**

Table 21-20: Capital Cost (USD) and Manhour Estimate Breakdown by Sub-Area (WBS Level 3)

Sub-Area (WBS Level 2/Level 3)	Supply Cost	Site Manhours	Direct Labour Cost	Direct Equip Cost	Construction Distributables Cost	Project Ancillary Cost	Freight Cost	Subtotal Cost	Contingency Cost	Taxes & Duties Cost	Project Total
0.1 – Construction Distributables - Contractors	-	1 887 334	-	-	26 850 708	-	-	26 850 708	4 110 848	-	30 961 556
0.2 – Site Construction Distributables - General	-	594	11 078	1901	1 207 086	200 000	-	1 420 065	207 010	-	1 627 075
0.3 – Site Construction Facilities	332 818	4831	47 182	16 278	62 000	-	82 175	540 453	80 543	12728	633 724
0.4 – Site Construction Facilities Other	9000	600	7500	3000	-	-	2250	21 750	3263	410	25 422
0.5 – Construction Operations	-	-	-	-	736 070	90 000	-	826 070	123 911	-	949 981
0.6 – Construction Accommodation	2 586 675	163 211	1 445 572	841 520	2 120 210	-	725 127	7 719 104	978 687	89 837	8 787 628
0 – Construction Distributables Total	**2 928 493**	**2 056 569**	**1 511 332**	**862 699**	**30 976 075**	**290 000**	**809 552**	**37 378 151**	**5 504 260**	**102 974**	**42 985 385**
1.1 – Treatment Plant – General	188 348	84 440	1 248 704	3 647 655	-	-	54 912	5 139 619	526 406	-	5 666 025
1.2 – Feed Preparation	12 066 624	231 740	2 030 425	1 046 107	-	-	2 877 018	18 020 174	1 935 260	497 453	20 452 886
1.3 – Milling	28 241 051	517 906	4 585 664	2 599 798	949 800	-	6 690 960	43 067 273	4 594 921	1 221 865	48 884 058
1.4 – Trash Removal & Thickening	2 992 879	70 047	628 110	310 253	-	-	986 195	4 917 436	533 439	136 625	5 587 500
1.6 – Leaching	8 225 109	164 684	1 782 368	1 048 798	-	-	2 994 994	14 051 268	1 598 774	394 125	16 044 166
1.7 – Elution & Gold Room	4 852 820	88 537	828 291	488 061	-	-	1 127 382	7 296 554	844 792	213 806	8 355 152
1.8 – Tails Handling	2 511 981	58 208	476 518	269 581	-	-	858 267	4 116 347	474 813	115 300	4 706 460
1 – Treatment Plant Costs Total	**59 078 812**	**1 215 562**	**11 580 080**	**9 410 252**	**949 800**	**-**	**15 589 727**	**96 608 671**	**10 508 405**	**2 579 173**	**109 696 248**
2.1 – Reagents	1 700 808	36 706	347 263	194 163	-	-	420 611	2 662 846	321 438	75 179	3 059 463
2.2 – Water Services	2 883 221	61 229	693 000	389 635	-	-	617 694	4 583 550	563 586	125 390	5 272 527
2.3 – Plant Services	884 480	20 430	282 580	68 117	-	-	183 191	1 418 368	174 662	37 865	1 630 895
2.4 – Air Services	1 707 241	29 321	335 629	186 485	-	-	350 841	2 580 197	331 439	74 810	2 986 446
2.5 – Fuels	31 842	528	6140	4042	-	-	6127	48 151	6487	1401	56 039
2.6 – Electrical Services	9 942 663	72 901	1 760 559	-	-	-	795 413	12 498 636	1 249 864	387 764	14 136 263
2 – Reagents & Plant Services Total	**17 150 255**	**221 115**	**3 425 173**	**842 443**	**-**	**-**	**2 373 877**	**23 791 748**	**2 647 476**	**702 409**	**27 141 633**

Table 21-20: Capital Cost (USD) and Manhour Estimate Breakdown by Sub-Area (WBS Level 3)

Sub-Area (WBS Level 2/Level 3)	Supply Cost	Site Manhours	Direct Labour Cost	Direct Equip Cost	Construction Distributables Cost	Project Ancillary Cost	Freight Cost	Subtotal Cost	Contingency Cost	Taxes & Duties Cost	Project Total
3.1 – Infrastructure – General	719 324	91 988	1 364 834	3 929 772	-	6 326 907	227 312	12 568 148	882 788	3296	13 454 232
3.2 – Environmental	-	7568	124 512	373 535	-	-	-	498 046	49 805	-	547 851
3.3 – Water & Sewerage	6 366 918	151 198	2 038 716	4 725 512	-	-	1 324 113	14 455 259	1 818 147	272 634	16 546 041
3.4 – Power Supply	271 248	68 238	383 926	527 775	-	15 125 842	3833	16 312 623	1 683 792	352 525	18 348 941
3.5 – Tailings Dam	6 203 924	206 788	3 051 537	8 518 367	-	-	269 796	18 043 624	1 839 985	49 800	19 933 409
3.6 – Buildings – Admin & Security	2 336 453	88 759	735 963	470 110	-	-	1 001 423	4 543 948	564 658	46 803	5 155 409
3.7 – Buildings – Plant	4 822 477	144 280	1 246 582	557 894	-	-	1 238 950	7 865 903	985 216	147 792	8 998 911
3.8 – Permanent Accommodation	3 912 298	68 145	920 152	985 175	-	-	686 073	6 503 698	769 331	84 582	7 357 611
3.9 – Gendarme Barracks	1 662 472	65 748	805 444	411 284	-	-	853 925	3 733 125	437 314	55 877	4 226 316
3 – Infrastructure Total	**26 295 113**	**892 711**	**10 671 666**	**20 499 424**	**-**	**21 452 749**	**5 605 424**	**84 524 376**	**9 031 037**	**1 013 308**	**94 568 721**
4.1 – Mining – General	-	10 004	150 062	450 186	-	10 569 227	-	11 169 475	588 486	-	11 757 961
4.2 – Mine Establishment	-	-	-	-	-	38 783 063	-	38 783 063	1 939 153	-	40 722 216
4.3 – Mining Pre-Production	-	-	-	-	-	5 074 949	-	5 074 949	445 521	-	5 520 470
4.5 – Mining Facilities	129 772	826	8324	3642	-	3 591 427	32 175	3 765 340	385 085	5869	4 156 294
4.8 – Mine Facilities – Other	-	-	-	-	-	1 565 992	-	1 565 992	234 899	-	1 800 890
4 – Mining Total	**129 772**	**10 831**	**158 386**	**453 828**	**-**	**59 584 658**	**32 175**	**60 358 819**	**3 593 144**	**5869**	**63 957 832**
5.1 – EPCM – Home Office	-	-	-	-	-	30 767 545	-	30 767 545	3 084 555	-	33 852 100
5.3 – Specialist Consultants – Design	-	-	-	-	-	2 439 328	-	2 439 328	243 933	670 815	3 354 075
5.5 – Vendor Representatives	-	-	-	-	-	763 750	-	763 750	99 288	-	863 038
5 – Management Costs Total	**-**	**-**	**-**	**-**	**-**	**33 970 623**	**-**	**33 970 623**	**3 427 775**	**670 815**	**38 069 212**

Table 21-20: Capital Cost (USD) and Manhour Estimate Breakdown by Sub-Area (WBS Level 3)

Sub-Area (WBS Level 2/Level 3)	Supply Cost	Site Manhours	Direct Labour Cost	Direct Equip Cost	Construction Distributables Cost	Project Ancillary Cost	Freight Cost	Subtotal Cost	Contingency Cost	Taxes & Duties Cost	Project Total
6.1 – Owners Costs – General	-	-	-	-	-	36 692 679	-	36 692 679	3 704 921	-	40 397 600
6.2 – Plant & Admin Pre-Production	-	-	-	-	-	17 209 630	-	17 209 630	2 217 161	-	19 426 791
6.3 – Admin Pre-Production Other	-	-	-	-	-	170 000	-	170 000	22 100	-	192 100
6.4 – Spare Parts	6 653 608	176	2539	1185	-	1 909 475	1 984 036	10 550 843	1 150 558	259 491	11 960 892
6.6 – Community	-	-	-	-	-	7 569 235	-	7 569 235	161 615	-	7 730 850
6.7 – Plant Mobile Equipment	7 736 448	-	-	-	-	-	-	7 736 448	1 063 016	301 721	9 101 185
6 – Owners Project Costs Total	**14 390 056**	**176**	**2539**	**1185**	**-**	**63 551 019**	**1 984 036**	**79 928 836**	**8 319 371**	**561 212**	**88 809 419**
Grand Total	***119 972 501***	***4 396 964***	***27 349 175***	***32 069 831***	***31 925 875***	***178 849 049***	***26 394 792***	***416 561 223***	***43 031 468***	***5 635 760***	***465 228 451***

Table 21-20 notes: Project Ancillaries are either 'Indirect Costs' or costs not directly related to the Project 'Direct Costs' but are borne by the Project. Said costs include, the off-site power transmission line, community payments, pre-production costs etc.

The CAPEX summary presented in Table 21-19 and Table 21-20 was subsequently revised by Endeavour, considering transport savings that are being realised (commercial contracts), a change to how operational spares are incorporated in the financial model, and the removal of Project/Site early works costs expended between 31 December 2021 and 1 June 2022 (the 'Effective Date' of the DFS/Report). The revised estimate as applied in the financial model is presented in Table 21-21.

Table 21-21: Revised Capital Estimate Summary (Endeavour, 2022b)

Main Area (WBS Level 2)	USD (M)	Comment
Estimate Total	465.23	From Table 21-19.
Transport Savings	-8.70	33% savings banked, based on updated transport costs
Spares	-2.20	Moved into working capital (Section 21.2.9.11)
Sunk costs	-6.19	Project/Site development costs incurred from 31 December 2021 and 1 June 2022 (Section 21.2.6.1)
Revised CAPEX Total	**448.14**	**Applied in Section 22, Financial Analysis**

A Monte Carlo analysis (Section 21.4.2.2) was conducted on elements of the capital cost estimate and the results provide confidence that the contingency included in the estimate, previously calculated by a deterministic assessment, is sufficient for a P_{80} (or better) confidence level with event modelling turned off and a P_{50} (or better) confidence level with the event modelling turned on.

21.4.2 Contingency

21.4.2.1 Overview

An amount of contingency has been provided in the estimate to cover anticipated variances between the specific items allowed in the estimate and the final total installed Project cost. The contingency does not cover scope changes, design growth or the listed qualifications and exclusions.

Contingency has been applied to the estimate using a deterministic approach by assessing the level of confidence in each of the defining inputs to the item cost including engineering, estimate basis and vendor or contractor information.

A contingency analysis has been applied to the estimate that considers scope definition, materials/equipment pricing and installation costs. Contingency applicable to various Owner's inputs have been specified by Endeavour.

The resultant overall contingency for the Project on the capital cost estimate is 10.3%.

21.4.2.2 Monte Carlo Analysis

A Monte Carlo analysis was conducted for the Project, to provide confirmation that the capital cost estimate has sufficient contingency to mitigate the various risks that could affect the Project completed cost, and to determine the estimate confidence level expressed as a percentile that provides the cumulative probability up to the estimate value, including contingency.

Scope

The Monte Carlo analysis assessed all scope items and costs that currently reside in the Project capital cost estimate compiled by Lycopodium.

The Monte Carlo analysis did not assess Project costs that are outside of the capital cost estimate, i.e., Endeavour costs that may be direct input into the financial model and not captured in the capital cost estimate.

Software

The cost model used to perform the Monte Carlo analysis was provided by Lycopodium, and was used to compile the estimated costs, input ranges and variables to enable the Monte Carlo assessment using the @Risk® software.

@Risk® is a proprietary Excel plug-in provided by Palisade that is widely used in the cost engineering industry to perform Monte Carlo simulations to assess risk for major construction and engineering projects.

Preparation

In preparation for the Monte Carlo process, the cost model was prepopulated with data from the latest capital cost estimate revision, summarised by supply packages, construction packages and major cost elements, to allow assessment of the ranges and variables for the estimated costs.

The estimating team in conjunction with the Lycopodium project delivery team, considered the ranges for each of the cost elements and prepopulated the cost model to provide initial guidance for the Monte Carlo workshop.

Monte Carlo Workshop

A Monte Carlo workshop that incorporated key stakeholders from both Endeavour and Lycopodium was held on 19 May 2022.

Range Analysis

The first phase of the Monte Carlo assessment was to conduct a range analysis where each element of the cost model was assessed for scope (quantity), supply cost and installation/construction cost, with percentage ranges assigned for 'minimum', 'maximum' and 'most likely' project outcomes.

The intent of the range analysis was to provide realistic range inputs to the cost model that would reflect likely real-world outcomes.

The quantity assessment ranges for each discipline and cost element were discussed and the pre-work guidance was considered. Where required, the ranges were adjusted by consensus and agreement of the workshop attendees.

The rates assessment ranges for each discipline and cost element were discussed and the pre-work guidance was considered where applicable. The discussion included consideration of the workshop attendee's previous project experience, and/or direct knowledge of the vendors and contractors performance on Lafigue and other projects. The cost ranges were then decided by consensus and agreement and included in the cost model.

Event Modelling

The second phase of the Monte Carlo workshop was to consider event modelling and wider project risks that can increase the project cost during implementation.

Event modelling adds an additional layer of risk assessment to the cost model and can be turned on or off, when running the Monte Carlo simulation. This allows the impact of event modelling on the estimate confidence level to be considered separately.

Contingency provisions for event modelling risks are not expected to be captured by the deterministic contingency assessment when compiling the estimate and can be considered as cost impacts over and above the contingency amount included in the capital cost estimate.

Should the results from the Monte Carlo simulation that has event modelling turned on, indicate that the capital cost estimate including contingency is below the 50% probability (P50) confidence level, specific contingent sums would then be added directly to the capital cost estimate. The cost model would then be updated, and the @Risk simulation re-run. This was not required at this workshop.

Risks that would fall under event modelling were discussed and added to the cost model. The current Project risk register was reviewed as part of this assessment. The relevant risk events were then assessed for estimated impact in project cost, should these events occur, and the likelihood of the events occurring expressed as a percentage.

Monte Carlo Simulation

Upon completion of the range analysis and event modelling phases of the workshop, the Monte Carlo simulation was run with 10 000 iterations per simulation. Multiple simulations were run to prove consistency in the cost model as the results of each simulation will inherently have a slight variance.

The simulations were run with event modelling turned off and then re-run with event modelling turned on, so that the difference in estimate confidence levels could be assessed.

Monte Carlo Results

- Simulation Results - Event Modelling Off

  The Monte Carlo simulation was run with the event modelling turned off, providing confirmation that the capital cost estimate total of USD 468.3 M (Capital Cost Estimate Rev F1), including the deterministic contingency provision, is sufficient to mitigate known risks to a confidence level of P_{80} or better. Refer to Figure 21-5 for a graphical representation of the results.

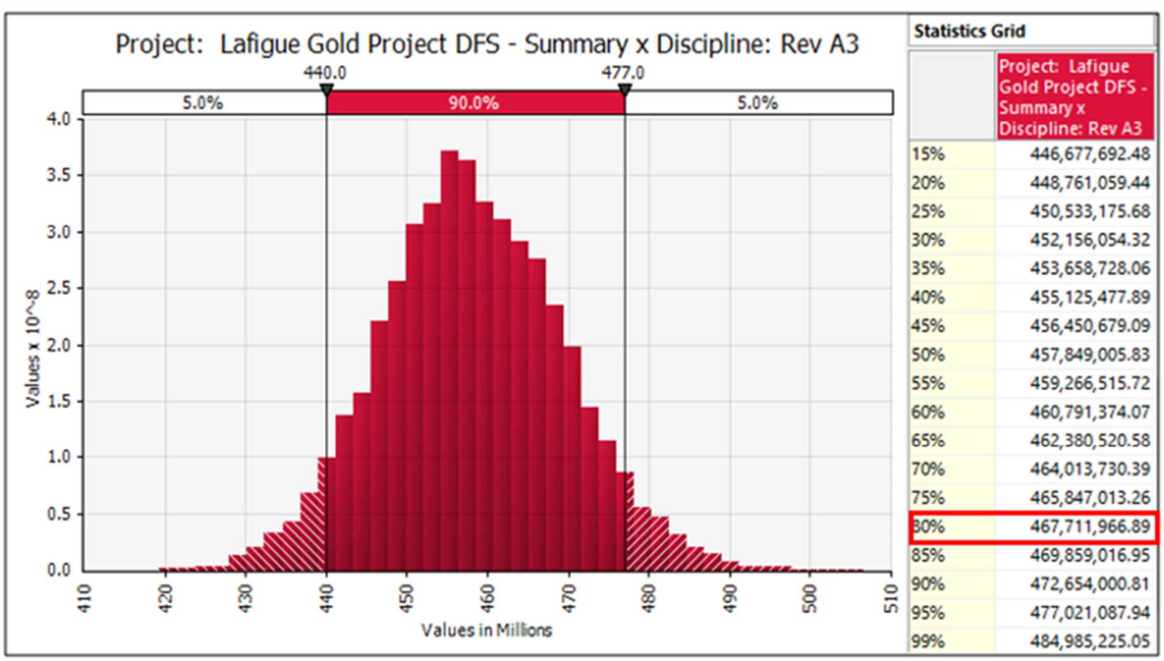

Figure 21-5: Estimate Confidence Level Excluding Event Modelling

- Simulation Results – Event Modelling On

  The Monte Carlo simulation was then re-run with event modelling turned on, providing confirmation that the capital cost estimate of USD 468.3 M (Capital Cost Estimate Rev F1), including the deterministic contingency provision, is sufficient to mitigate known risks including event modelling to a confidence level of P_{50} or better. Refer to Figure 21-6 for a graphical representation of the results.

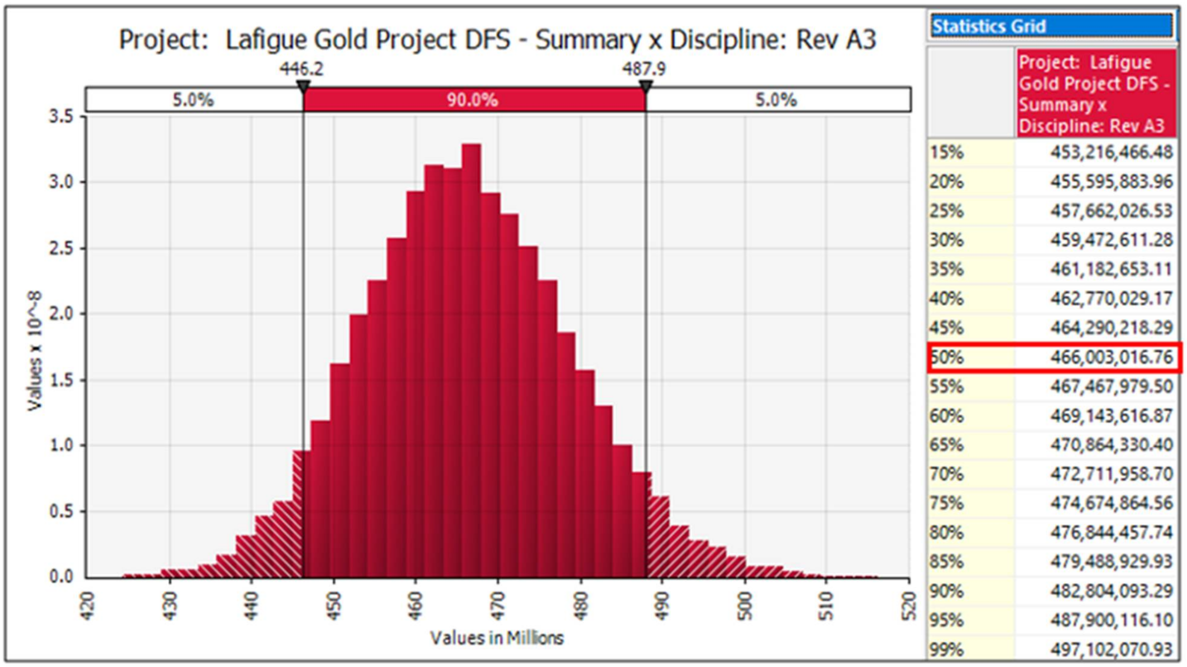

Figure 21-6: Estimate Confidence Level Including Event Modelling

Monte Carlo Conclusion

The Monte Carlo workshop prompted broad discussion on expected ranges and potential outcomes for the major cost elements and packages that make up the Project capital cost estimate.

The Monte Carlo results provided confidence that based on the assumptions, the contingency included in the estimate, previously calculated by deterministic assessment, is sufficient for a P80 (or better) confidence level with event modelling turned off and a P_{50} (or better) confidence level with the event modelling turned on.

Therefore, it can be concluded that the estimated amount of contingency currently in the capital cost estimate is sufficient to service the Project, and the expectation should be that most of the contingency will likely be spent.

21.4.3 Sustaining Capital

The basis for the development of sustaining capital costs, and their presentation by year is summarised as follows. Sustaining capital costs are presented in the financial analysis Section 22, and not in the capital cost summary.

21.4.3.1 Mining

The sustaining capital costs associated with mine mobile equipment and infrastructure are included within the mining contractor's rates, and as such; are not reported separately or here.

21.4.3.2 Plant and Infrastructure

Sustaining capital costs for plant and infrastructure is typically based on a factor applied to the direct capital cost for Plant and Infrastructure, or the direct cost by discipline (i.e., earthworks, concrete, platework, structural steel, etc.) for Plant and Infrastructure. It is important to note that typical industry factors applied are based on a 20 to 30 year plus useful life of facilities. In the case of this Project, Endeavour considers it appropriate to reduce the magnitude of the factors applied based on a shorter facility life.

For the estimate of sustaining costs, a Plant (only, excluding plant services) direct cost of USD 78 M was used, and a factor of 3 % was applied to obtain the maximum annual sustaining capital costs (USD 2.4 M). This was later on a year-by-year basis, modified to reflect that sustaining costs in the first and last years of operation will be less than the maximum indicated. This approach gave a weighted average annual sustaining factor of 1.7 % (USD 1.5 M/a) of the direct Plant cost over an 11-year Plant life. Sustaining costs by year as applied in the financial model are as noted in Table 21-22.

Table 21-22: Plant Sustaining Costs

	2024	2025	2026	2027	2028	2029	2030	2031	2033	2034	2035	Total
Factor applied	0.5%	1.5%	2.0%	2.5%	3.0%	3.0%	3.0%	3.0%	1.0%	1.0%	1.0%	
Sustaining Capital USD (M)	0.4	1.2	1.6	2.0	2.4	2.4	2.4	2.4	0.8	0.8	0.8	**16.9**

21.4.3.3 Tailings

In accordance with the mine plan (Schedule 12), KP defined the volume of material required for each lift, whilst Endeavour applied rates to define the sustaining capital requirements (Table 21-23).

Table 21-23: Tailings Storage Facilities Lifts/Sustaining Capital

	2024	2025	2026	2027	2028	2029	2030	2031	2033	Totals
TSF Lift		2	3	4	5	6	7	8		
Sustaining Capital USD (M)			3.7	3.9	3.3	3.8	3.8	3.8	3.4	25.6

21.4.4 Closure Costs

The basis for the development of the closure costs and the presentation thereof, is discussed in Sections 4 and 20. Closure costs are applied in Section 21.

21.5 Data Verification

The approach to verifying and presenting the data used in Section 21, is discussed in Section 12.

21.6 Comments on Section 21

21.6.1 Mining and MSA CAPEX & OPEX costs

The CAPEX & OPEX estimate for mining is based on two detailed tenders, with clearly defined commercial terms. The QP for the mining cost development and presentation believes that the data as presented is suitable for a DFS and is in accordance with the accuracy provisions stated.

21.6.2 Processing and Infrastructure CAPEX Costs

The QP for the Plant and Infrastructure components of the capital cost estimate, is of the opinion that the cost estimates developed are valid and the data used, and subsequent outputs, are aligned with the requirements of the DFS/Report and are within the limits of the DFS accuracy provision.

21.6.3 Processing and G&A OPEX Costs

The QP for the Plant and G&A components of the operating cost estimate, is of the opinion that the cost estimates developed are valid and the data used, and subsequent outputs, are aligned with the requirements of the DFS/Report and are within the limits of the DFS accuracy provision.

21.7 Interpretations and Conclusions

Interpretations, conclusions and risks for Section 21, are presented in Section 25 of this Report.

21.8 Recommendations

Recommendations for Section 21 are presented in Section 26 of this Report.

21.9 References

References cited in the preparation of Section 21 are presented in Section 27 of this Report.

22. ECONOMIC ANALYSIS

22.1 Introduction

The following economic analysis presents the business case for the Issuer's interests in the Project in CI. The data as presented is sourced from the Lafigué Project Definitive Feasibility Study (DFS) or 'Study' (Lycopdium, 2022).

22.2 Economic Basis and Assumptions

Section 22.2.1 to 22.2.18 following outlines the inputs and associated basis of the financial model.

22.2.1 Base Date

The 'Base Date' for the CAPEX and OPEX estimate, and the associated Financial Model is 1 June 2022 or Q2 2022.

22.2.2 Exchange Rates

The exchange rates used in this Study, are as defined in Section 19 and 21.

22.2.3 Metal Pricing

As per Section 19, a gold price of USD 1500/oz has been used in the financial model. The price given is in accordance with the Issuer's internal standards for metal pricing (Endeavour, 2022a). This data, in combination with information on the cost of transport and refining the product (Section 22.2.4), is used to calculate the royalties and levies payable to the Government of CI (Section 22.2.6).

22.2.4 Freight and Product Treatment Charges

Gold product freight, vaulting, refining and metal pay abilities applied in the financial model are USD 2.86/oz and 99.95% (Endeavour, 2022a).

Table 22-1: Cost of Metal Sales (Endeavour, 2022a)

Mine and/or Project	Gold Grade	Shipment Size	Transport Charge[144]	Vaulting Charge	Refining/ Treatment Charge	Payability	
	(% m/m)	(kg)	(USD/ Au ozt)	USD/Au ozt)	(USD/ozt)[145]	Au (%)	Ag (%)
Lafigué Project	93±2%	>250	2.21	0.35	0.30	99.95	99

22.2.5 Discount Rate

In accordance with the Issuer's internal standards, a 5% discount rate has been applied for the purpose of calculating the Net Present Value (NPV) of the Project.

[144] Transport charge determined by calculating the total mass transported (gold + silver + box) and dividing by the quantity of gold. Effectively, no transport charges for silver for the grade specified. Insurance included

[145] Total ounces including Au, Ag + other.

22.2.6 Royalties and Levies

The royalties and levies payable on gold revenue, as per the gold price used (USD 1500/oz), and as applied in this financial model, are defined in Section 4.

22.2.7 Taxation and Other Payments

Taxes and other statutory payments applied in the financial model are as defined in Section 4. Importantly, taxes and other statutory payments payable for both the 'Exploitation Permit' holder (SML) and its appointed contractors for both the 'Construction' and the 'Production'[146] period, will only be fully defined after the 'Lafigué Mining Convention' for PE 58 has been approved/signed.

With the exception of the 'Business Patente Tax' (BPT), it has been assumed that for SML, taxes and other statutory payments will be as per the applicable CI Mining and Tax codes as stated in Section 4, and no exonerations or favourable terms will be granted.

The permit holder is granted an exoneration on 'BPT' for up to three years after first gold pour. For the financial model however, it has been assumed that SML in the Lafigué Mining Convention will obtain full BPT exoneration over the LoM. The impact of this assumption is discussed in Section 25.16.

Subject to the terms of the Lafigué Mining Convention signed, other favourable terms may or may not be obtained, for SML and the appointed mine contractors/service providers.

22.2.8 JV/Minority Share Holder Costs

The financial model presented is on a project basis, and therefore includes 100% of the cashflows of the project. Details of the Government of CI's 10% free carry interest, and SODEMI's 10% equity stake are described in Section 4.

22.2.9 Escalation/Inflation

All capital and operating costs as reported herein are as per the Base Date. No escalation has been allowed for.

22.2.10 Sunk Costs

Sunk costs as defined in Section 21, have not been included in the financial model.

22.2.11 Hedging

Foreign currency is not hedged; hence no hedging charges are incorporated in the financial model.

22.2.12 Financing Charges

Working capital, all capital expenditure (including initial project capital) and Tailings Storage Facility (TSF) expansions) are assumed to be self-funded by the Issuer, and thus no financing charges are included in the financial model.

[146] Construction and Production definitions from a mining code perspective, are defined in Section 4.

22.2.13 Mine and Production Schedule

A full Life of Mine (LoM) production summary, comprising the mining and process schedules as applied in the financial model are presented in Section 16, and in Table 22-2 following (SRK, 2022). Importantly, whilst the mine production schedule is aligned to the calendar year, the gold production schedule provided in this section, is by 12-month period, from the date of first gold pour (Q2 2024).

22.2.13.1 OPEX Summary

Operating expenditure (OPEX) costs are cash flowed by year and are aligned with the LoM production schedule (Section 22.2.13), and the associated ore processing characteristics over the LoM.

OPEX costs by year and as applied in the financial model, are presented in Section 21.

22.2.14 CAPEX Summary

A full Capital Expenditure (CAPEX) summary as applied in this financial model, is presented in Section 21. In reviewing the CAPEX costs presented, consideration should be given to the approach used to define 'Sunk Costs' (Section 22.2.10) and 'Pre-production Costs' (Section 22.2.15).

22.2.15 Pre-Production Costs

The basis for determining pre-production costs is as defined in Section 21.

In summary, with the exception of 'Sunk Costs', all operating costs up to first gold pour have been capitalised. Pre-Production costs should also be viewed in conjunction with the Project development schedule presented in Section 24.

22.2.16 Working Capital Costs

The current economic model assumes a forward-looking level of the working capital balances for inventories, accounts payable and accounts receivable. Working capital is returned to a nil balance at the end of the mine life.

The costs associated with the mining pre-production pre-strip have been allocated to stockpile inventories within working capital and are utilised over the life of the mine and returned to a nil balance at the end of the mine life.

22.2.17 Sustaining Capital Costs and All in Sustaining Capital Costs (AISC)

The basis for the derivation of annual sustaining capital costs for the mine, is presented in Section 21. The costs are presented annually in the financial model.

All in sustaining capital costs (AISC) by year and over the LoM are presented in Section 22.3.2.

22.2.18 Closure and Salvage Costs

A summary of the 'closure costs', as applied in this financial model, are as defined in Section 20. The model assumes a 'salvage value' of zero at the end of the mine life.

The basis for the payment of closure costs (20% of the final closure costs are paid in annual increments over the life of mine, with a commercial bond taken out for the remaining 80%). Further detail is provided in Section 4.

22.3 Financial Results

22.3.1 Summary

The economic model show robust financial results; and applying a long-term gold price of USD 1500/oz on a flat line basis from the Base Date, delivers:

- an undiscounted LoM net after-tax cash flow on a 100% basis, of USD 803 M;
- an after-tax NPV5% of USD 477 M;
- an after-tax IRR of 21%; and
- a Project payback period of 4.2 years.

The LoM average cash cost per ounce is USD 721, and with the addition of royalties and sustaining capital, the LoM average 'All In Sustaining Cost' per ounce (AISC/oz) is USD 871.

Based on the production schedule (Table 22-2), a summary from the economic model output, is presented in Table 22-3.

Various NPV scenarios; before and after tax; on a 100% ownership basis are considered, and at zero and five per cent discount rates, are presented in Table 22-4.

Table 22-2: LoM Production Schedule

	Units	Totals	Pre-Prod.	Y-1	Y-2	Y-3	Y-4	Y-5	Y-6	Y-7	Y-8	Y-9	Y-10	Y-11	Y-12	Y-13
Mining																
• Total Material Mined	Mt (db)	491.61	14.97	45.60	54.93	55.00	55.15	55.40	52.52	46.71	43.51	32.72	19.92	9.08	6.08	0.03
• Waste	Mt (db)	441.80	13.09	41.88	50.17	51.00	50.81	52.02	48.12	41.87	38.61	29.23	16.18	5.39	3.42	0.01
• Ore	Mt (db)	49.81	1.88	3.72	4.76	4.00	4.34	3.37	4.40	4.84	4.89	3.49	3.75	3.68	2.66	0.02
• Au Grade	g/t	1.69	1.46	1.46	1.44	1.47	1.92	1.96	1.72	1.54	1.57	2.10	1.89	1.86	1.71	1.96
• Contained Gold	koz	2714	88	175	220	189	267	213	243	239	247	236	228	220	146	1
• *Strip Ratio*	*W:O*	8.87	6.97	11.24	10.53	12.74	11.72	15.43	10.95	8.64	7.89	8.37	4.32	1.46	1.29	0.35
Processing																
• Ore Processed	Mt (db)	49.81		3.27	4.00	4.10	4.01	4.02	4.01	4.00	4.00	4.01	4.00	4.00	3.85	2.55
• Ore Grade	g/t	1.69		2.01	1.64	1.52	2.05	1.75	1.85	1.76	1.80	1.91	1.81	1.75	1.32	0.44
• Contained Gold	koz	2714		211	211	200	264	226	238	226	232	247	233	226	163	36
• Recovery	%	95%		95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	94%
• Recovered Gold	koz	2584		201	201	190	251	216	227	215	221	235	222	215	155	34

Table 22-3: Economic Model Summary

	Unit	Total/Avg.	Pre-Prod	Y-1	Y-2	Y-3	Y-4	Y-5	Y-6	Y-7	Y-8	Y-9	Y-10	Y-11	Y-12	Y-13	Y-14
Operating Cash Flow Summary																	
Gold Revenue (A)	USD M	**3875**		302	302	285	377	323	340	322	331	353	333	322	233	51	
• Mining & Rehandling	USD M	**(1263)**		(106)	(129)	(132)	(141)	(144)	(142)	(135)	(123)	(94)	(60)	(33)	(23)		
• Processing	USDM	**(577)**		(36)	(45)	(48)	(47)	(47)	(47)	(47)	(47)	(47)	(47)	(47)	(45)	(30)	
• General & Administrative	USD M	**(235)**		(16)	(19)	(19)	(19)	(19)	(19)	(19)	(19)	(19)	(19)	(19)	(19)	(14)	
• Inventory Adjustments & Other	USD M	**212**		29	38	20	40	80	33	3	35	2	(26)	(20)	(14)	(8)	
Subtotal: Total Cash Cost (B)	USD M	**(1863)**		(129)	(154)	(178)	(166)	(130)	(174)	(198)	(153)	(158)	(151)	(119)	(100)	(52)	
Subtotal: Total Cash Cost	USD/oz	**721**		643	768	936	662	601	768	920	695	672	681	554	643	1522	
• Sustaining Capital	USD M	**(386)**		(49)	(39)	(38)	(47)	(75)	(53)	(19)	(46)	(15)	(4)	(1)	(1)		
Subtotal: All-In-Sustaining Costs (C)	USD M	**(2249)**		(179)	(194)	(216)	(214)	(205)	(227)	(216)	(200)	(173)	(155)	(120)	(101)	(52)	
Subtotal: All-In-Sustaining Costs	USD/oz	**871**		887	963	1134	851	949	1001	1007	904	734	700	558	648	1523	
Sustaining Margin (A-C)	USD M	**1625**		123	108	70	163	119	113	106	131	180	178	202	132	(1)	
• Working Capital Movement	USD M		(39)	33	(13)	10	7	3	(3)	(7)	(7)	8	6	4	(1)		
• Taxes	USD M	**(287)**				(20)	(3)	(18)	(32)	(27)	(13)	(28)	(36)	(30)	(43)	(38)	
FCF Before Non-Sustaining Capital	USD M	**1338**	(39)	156	95	59	168	104	78	72	112	160	147	176	88	(39)	
• Non-Sustaining Capital	USD M	**(101)**		(6)		(10)	(25)	(31)			(7)	(22)					
• Closure costs (incl. Bond payments)	USD M	**(24)**				(0.3)	(0.3)	(0.3)	(0.3)	(0.3)	(0.3)	(0.3)	(0.3)	(0.3)	(0.3)	(4.0)	(18)
• Growth Capital	USD M	**(409)**	(409)														
Mine Free Cash Flow	**USD M**	**803**	**(448)**	**150**	**95**	**49**	**142**	**73**	**78**	**72**	**105**	**138**	**147**	**176**	**88**	**(43)**	**(18)**

Table 22-4: After-Tax NPVs at a Gold Price of USD 1500/oz

After-Tax NPV	USD (M)
After-tax NPV 0%	803
After-tax NPV 5%	477

22.3.2 All in Sustaining Costs (AISC)

The LoM All in Sustaining Costs (AISC) are forecast to come to USD 2249 M in total, or USD 871/oz. The breakdown by business area and year is presented in Figure 22-1 following.

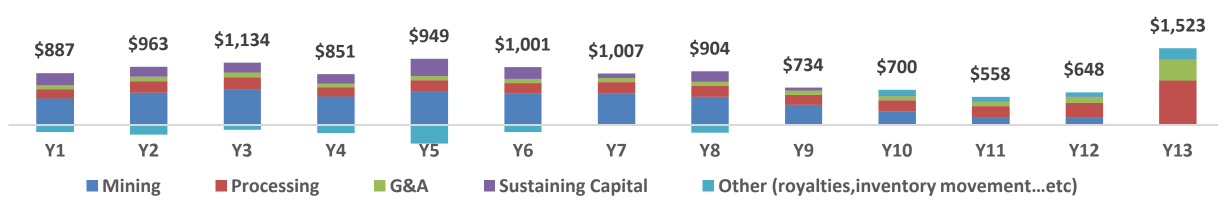

Figure 22-1: AISC Breakdown by Year

22.4 Sensitivity Analysis

22.4.1 Overview

A sensitivity analysis was performed by flexing a number of key variables including gold price, head grade, CAPEX and OPEX per cent change, to assess the impact on the after-tax NPV5% on a 100% basis. These were assessed independently whilst holding all other assumptions consistent to the base case presented.

The impact of gold price on Project NPV is illustrated in Table 22-5 following. In reviewing the data presented, consideration should be given to the forward forecast pricing of gold by year, and the LTP in real terms (Section 19). From Section 19, it can be seen that based on consensus pricing forecasts, and the three-year moving daily average gold price, the use of USD 1500/oz is considered relatively conservative.

Further, the Project business plan presented has been optimised using a USD 1300/oz gold price for reserves, and in a scenario where the gold price could be depressed for a prolonged period, i.e., less than the USD 1500/oz (base case), it is likely the business plan would be re-optimised in order to maximise economic value.

Table 22-5: Gold Price Sensitivity on Post-Tax NPV

Parameter	-20%	-10%	0%	+10%	+20%
Gold Price USD/oz	1200	1350	1500	1650	1800
NPV5% (USD M)	65	267	477	662	870

A sensitivity analysis (-20 to +20%) has been applied independently to a number of key operating factors to assess the impact that changes in CAPEX (excluding waste capitalisation), OPEX and gold grade, would have on the after-tax NPV5% (post tax) on a 100% basis, see Table 22-6 and Figure 22-2 following.

Table 22-6: NPV5% Sensitivity Analysis (USD M) After Tax (USD 1500/oz gold price)

Parameter	-20%	-10%	0%	+10%	+20%
Head Grade	57	267	477	686	895
CAPEX (excl. waste capitalisation)	548	512	477	441	405
OPEX	720	598	477	355	233

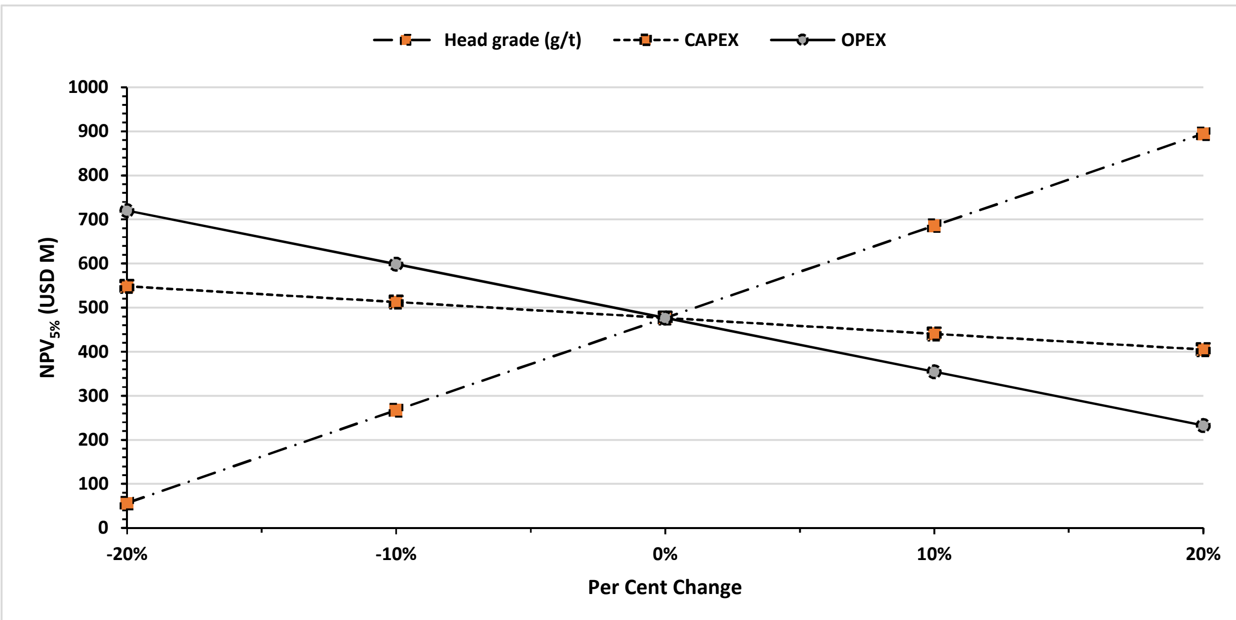

Figure 22-2: Financial Sensitivity Analysis (Post Tax)

The sensitivity analysis is in line with expectations, with the analysis showing that the Project has a relatively low sensitivity to capital and operating costs but is highly sensitive to movements in the LoM head grade. With respect to the latter, gold price and LoM head grade are largely correlated with respect to NPV impact.

22.5 Comparison with Historical Studies

A direct comparison with the historical prefeasibility study for the Project (Lycopodium, 2021) is difficult, given the change in study basis. Notwithstanding this, key parameters for the PFS and DFS are presented in Table 22-7.

Table 22-7: Financial Comparison Between Historic Studies

Metrics	Units	PFS	DFS
Estimate Base Date:		Q4 2020	Q2 2022
Estimate Accuracy Provision	%	±25	-5 to +15
Plant Capacity (LoM Blend)	Mt/a (db)	3.04	4.0
LoM Au Grade		2.05	1.69
Strip Ratio	w:o	10.34:1	8.87:1
Fuel Price	USD/L	0.90	0.91[147]
Power Price	USD/kWh	0.090[148]	0.112
Gold Recovery	%	94.6	95
Gold Produced (LoM)	Moz	1.985	2.584
OPEX (LoM avg.)	USD M/a	149	143
CAPEX	USD M	338	448
Internal rate of return (IRR) Post Tax	%	38	21
Net Asset Value (NAV) @ 5% discount rate (pre-tax)	USD M	663	664
Net Asset Value (NAV) @ 5% discount rate (post tax)	USD M	479	477
Payback Period (pre-tax)	Years	2.58	3.93
Payback Period (post-tax)	Years	2.66	4.17

Key differences between the financial results for the PFS and DFS is largely driven by the change in gold grade between the two studies, which was driven by a review and update of minerals reserves by SRK in 2022.

22.6 Independent Audits/Reviews

No independent reviews/audits have been undertaken on the financial results presented herein and in the underlying financial model.

22.7 Comments on Section 22

Taking into consideration the financial model 'basis/assumptions' outlined in this Report, the QP for Section 22 considers the economic data presented and underlying model to be valid, within the limits of the DFS accuracy provisions. See also comments within Section 3, 'Reliance on Other experts'.

22.8 Interpretations and Conclusions

Interpretations, conclusions and risks for Section 22 are discussed in Section 25 of this Report.

22.9 Recommendations

Recommendations/forward work programme activities for Section 22, are presented in Section 26 of this Report.

[147] USD 1/L in construction phase includes (duties)
[148] Hybrid grid/solar solution

22.10 References

References cited in the preparation of Section 22 are presented in Section 27 of this Report.

23. ADJACENT PROPERTIES

All of the information contained in this section is derived from sources available in the public domain.

Properties adjacent to the Lafigué Project (defined as within a 50 km radius of the centre of PR 329), comprise a series of artisanal mining or semi-industrial[149] claims, and eleven Exploration Licences (PR, Permis de Recherche). Permits and artisanal mining activities are illustrated in Figure 23-1, with the holders and status of each Permit detailed below (Cote d'Ivoire mining cadastre portal, 2022).

- Eburnea Gold Resources (PR 575, granted in 2020), including the Bouaké North Project developed by Turaco Gold as part of the global Eburnea Project.
- Resolute Cote d'Ivoire SARL (Resolute) (PR 544, granted in 2016), including the Satama Project developed by Turaco Gold as part of the global Eburnea Project.
- Managem CI SA (PR 671 and 680 granted in 2017, currently being renewed).
- Yam's Mining (PR 870 and 892, granted in 2020 and 2021, respectively).
- Sodinaf-CI (PR 337 and 338, granted in 2013, currently being renewed).
- Sodemi (PR 860, granted in 2020).
- International Goldfields (PR 426, granted in 2014).
- XMI SARL (PR 573, granted in 2015).

With respect to the aforementioned permits, public domain information relating to geological and exploration data is limited to the PR 575 and PR 544 permits only, which are being developed by the ASX-listed Turaco Gold Group.

[149] As per the 2014 Mining Code, the Lafigué Mine falls within the semi-industrial category

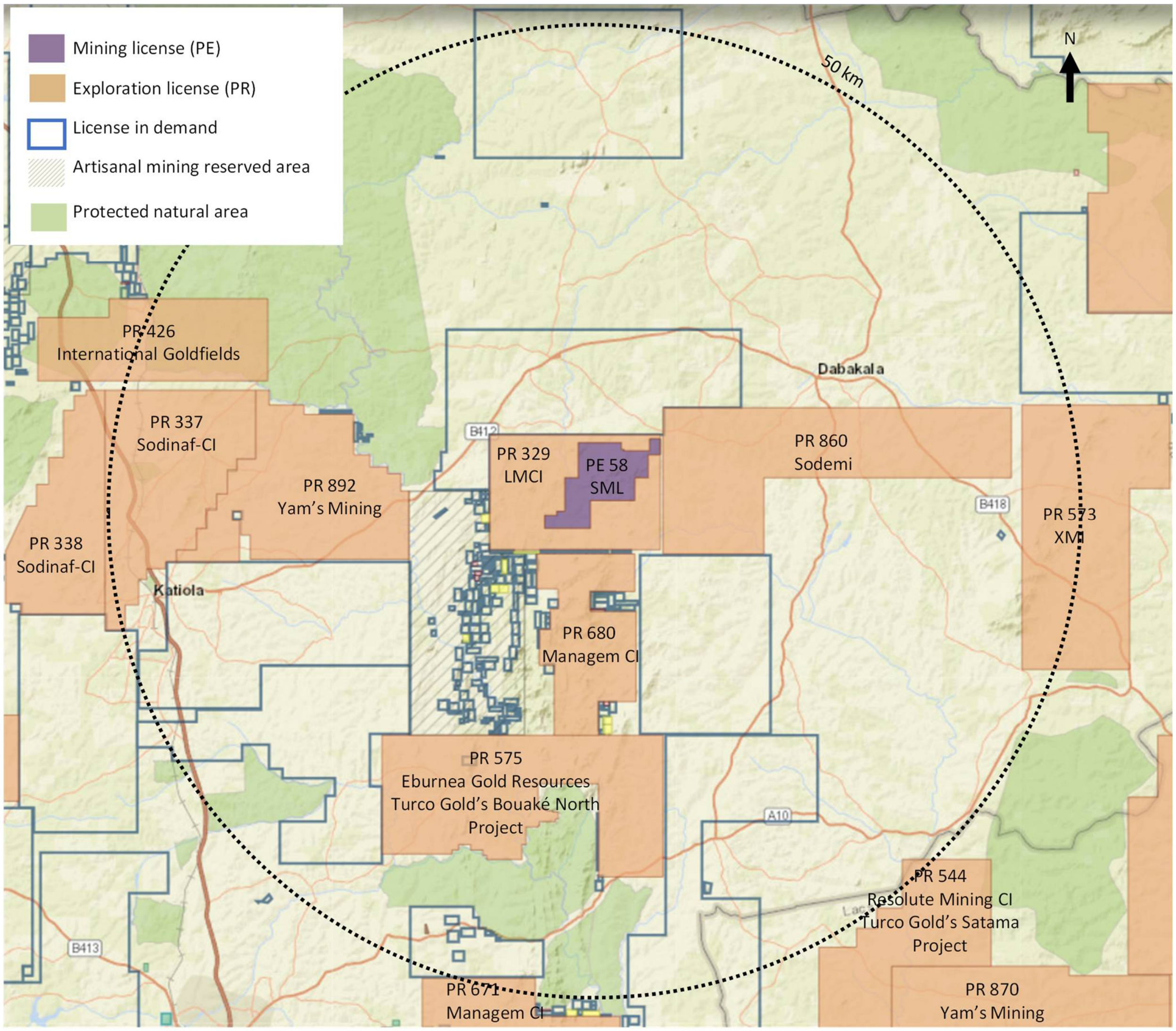

Figure 23-1: Location of the Adjacent Permits and Artisanal Mining Activities (Trimble, 2022)

23.1 Eburnea Gold Resources, property (PR 575)

PR 575 comprises the Bouaké North Project, which is currently being developed by Turaco Gold, an ASX listed company (Turaco) which holds a contractual right to an 80% interest in the Permit (with a right to acquire a further 10% interest) in joint-venture with local partner, Eburnea Gold Resources. The Bouaké North Project is one of the two components of Turaco's 690 km^2 Eburnea Project (Figure 23-2) which also includes the Satama project (Section 23.2).

The Bouake North Project is positioned on the Oume-Fetekro greenstone belt, along the margin of the Birimian Comoé basin, approximately 35 km south of the Lafigué deposit. The geology in the area comprises of porphyritic dykes intruding fine grained volcano-sediments, with gold mineralisation associated with zones of quartz veining close to dyke margins.

The permit area includes numerous soil geochemistry anomalies covering a 7 x 4 km area, initially highlighted in 1994 and further confirmed by Turaco in 2020 and 2021 through additional soil sampling (1600 samples collected on a 500 x 500 m grid and systematic auger drilling on a 25 x 200 m grid at 3 to 10 m depths. The auger drilling delineated six saprolite +100 ppb gold targets each extending for more than 1000 m of strike (Figure 23-3). Follow-up shallow RC/AC drilling by Turaco in 2022 tested two of the six anomalies (Figure 23-3 and Figure 23-4), with some of the best intercepts including:

- 3 m @ 35.79 g/t Au from 40 m depth (BNRC008);
- 8 m @ 1.44 g/t Au from 56 m depth (BNRC004);
- 12 m @ 1.38 g/t Au from 8 m depth (BNAC0147);
- 13 m @ 1.05 g/t Au from 3 m, incl. 1 m @ 11.49 g/t Au from 3 m depth (BNAC0144);
- 7 m @ 1.82 g/t gold from 0 m, incl. 4 m @ 3.07 g/t Au from 3 m depth (BNAC0115); and
- 2 m @ 3.94 g/t gold from 27 m depth (BNAC0037).

Further results are pending from the remainder of the Air Core Drilling (AC) programme. Turaco plans additional drilling, trenching and geochemistry on the permit.

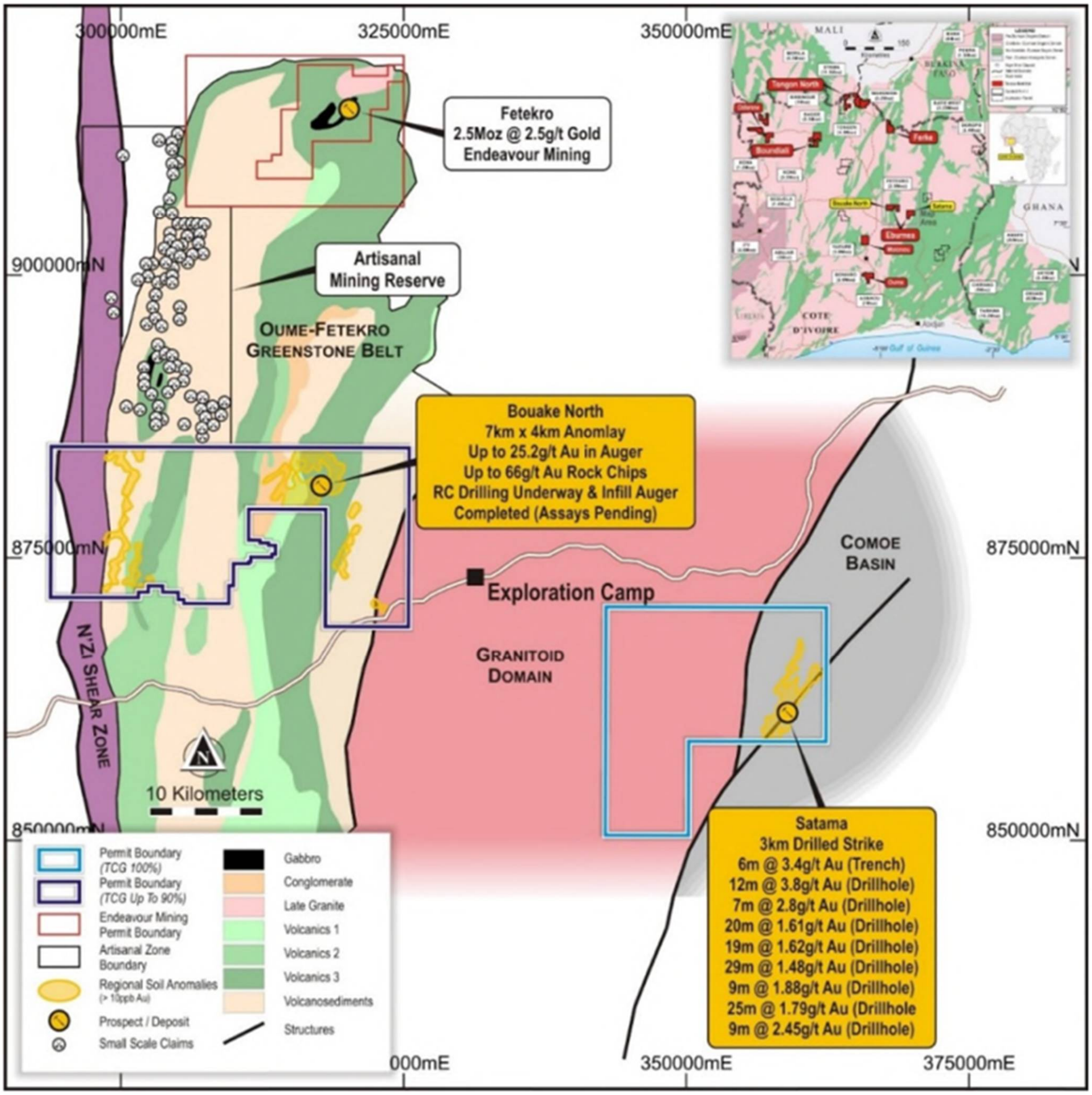

Figure 23-2: Turaco Gold's Eburnea Project location and geology (Turaco Gold, 2022a)

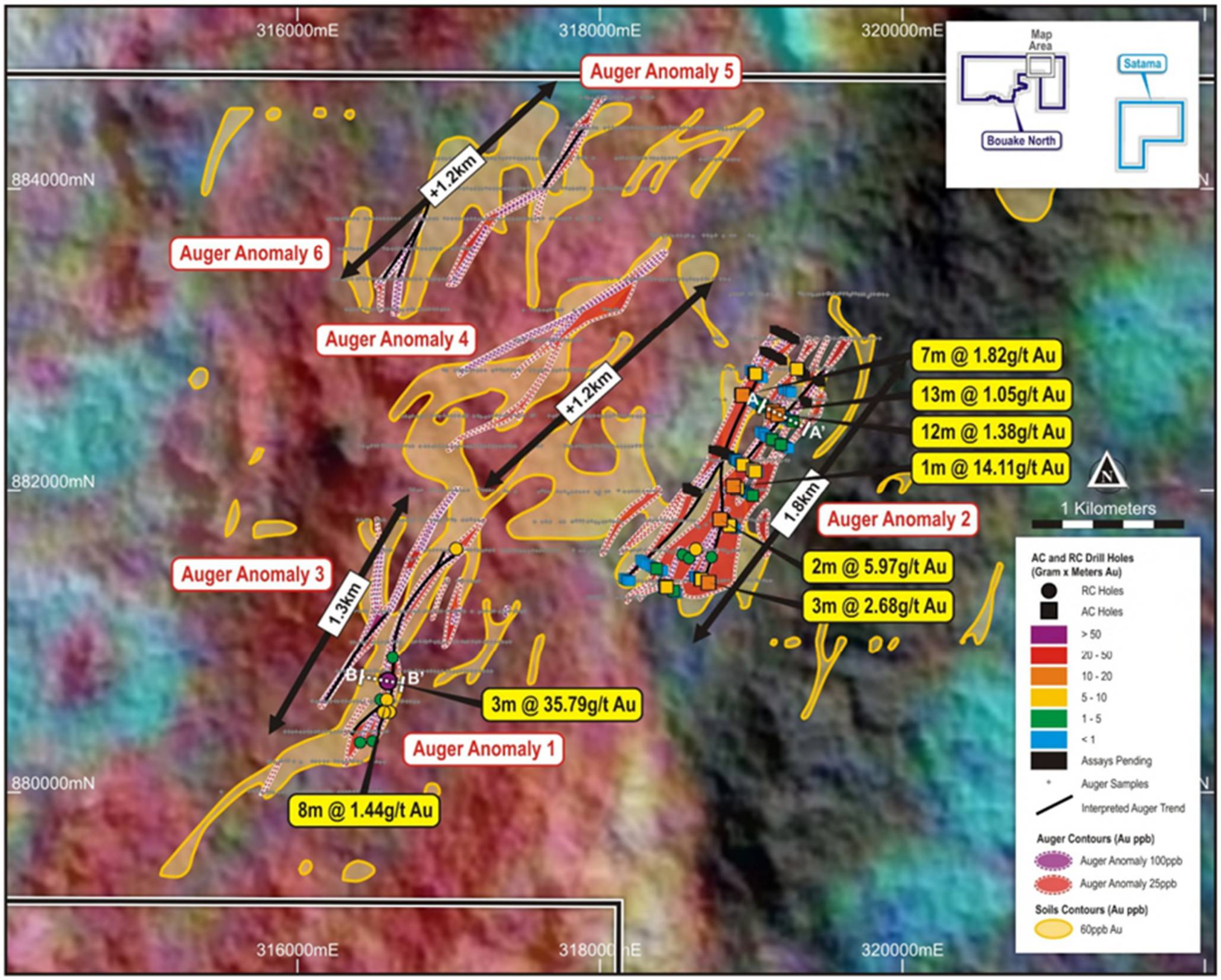

Figure 23-3: Bouaké North Project: Drillhole Collars with Gold-in-Soil and Auger Anomalies over Radiometrics (Turaco Gold, 2022a)

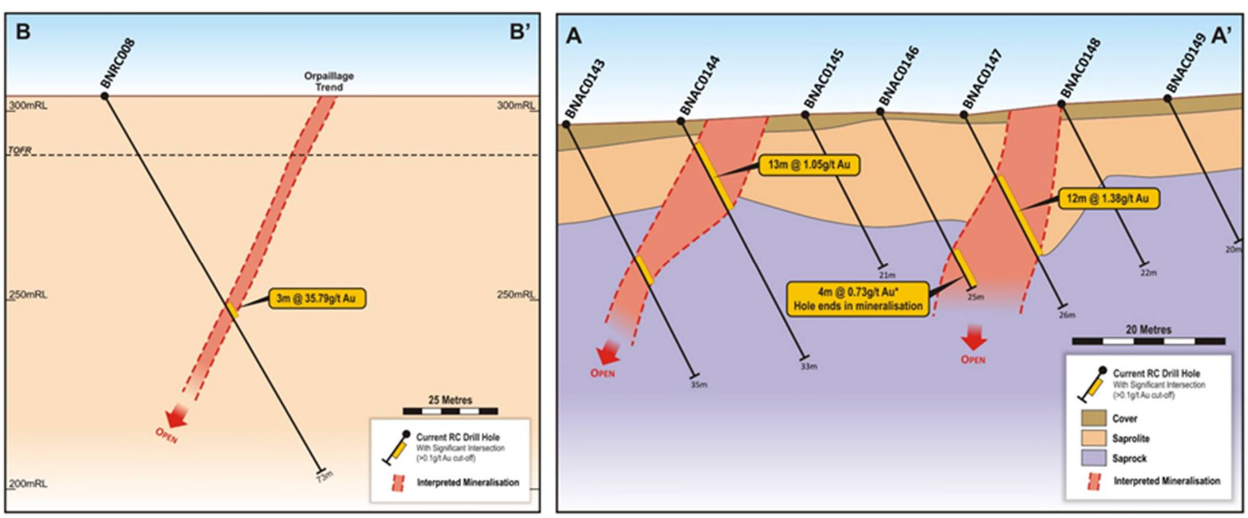

Figure 23-4: Geological sections for Bouaké North Project (Turaco Gold, 2022a)

23.2 Resolute Cote d'Ivoire SARL (PR 544)

PR 544 comprises the Satama Project which is currently being developed by Resolute, a wholly-owned subsidiary of the ASX-listed Turaco Gold company. The Satama Project is the second of the two permits of comprising Turaco's 690 km^2 Eburnea Project (alongside Bouaké North) (Figure 23-2).

PR 544 is located approximately 50 km southeast of the Lafigué deposit. The permit area includes a northeast-trending shear splaying off the crustal-scale Ouango-Fitini shear, which marks the margin of the Birimian Comoé basin (Figure 23-2).

An initial programme of auger drillings was completed by Turaco in 2021, which tested soil geochemical anomalies previously defined by Resolute (773 holes for 5660 m, based on 250 m-spaced traverses with auger spacing of 25 to 50 m). The results confirmed an anomaly over a 3.5 km strike length (Figure 23-5) with a central 2.5 km of strike returning high grades across a width of up to 600 m (with a best result of 9 m @ 4.49 g/t gold from 1 m) (Turaco Gold, 2022a)

Follow-up AC drilling in early 2022 (7226m with broad 250 to 300 m drill traverses), returned consistent oxide mineralisation across 3 km of strike length, which remains open to the northeast. In addition, a second trend striking north-northeast which remains open for at least 1.5 km along strike to a trench that returned 6 m at 3.36 g/t gold has been identified. Turaco commenced an RC drilling program in March 2022 (planned 4500m) to reduce the drill traverse spacing down to a nominal 160 m and to test downdip mineralisation continuity to vertical depths of around 120m beneath the weathered zone (Figure 23-5).

As of June 2022, over 35 RC holes have been drilled and results have been received for the first 15 holes (2125m) (Figure 23-5 and Figure 23-6 Turaco Gold, 2022b). Best RC drilling intercepts include:

- 25 m @ 1.79 g/t gold from 101 m depth, including, 3 m @ 6.40 g/t gold from 109 m (STRC0030);
- 9 m @ 2.45 g/t gold from 18 m depth (STRC0026);
- 6 m @ 1.80 g/t gold from 72 m depth (STRC0022);
- 6 m @ 1.80 g/t gold from 132 m depth (STRC0015); and
- 21 m @ 0.92 g/t gold from 116 m depth (STRC0020).

The RC drilling programme indicates that oxidation extends to an average depth of 80 m, where fresh rock is encountered. Mineralisation is hosted in a strongly carbonate-silica altered fine-grained sandstone. Sulphides, dominantly pyrite, are disseminated and associated with quartz veinlets.

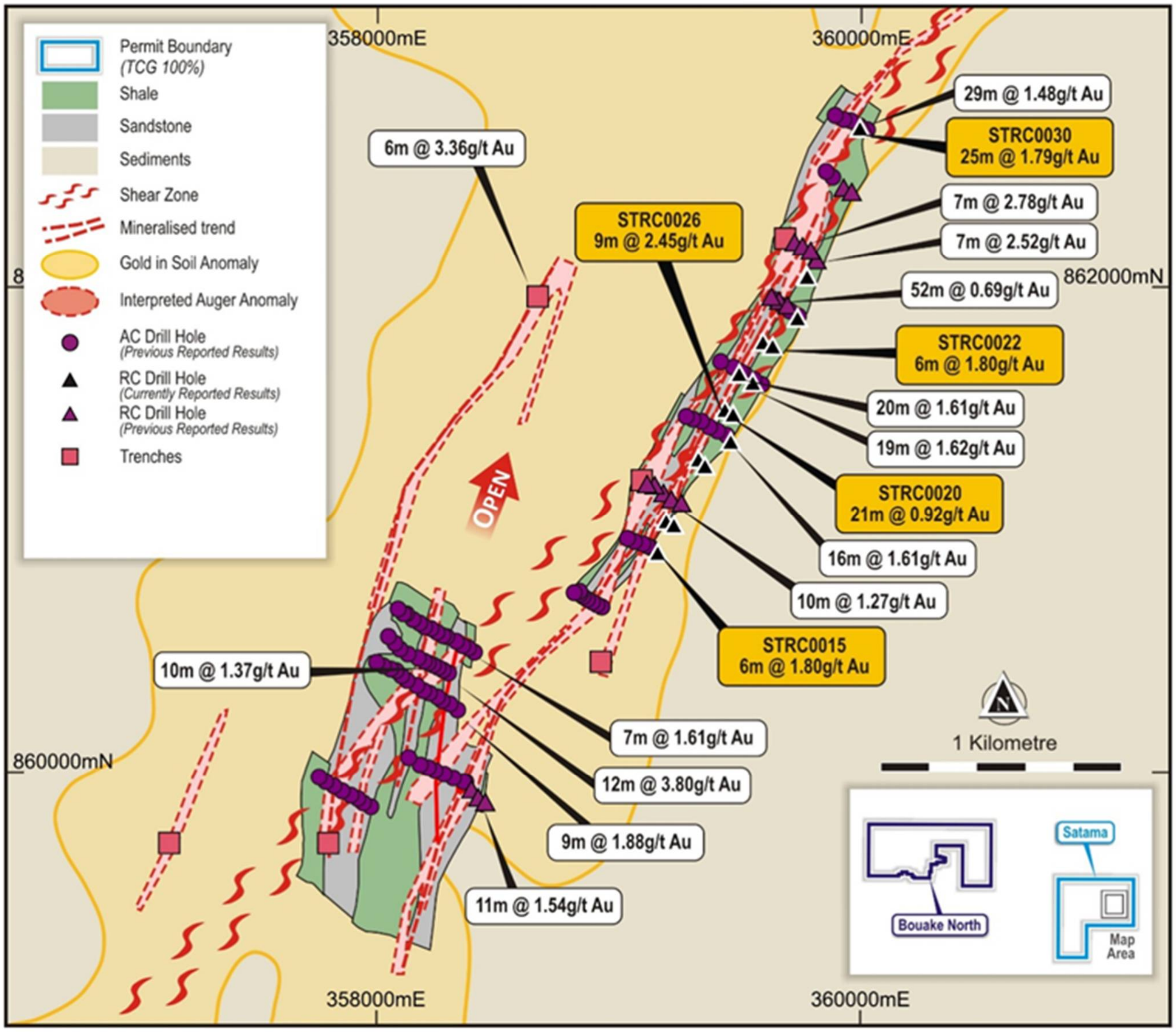

Figure 23-5: Satama Project AC and RC drilling plan with Gold-in-Soil and Auger Anomalies (Turaco Gold, 2022b)

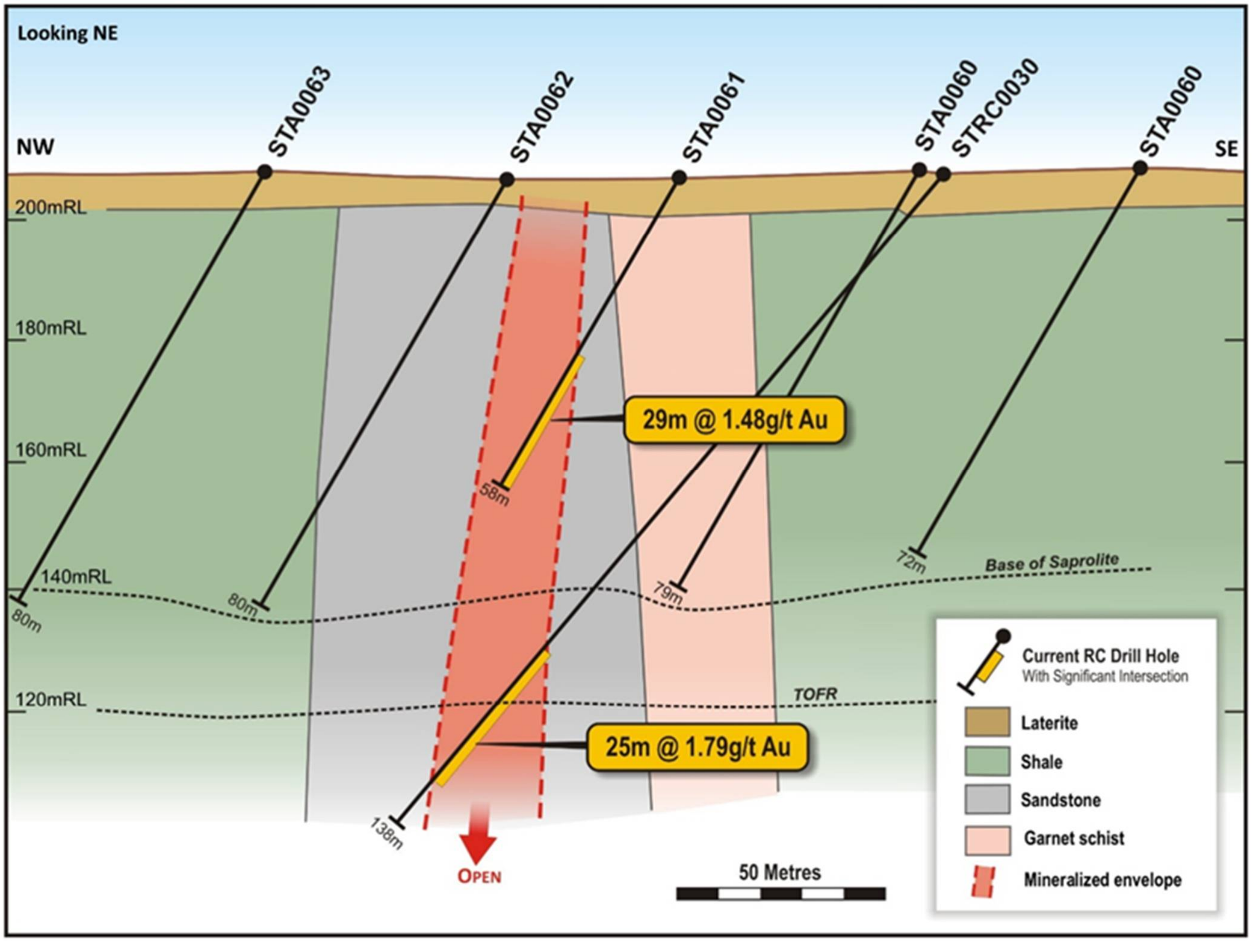

Figure 23-6: Geological Sections across Satama Project (Turaco Gold, 2022b)

23.3 Data Verification

The data verification process applied with respect to the presentation of data in Section 23, is detailed in Section 12.

23.4 Comments on Section 23

Based on limited publicly available information, several Exploration License areas adjacent to the PR 329 EL host gold mineralization in broadly analogous geological terranes. It remains unclear whether any such occurrences will be developed into publicly reported Mineral Resources.

23.5 Interpretation and Conclusions

Interpretations, conclusions and risks associated with Section 23, are presented in Section 25 of this Report.

23.6 Recommendation

Recommendations pertaining to Section 23, are presented in Section 26 of this Report.

23.7 References

References cited in the preparation of Section 23, are detailed more fully in Section 27 of this Report.

24. OTHER RELEVANT DATA AND INFORMATION

24.1 Human Resources

24.1.1 Overview

Section 24.1 as outlined herein, provides a brief overview of the CI labour legal framework, the labour operating basis for the mine, the recruitment and training strategy, labour sourcing, organisation support structures, stakeholders, and labour numbers by business area/department.

24.1.2 Legal Framework

Article 1 of CI Decree no. 96-203 of March 7, 1996, specifies the application of the legal weekly working hours by stipulating that; 'Subject to the rules relating to equivalence, the recovery of hours collectively lost, overtime and permanent or temporary derogations provided for in Articles 13 and 14, the weekly working time may not exceed:

- forty hours per week, for non-agricultural companies; and
- forty-eight hours per week for farms, establishments, agricultural enterprises, and similar enterprises, up to a maximum of two thousand four hundred hours per year'.

Article 5 of the same Decree, which details the daily distribution of working hours, states that; 'Subject to the implications arising from equivalence, the employer shall determine the daily working hours applicable in the ; farm establishment or enterprise, according to one of the following methods of distribution:

- limitation of actual work to eight hours per day, for five working days of the week;
- limitation of the actual work to six hours and forty minutes per working day of the week; and
- unequal distribution among the working days of the forty hours per week, with a maximum of eight hours per day'.

Further, Article 26 of Decree No. 96-203 of March 7, 1996, on Working Hours, limits overtime to 75 hours per year per worker.

In consideration of the above, SML may consider setting up a work roster system which exceeds the work week in favour of paragraph 1 of Article 11 of Decree No. 96-203 of March 7, 1996, Relative to the Duration of Work which states in extension: 'The work method of the company, establishment or operation may be organized, according to a system of rotation of personnel, in the form of a work cycle, the duration of which exceeds the week'.

24.1.3 Mine Operations - Working Hours, Rosters and Roster Basis

The mine operates 24 h/d, 365 d/a; thus, for a number of roles, 24 h/d coverage is required. In accordance with Section 24.1.2, this is typically met with either three 8 h shifts per day, or two 12 h shifts per day. Working hours and the number of days per shift cycle are largely governed by:

- the CI labour law governing working hours per week (40 h), and overtime hours allowed per year (75 h/a); and
- the implications on worker health, such as; sleeping conditions, the physical location of employees homes, and the time spent travelling to and from work.

Nationals and expats not living close to the mine (distance to be defined) will be employed on single status basis.

Rosters and leave basis are as noted below.

- Residential Basis
  - Nationals: day work (Local): 5 days on/2 days off (8 h/d, 5 d/week), 4 weeks annual leave per annum.
- Non-residential basis[150]:
  - Nationals (Site - Cadre): 4 weeks on/2 weeks off (10 h/d, 6 d/week), 4 weeks annual leave per annum
  - Nationals (Site - Agent de Maîtrise/Managers): 4 weeks on/2 weeks off, 4 weeks annual leave per annum.
  - Nationals (Site – Ouvrier/Employé): 4 days on/ 4 days off (12 h/d), 4 weeks annual leave per annum.
  - Expatriates: 6 weeks on/3 weeks off (10 h/d, 6 d/week), 4 weeks annual leave per annum.

Workers have a legal break of at least 30 minutes per shift; that is, an effective working time of 7 hours 30 minutes or 11 hours 30 minutes, for 8 and 12 hour shifts respectively. In all cases, the 8 or 12 hours that the shift lasts are fully paid; the break being considered as working time.

Travel time from home to the Site, and time spent in the change room, are not taken into account in the calculation of the 8 or 12 hours paid per shift, which only begins when the worker arrives for the shift, and ends when the worker leaves his/her place of work for the change room.

Whilst still being reviewed/refined, the shift panel system indicated in Table 24-1 and Table 24-2 are seen as likely options for moving forward. Whilst there are benefits of having one shift system for the mine, it is not a given that all contractors providing a service to the mine will operate on the same shift basis.

Table 24-1: 12 h/d, 4 Panel Shift System

Panel	Day 1 to 7							Days 8 to 14							Days 15 to 21							Days 22 to 28						
P1	D	D	D	D	O	O	O	O	N	N	N	N	O	O	O	O	D	D	D	D	O	O	O	O	N	N	N	N
P2	N	N	N	N	O	O	O	O	D	D	D	D	O	O	O	O	N	N	N	N	O	O	O	O	D	D	D	D
P3	O	O	O	O	D	D	D	D	O	O	O	O	N	N	N	N	O	O	O	O	D	D	D	D	O	O	O	O
P4	O	O	O	O	N	N	N	N	O	O	O	O	D	D	D	D	O	O	O	O	N	N	N	N	O	O	O	O

Table 24-1 notes: D = Day, N = Night, O = Off

[150] Rotations for nationals subject to review/change.

Table 24-2: 8 h/d, 4 Panel Shift System

Panel	Day 1 to 7							Days 8 to 14							Days 15 to 21							Days 22 to 28						
P1	D	D	A	A	N	N	O	O	D	D	A	A	N	N	O	O	D	D	A	A	N	N	O	O	D	D	A	A
P2	O	O	D	D	A	A	N	N	O	O	D	D	A	A	N	N	O	O	D	D	A	A	N	N	O	O	D	D
P3	N	N	O	O	D	D	A	A	N	N	O	O	D	D	A	A	N	N	O	O	D	D	A	A	N	N	O	O
P4	A	A	N	N	O	O	D	D	A	A	N	N	O	O	D	D	A	A	N	N	O	O	D	D	A	A	N	N
Table 24-1 notes: D = Day, A = Afternoon, N = Night, O = Off																												

24.1.4 Meals

All persons not living in the camp will be provided one meal per day, for both 8 and 12 hour shifts. Camp personnel will be eligible for three meals per day.

24.1.5 Recruitment and On-boarding Process

The Project/Mine will require a large number of people for key positions over a short period of time, both for construction and operations. Where possible, people will be moved from construction into operations.

To ensure that all recruits are adequately qualified and have the required skillsets for their respective roles, a verification of competencies process will be followed as part of the recruitment process.

Whether directly or indirectly employed, all persons working on the Project/Mine will have:

- pre-employment medical examinations (a legal and mandatory obligation); and
- insurance for health coverage and life insurance.

A welcome manual will be prepared and will be given to all staff and new employees. This booklet will be the reference for the Human Resources (HR) standard operating practises (SOPs), and will cover:

- working hours;
- disciplinary/sanction procedure;
- absence procedure;
- leave procedure;
- contract;
- payroll;
- workplace accident reports; and
- health and safety at work.

The onboarding process will aim to instil the Endeavour/SML culture and ensure that it is embedded across the Site, irrespective of the employer.

The HR team will work with the Social Performance Department (SPD) on the employment/development of:

- local unskilled community members; and
- local woman (integral part of Group policy).

To assist the HR team in the recruitment process, additional resources will be hired internally and externally, namely:

- Internally: the project HR team will be assisted by the Group's recruitment team on permanent senior/management positions, who will be appointed during the construction/operational phases.
- Externally: the selected subcontractors (TECTRA CI[151] and Aldelia Global Manpower[152]) for construction, will be responsible for providing the temporary staff required in compliance with local labour laws. One or more of these companies may be taken through to operations. Further to assist in training, a third party service provider will be employed during the operations phase.

24.1.5.1 Workforce Composition - Nationalisation and Gender

The organisational targets for different population demographics are:

- Unskilled/low skilled: 100% Nationals.
- Skilled/Technical/Professional/Management: >90% Nationals.
- Woman in the workplace: 25%.

24.1.5.2 Workforce - Local Recruitment Pools

The Hambol region is largely agrarian in nature, with its economy mainly based on agriculture and livestock rearing, with some artisanal mining in the areas surrounding the mine (see Section 5). In the Hambol region, there are no other gold mines or similar industrial facilities that could provide some of the higher level skills required to operate a mine.

Where possible, employment priority will be given to suitably qualified and able candidates from the local population. Nearby population centres comprise the villages and towns noted in Table 24-3. These data should be used as guidance only, as the data source has not been stated/verified.[153]

A survey has not been undertaken yet to identify: area age/sex demographics, literacy levels, the relevant labour skills/institutional capacity locally and regionally to support the development/operation of the mine (see Recommendations, Section 24.5.). Country age demographics and literacy rates are presented in Section 5 as a guideline.

Table 24-3: Local Villages and Towns

Village Name	Population	Distance from Site
Lafigué	1304	2 km
Sokorhogo	1425	8 km
Oualeguera	880	9 km
Fenessedougou	625	11 km
Kaniemene	1155	15 km
Koundodougou	2150	15 km

151 www.tectra.ci

152 www.aldelia.com

153 Likely for the RGPH 2014 general census survey, see www.gouv.ci

Table 24-3: Local Villages and Towns

Village Name	Population	Distance from Site
Lognene	1080	18 km
Toledougou	951	19 km
Bounadougou	1359	21 km
Boniérédougou	28 104	22 km
Dabakala	67 638	35 km
Katiola	68 470	72 km

24.1.6 Regional & Corporate Functions

The Lafigué Mine, as with other Endeavour mines, will be supported by Endeavour's head office in London and a regional office in Abidjan in CI. In general, the mine functional areas noted below are replicated at a corporate/regional level, and where appropriate:

- certain Site mine functions will be supported solely at regional/corporate level; and
- SML personnel may be permanently based in Abidjan and support the site operations remotely.

Mine functional business areas/departments include:

- HR.
- Social Performance.
- Finance.
- Supply Chain (especially logistics, customs clearance and freight forwarding).
- Government Relations, Public Affairs and Communications.
- Safety Health and Environment (SHE).
- Security.
- Information and Communication Technology (ICT).
- Exploration.
- Mining.
- Technical Services.
- Process, Engineering and Projects.

24.1.7 Organisational Structure

The general organisational structure for mine operations has five levels of practise, namely:

- General Manager;
- Department Manager (Head of Department);
- Superintendent/Coordinator;
- Supervisor and Senior Supervisor; and
- Officer/Assistant/Tradesperson.

24.1.8 Training

Endeavour at a corporate level has a Training and Development Programme that provides both guidance and the necessary tools to optimise the skillset of new employees.

No training department has been provided at Site, on the basis that the HR group will work with the functional department heads to conduct a skills GAP analysis for the required roles. Training programmes will subsequently be developed/tailored accordingly. Further, 5% of the total cost to company (TCTC) for Nationals is allocated to the training budget and is reported as a 'General and Administration' cost.

Multi-skilling has been identified as an opportunity for increasing efficiencies and reducing numbers of personnel at the mine. To realise this opportunity, tasks/roles that can be multi-skilled will be identified within each department. A training matrix for each role will be developed.

Training facilities requirements will be defined in line with the training and learning programme.

To assist the HR team to optimise the skillset of the employees and identify and implement multi-skilling within the organisation, a third-party consultant will be engaged to aid and fast track this process.

In addition, a training section will be developed on site in collaboration with the SHE department for practical and technical training to ensure that personnel have the required skills to meet their role requirements.

24.1.9 Stakeholders

Key labour stakeholders are:

- Labour Inspector: to ensure mine compliance with in-country labour laws.
- Union: to act as an intermediary between staff and HR and play an important role in maintaining a good social climate. Union representatives will be selected by the mine workers.
- Social Security[154]: for the declaration of work accidents and professional illnesses/care of pregnant women/payment of family allowances/retirement plan.
- Insurance providers: in case of a disaster/health and life insurance.
- Schools of Excellence of the Mines: to establish internship agreements to promote the mine and develop the future skills of young graduates.

24.1.10 Labour Numbers

The Lafigue Mine will employ a mix of Owner's team personnel and contractors. Personnel numbers by business area/department/functional requirement, are as defined in Table 24-4. Importantly:

- Where half a person has been allocated to a Mine functional role (7 positions in total) in the Lycopodium Labour Model (Lycopodium, 2022a), this has been considered for now, a shared corporate service (50:50), with no Site presence. Labour numbers for this shared service are not presented within the 'Total' in Table 24-4. Costs for the shared service, however, are covered within the labour OPEX estimate.
- Labour numbers for 'Mining Contractor and Ancillaries' are based on updated labour numbers received from the preferred mining contractor.

[154] See also: www.ssa.gov/policy/docs/progdesc/ssptw/2018-2019/africa/cote-divoire.html.

- Labour numbers for 'Mining Emulsions and Blasting' are based on an internal Endeavour estimate and are based, in part, on labour numbers from Endeavour's Ity Mine.
- The mine will employ approximately 51 expatriates (Mining Contractor (35) and Owner's team (16)). This represents approximately 3% of the Site work force.
- No training department has been provided on the basis that: the training strategy and records keeping will be undertaken by the HR team; whilst the actual training is to be done by the various Mine business units/departments and an external training service provider(s).

Table 24-4: Mine Labour Numbers (Owner's Team and Outsourced)

Business Areas	Owner's Team (100% Positions)	Owner's Team (50% Positions)	Total Full Time (100%) Site Employees	Day Work	Number/Shift Panel	Maximum Persons on Site at any One Time
Administration	2	0	2	2		2
Human Resources	5	2	5	5		5
Camp & Travel	7	0	7	7		7
Training[155]	0	0	0			0
Information and Communications Technology	4	0	4	4		4
Social performance	8	2	8	8		8
Finance	15	0	15	15		15
Health Safety and	30	0	30	14	4	18
Fuel Supply		0	6	2	1	3
Supply chain	13	3	13			0
Mineral Resource Management	19	0	19	11	2	13
Mining	6	0	6	6	0	6
Mining Technical Services	22	0	22	18	1	19
Mining Contractor & Ancillaries		0	699	139	140	279
Mining - Emulsions		0	37	3	17	20
Process Plant	53	0	53	21	8	29
Plant/Inf. Maintenance	73	0	73	61	3	64
Camp, kitchen, laundry and maintenance		0	83	35	12	47
SML Security	28	0	28	16	3	19
Gendarmes	0	0	36	4	8	12
Security Contractor	0	0	372	44	82	126
Medical/Clinic	0	0	6	2	1	3
Laboratory	0	0	27	11	4	15
Totals	**285**	**7**	**1551**	**428**	**286**	**714**

[155] Training numbers for an external training service provider are not defined.

24.1.11 Indirect Employment

In a 2012 report by PricewaterhouseCoopers (PWC) for Mines in British Columbia, it was noted that for every person employed at a mine (owner's team and contractors) a further 0.8 indirect and 0.4 induced jobs were created. Thus, in a western country, a multiplier of 2.1 could be used to determine the total number of jobs created per mine (PWC, 2012). Cordes (Cordes, 2016) noted that Rio Tinto for their Simandou iron ore project (Guinea), assumed a multiplier of 6.3 to calculate the total number of jobs created (direct, indirect and induced). Other studies have noted a much higher level of induced employment in developing countries (Cordes, 2016).

Whilst SML is not bound by fixed legislative targets in CI with respect to: the employment of local tribal/religious/ethnic groups; Nationals; expatriates; woman and disabled persons, SML is committed to supporting and developing local communities and CI as a whole. Thus, there will be over the coming years, a drive to reduce the number of expatriates employed, empower women (25% employment target), upskill and employ local persons and grow local/regional procurement and by association, businesses.

24.1.12 Comments on Section 24.1

Labour costs are based on another Endeavour operating mine in CI. These costs will need to be updated once the shift rosters are defined. Whilst the labour numbers and costs presented are suitable for use in the DFS OPEX estimate, the level of technical/discipline development is at PFS/DFS level of development. This needs to be addressed in the forward workplan (see 'Recommendations', Section 24.5).

24.2 Project Implementation

24.2.1 Overview

The implementation approach proposed for the Project is for Endeavour/SML (the 'Owner') to engage a principal Engineering, Procurement and Construction Management (EPCM) contactor to provide: design, procurement, and construction management services for the execution of the process plant (the 'Plant') and selected infrastructure facilities, which will be handed over to the Owner's operating team on completion. The construction of the mine, tailings dam, water storage and harvest dams, incoming high voltage transmission line, 225 kV switchyard, camps, and non-process infrastructure buildings will be either self-performed by the Owner's team or by specialist consultants/contractors engaged directly by the Owner.

This project execution approach was used as the basis for the Preliminary Implementation Schedule (PIS) and the capital cost estimate developed for the DFS.

A comprehensive Project Execution Plan (PEP) incorporating a schedule and control budget will be developed for the Project, detailing the overall management methodology for the delivery of the Project, including engineering, procurement, construction, commissioning and handover.

The PEP will include strategies for all aspects of project management and control across all the Project functions and phases.

Overall responsibility for health and safety, scope, schedule, budget, and quality within the boundaries of the EPCM contract will rest with the EPCM Project Manager. The EPCM Project Manager will be supported at a corporate level by the EPCM Project Sponsor and peer review team, who will act as coordinators and advisors regarding the EPCM Contractor's corporate quality requirements.

24.2.2 Owner's Team

The Owner's team will be progressively expanded to widen its skills and knowledge base to meet the needs of the Project.

The Owner's team will manage both the in-country and offshore activities of the principal EPCM contractor and specialist subcontractors, as well as providing specialist technical input into the Project design.

Key onshore mine operations roles will be filled early to contribute to the mine design and manage the early mine development works on site.

An Owner's onsite management, administration and services department will manage environment and community issues and prepare the site for the coming influx of operating personnel.

The EPCM Contractor's offshore design, procurement and commissioning team will all report to the EPCM Project Manager. The EPCM management team comprising of a Design Manager, Lead Project Engineer, Contracts Engineer, Lead Procurement Officer and Health and Safety Manager will all report to the EPCM Project Manager. The EPCM Contractor's onshore construction team will report to the EPCM Construction Manager who will report directly to the EPCM Project Manager and indirectly to the Owner's Construction Manager.

24.2.3 Logistics

West Africa has a well-developed mining industry, serviced by a network of air, sea, and road routes. The construction/operational logistics needs of the Project are relatively modest on a regional/global mining scale and the port facilities at Abidjan have sufficient capacity and facilities to act as the main gateway for the importation of construction equipment and materials, plus ongoing operating consumables. Road links within the country are generally good with significant investment in roads in CI. Logistics routes and options are discussed in Section 5.

Whilst a preliminary construction and operations logistics survey has been undertaken (MOVIS CI, 2021), this will be updated over the course of the Front-End Engineering Design Phase (FEED).

24.2.4 Mine Development

Mining and associated services (drilling, explosives, emulsion and grade control) will be outsourced to third parties, who will provide the requisite physical infrastructure and provide a service to the Owner.

A Mining Services Area (MSA) has been located southwest of the Plant, with good access to the mine pit, waste dumps, and ROM pad from mine haul roads. The MSA will consist of mine support facilities and infrastructure such as offices, workshop, wash bay, re-fuelling bay and workforce facilities.

24.2.5 Plant and Surface Infrastructure

For the Plant and surface infrastructure, the implementation strategy is an EPCM approach where an EPCM Contractor (an internationally accredited EPCM company) will be responsible for managing all aspects of the design and procurement, field engineering, quality assurance and control, safety and commissioning under the broad direction of the Owner's team. The EPCM Contractor will also provide key supervisory roles for construction activities under the direction of an Owner's Construction Manager.

24.2.6 SHE Management Plan

In consultation with Owner, the EPCM Contractor will prepare a SHE Management Plan for the Project, referred to as the SHE Contractor, or SHE, Management Plan.

The SHEC Management Plan will be issued to all contractors tendering for site work as part of the enquiry document. Each contractor will be required to demonstrate a satisfactory prior commitment to safety and present a site-specific plan for their proposed involvement in the Project.

24.2.7 Design Management

An Engineering Plan will be developed for the Project defining the principles and execution guidelines to be adopted by the EPCM's design team during the design phase of the Project. It will also describe the handover of various engineering deliverables at procurement, tender, construction, commissioning, project close-out and handover stages.

The majority of design outputs are produced during the early works and detailed design process. These take the form of equipment specifications, datasheets, drawings, and purchase requisitions. Support documentation includes lists, material take-offs, calculations, check prints, vendor data, and field installation checklists for construction and commissioning.

The schedule has allowed for a period of early works prior to full funding approval to achieve as short as possible timeline for the overall project development. This period will concentrate on finalising the layout/process design criteria and the development of engineering deliverables relating to the tendering and award of early contracts including bulk earthworks, camp facilities and related infrastructure including the construction and permanent facilities. This period will also allow for the completion of the long lead procurement packages commenced in the DFS phase to allow a recommendation for award to be issued prior to funding approval.

24.2.8 Project Controls

Effective project controls are critical to the successful completion of a project providing relevant and consistent budget, costs, and schedule reporting to the Project team. This provides the tools to efficiently manage the Project at the level of detail necessary to meet project cost and schedule objectives.

The project controls requirements will be outlined in the PEP addressing cost control, planning, progress measurement, project reporting, asset capitalisation, and close-out. The scope of project controls is to provide a framework of the work processes, workflows, and information relating to the standard project controls and accounting interface tools, systems and procedures that will be utilised during the execution of the Project.

The capital cost estimate developed during the DFS will be used as the control budget for the Project.

Costs will be measured and reported by activity in accordance with the Project Work Breakdown Structure (WBS) developed during the DFS and contract award phases. This will serve to keep the Project informed on a timely basis regarding the status and risks associated with cost and time.

Monthly cost reports will be prepared to show the original budget, approved changes, revised budget and current forecast costs. Committed and incurred costs and paid expenditures will also be included in the report.

24.2.9 Project Schedule and Basis of Schedule

The implementation strategy is structured into four broad stages:

- Detailed design of the process plant and infrastructure.
- Procurement.
- Construction.
- Commissioning and handover.

A preliminary project implementation schedule is provided at a high level in Figure 24-1.

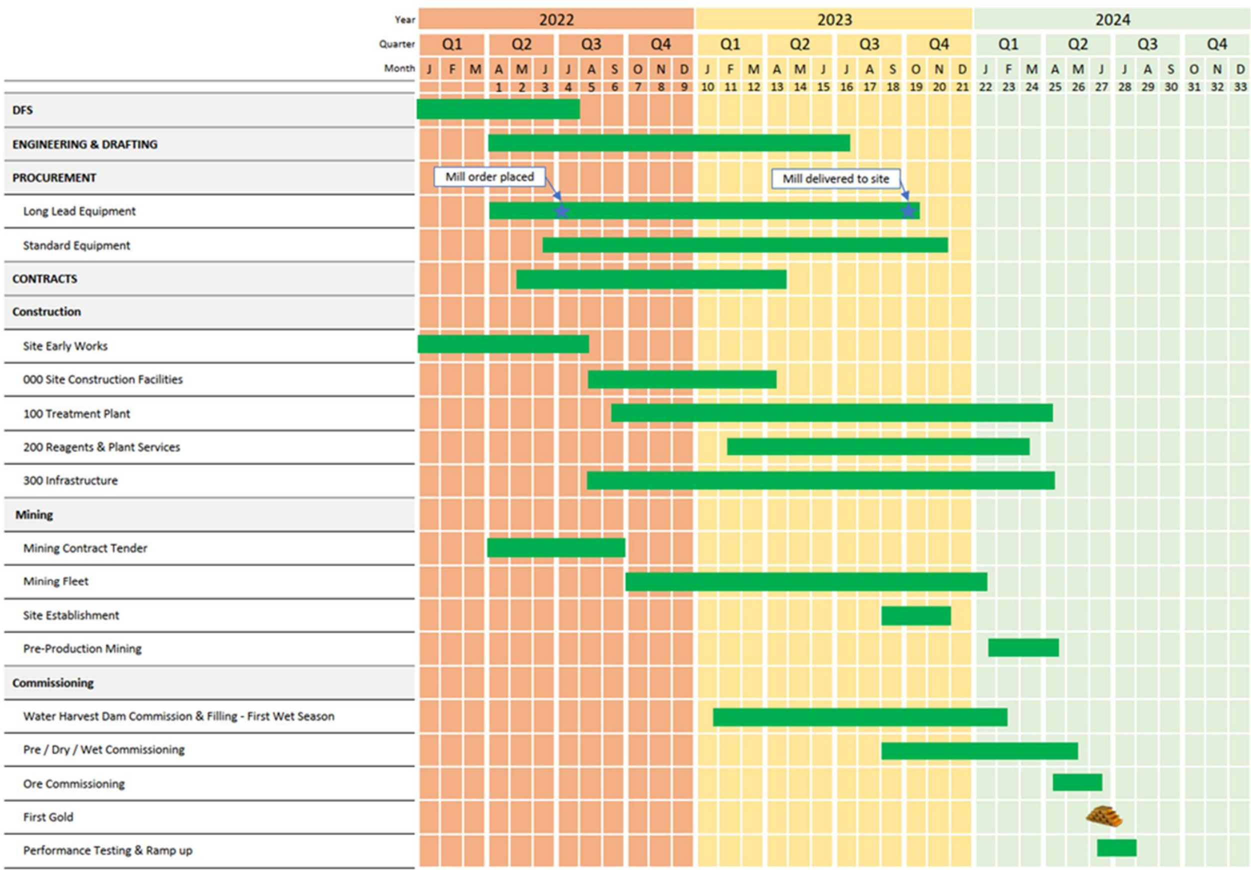

Figure 24-1: Project Implementation Schedule Summary

The detailed implementation schedule prepared by Endeavour for the overall Project is provided in Appendix 16.3. Additionally, Appendix 16.4 provides the detailed implementation schedule prepared by Lycopodium, which is more focused on the process plant scope.

The schedule critical path is aligned with activities related to the supply, manufacturing, transport and installation of the Ball Mill as shown in Appendix 16.2.

The associated Project construction manning histogram is provided as Appendix 16.5.

24.2.9.1 Calendars

The following calendars have been setup and assigned in the preparation of this schedule:

- Engineering: 5 days per week and 8 hours per day including public holidays and Christmas break.
- Procurement: 5 days per week and 8 hours per day including public holidays and Christmas break.
- Fabrication: 5 days per week and 8 hours per day including public holidays and Christmas break.
- Delivery: 7 days per week and 10 hours per day excluding public holidays and Christmas break.
- Mobilisation and Site Establishment: 5 days per week and 8 hours per day including public holidays and Christmas break.
- Construction: 6 days per week and 10 hours per day (allowance had been made for an 11 to 12 day Christmas break).
- Dry and Wet Commissioning: 6 days per week and 10 hours per day.
- Ore Commissioning: 7 days per week and 10 hours per day excluding public holidays and including Christmas break.

Note that no specific calendar was set up in the schedule to allow for the wet season; however, due consideration was given to the estimated duration of activities performed during these periods.

24.2.9.2 Resources

Construction resources covering all disciplines have been created in the Primavera P6 schedule and all direct manhours for construction have been assigned/loaded into the master schedule.

24.2.9.3 Engineering/Design

Design activities had been planned to a combined area and discipline level for the main areas of work.

24.2.9.4 Procurement

The procurement phase has been planned with two focus areas. Firstly, placing orders for the long lead and critical items to ensure the earliest completion possible. Secondly, focusing on expediting and tracking the fabrication and delivery of the critical equipment, while at the same time procuring the remaining equipment and expediting certified vendor information to ensure timely completion of the design phase, so as not to delay fabrication of steelwork and piping required for the completion of construction.

The long lead items have been identified in the schedule and lead times adopted based on the vendor budget quotations received. In most cases, 12 weeks have been allowed from preparing the enquiry through to placement of order.

24.2.9.5 Fabrication and Delivery

Fabricated items have been tracked in the schedule at a combined facility level, to align with the estimate. Commencement of fabrication is determined by design completion. General delivery duration of 1 to 2 weeks has been allowed for items procured within the borders of the country.

24.2.9.6 Contracts

The major construction and fabrication contracts required have been detailed in the schedule. Timelines for the issuance of contracts, has been typically determined by the availability of sufficient design detail.

24.2.9.7 Construction

Construction works have been detailed to Level 3 (discipline level). As far as possible, works have been planned to consider:

- Clear uninterrupted access to cranes and support equipment.
- Minimising double handling of materials and equipment.
- Resource limitations around confined spaces and confined working areas (where applicable).
- Allowing for continuation of work to prevent standing time.

The construction sequence and methodology will be as follows:

- Concrete construction sequence planned to use two contractors and focusing on the larger areas first. The first contractor will focus on the main areas, with the second contractor constructing the minor areas as well as infrastructure.
- Structural, mechanical and piping (SMP) installation will follow the same construction sequence as the concrete but will also start erection from the centre outwards and conveyors happening only once the main areas are well advanced so as not restrict crane and vehicle access around the major areas like the mills, HPGR, thickeners, and stockpile/reclaim.

24.2.9.8 Commissioning and Handover

Commissioning has been detailed in three phases:

- Pre-commissioning at an area/facility level.
- Load commissioning at an area/facility level.
- Process ore commissioning at a plant level.

24.2.9.9 Schedule Build

Constraints

The schedule contains limited constraints used only where suitable relationship logic could not be applied or for special circumstances.

Relationship Lag

As a general rule, lag has been kept to a minimum.

Lag has been used in certain cases to achieve the following goals:

- Reflect the period of time where a vendor will submit data.
- Reflect the period of time where a discipline is reliant on another to progress.

Relationship lag in the schedule will be mostly applied to Finish-Finish (FF) and Start-Start (SS) relationships. Finish-Start (FS) with negative lag will be used only when there are no other relationship options.

Procurement Logic

The interaction of procurement and design on the availability of vendor data has been represented by a standard timeframe, with vendor data planned for receipt two to four weeks post award. Standard logic and timeframe for tendering is shown in Table 24-5.

Table 24-5: Timeframe for Tendering

Activity	Duration	Successor
Finalise and Issue Tender	2 weeks	Tender Period
Tender Period	4 weeks	Review proposals and award
Review Proposals and Award	6 weeks	Receipt of initial vendor data
Receipt Of Initial Vendor Data	2 to 4 weeks post award	Vendor data review and return. Mechanical design commencement.

Schedule Critical Path

The longest path is made up of the procurement (50 weeks based on NCP's lead time), delivery (12 weeks) and installation (18 weeks) of the Ball Mill, followed by the completion of piping and electrical construction and then the relevant commissioning activities. Refer to Appendix 16.2 for critical path details.

24.2.9.10 Schedule Opportunities

Due to the long lead time and being on the critical path, there is the opportunity to save time by ordering the Ball Mill ahead of full project funding.

24.2.9.11 Durations/Metrics

The following metrics have been assumed for activity durations:

- Design: determined via process plant requirements and equipment lists, durations of design activities (in the absence of man hours) have been determined from recent design history.
- Procurement and fabrication lead times: taken from budget quotes received during the study.
- Civil works: determined via estimated quantities and recent construction history.
- Steelwork installation: determined via estimated quantities and recent construction history.
- Mechanical Installation: determined via estimated quantities and recent construction history.
- Electrical and Instrumentation work: determined via estimated quantities and recent construction history.

24.2.9.12 Project Milestones

Key milestone dates for the Project are listed in Table 24-6.

Table 24-6: Key Milestone Dates

Activity	Date
Approval to Proceed with Detailed Design	04-Apr-22
Commence Procurement of Long Lead Equipment	4-July-22
Commence Process Plant Earthworks	19-Sep-22
Commence Process Plant Concrete Works	04-Jan-22
Design & Engineering Complete	14-Jul-23
Commence Commissioning	14-Dec-23
Ore to Mill	12-May-24
First Gold Product	17-Jun-24

24.2.10 Quality

The Project Quality Plan (PQP) will cover all work to be undertaken and all services provided by the EPCM Contractor and its subconsultants including the provision of EPCM services, subconsultants and other suppliers contracted to undertake work on the Project.

The PQP sets out the quality objectives for the Project and provides the framework for effective quality management during execution. The document also sets out the measures by which Project achievements can be assessed against key performance indicators (KPIs).

24.2.11 Procurement and Contracting

The PEP will address the major procurement and contracting activities, and detail the strategies, methodology, procedures and controls that will be adopted during the delivery of the Project.

Packages will draw on the similarities of the Owner's recent supply of equipment to previous gold projects in West Africa. Where the specified equipment is identical to previously purchased equipment, the previous quotation will be revalidated, and the quotation checked for technical conformance. Other packages will be competitively tendered to achieve competitive pricing, and an effective negotiating position to provide value for money to Endeavour. Packages to be sole sourced will be duly justified and first agreed with Endeavour.

Equipment suppliers will be selected on the basis of technical compliance, previous performance and availability to supply relevant equipment within the Project timeline. Contractors for site works will be selected on the basis of their safety record, IR record, previous experience with similar type projects, cost, schedule, availability and capability to perform the work.

A logistic services provider will be engaged to consolidate all Project freight, provide sea passage to Abidjan, arrange port and Customs clearance, and arrange road transport to site. With existing mines in operation in the region, no insurmountable logistics issues are anticipated.

Local contractors and suppliers will be encouraged to tender for all project works and contracts for which they are qualified to undertake and will be assessed based on their ability to meet the required conditions. It is planned that direct negotiations will be undertaken with smaller local business groups with specific contract packages to encourage local sourcing of project requirements.

Construction contracts will generally be tendered as horizontal packages, that is, by discipline of construction work (e.g., concrete, SMP erection) although in some cases, a vertical design/supply/install contract may be considered.

24.2.12 Construction Plan

The construction management requirements will be outlined in the PEP and details the strategies and resources required to construct the works. This includes defining the responsibilities of all parties during construction activities to ensure they are undertaken in a safe and organised manner.

As previously identified, the construction strategy will be largely governed by the Owner's team with the EPCM Contractor providing key supervisory roles within the Owner's onshore structure.

Bulk earthworks for the process plant will be undertaken either by an earthworks contractor or self-performed by Endeavour. The works will initially focus on areas required for temporary facilities and priority buildings to allow these facilities to be established prior to the major plant packages commencing. Temporary facilities include the EPCM's construction offices and project laydown areas.

Concrete works will commence in areas identified to provide earliest access to install major structural steel and site erected tanks.

SMP and E&I installation packages will be structured to provide maximum overlap of activities with preceding disciplines, but without causing excessive interface issues.

Construction activities will be prioritised and managed to facilitate an orderly handover for pre-commissioning activities which will then lead into dry commissioning as operable sections of the plant and infrastructure become available.

Handover to operations for wet commissioning will be on an area-by-area basis to facilitate the early commencement of operational activities and transition to the operations phase.

24.2.13 Project Commissioning

A Commissioning Execution Plan will be prepared for the Project.

This document will outline the plan for pre-commissioning and wet commissioning of the process plant and infrastructure. It will also outline the plan for process (or load) commissioning of the plant followed by ramp up to design capacity and execution of performance tests.

The EPCM Contractor will provide commissioning services and facilities to ensure the proper execution of the various commissioning phases and bringing the Project into service in a controlled and timely manner to the satisfaction of Endeavour.

Assistance will be provided to Endeavour, if required, for developing and implementing its operational readiness plan.

The EPCM Contractor will assist with initial commissioning runs to ensure that plant performance is in accordance with the specified design/performance criteria and to provide such additional supervision and expertise as is required to identify and rectify defects and thereby enable the plant to operate at its specified parameters.

Formal performance trials will be carried out to confirm the completed plant meets its key performance criteria.

24.2.14 Project Close Out

At the completion of all construction and commissioning activities, the EPCM Contractor will provide the following close out information to Endeavour:

- As built drawings covering piping and instrumentation diagrams (P&IDs), electrical, and others as agreed.
- Testing and commissioning data and records including instrument calibration sheets.
- Documentation for the discharge of vendor and contractor bank guarantees and warranties.
- Project close out report.
- Quality records.

24.3 Data Verification

The data verification process employed for Sections 24.1 and 24.2, is discussed in Section 12 of this Report.

24.4 Interpretations and Conclusions

Interpretations, conclusions and risks for Sections 24.1 and 24.2, are presented in Section 25 of this Report.

24.5 Recommendations

Recommendations for Sections 24.1 and 24.2, are presented in Section 26 of this Report.

24.6 References

References cited in the preparation in Section 24 are presented in Section 27 of this report.

25. INTERPRETATIONS AND CONCLUSIONS

25.1 Property Description and Location Section

SML have the required permits to start developing the Project on PE 58. Further the QP is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform the proposed work programme on the Properties held by the Issuer.

However, it is notable that:

- The renewal Permit for PR 329 is outstanding.
- The relevant Permits are in place for starting to develop a project on PE 58 but the signing of the 'Mining Convention' is significantly outside of the required timelines[156] as defined in the CI 2014 Mining Code. This goes to defining the tax derogation basis for SML and its contractors.
- The approved ESIA and MRCP was based on the pre-feasibility study results. Whilst the DFS is not significantly different, as per Article 6 of the Environmental authorisation No. 00044/MINEDD/ANDE, dated 18 February 2021, ANDE must be notified accordingly of scope changes to the original ESIA (not done as per the 'Effective Date' of this Report).
- The award of PE 58 was based on a pre-feasibility mine plan and production schedule, and as per Article nine of Decree n° 2021-538 of September 2021 granting PE 58 to LMCI, the Issuer needs to notify the Minister of Mines, Petroleum and Energy, that the plan is now different to that proposed.
- The Mine Closure and Rehabilitation Bond Basis needs to be finalised, specifically the:
  - escrow account is to be opened within 20 days following first commercial production; and
  - bank guarantee to be put in place within 120 days from date of first commercial production.
- A Permitting/agreement/notification register is under development for the construction, operational and closure phases of the Project's/Mine's life cycle. Until such time as this is complete and aligned to the construction and operations schedule, it is not possible to say with certainty that all relevant permits will be in place in time. Notwithstanding this, there is likely sufficient time to address if acted upon expediently.

25.2 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The positioning of PE 58, just off the main arterial route from Abidjan to Burkina Faso, coupled with in-country; political stability; high GDP growth rates and associated infrastructure development spend (roads, energy, communications, ports and rail); and, public private partnerships as described herein, are seen as highly favourable for the Issuer's interests in the Lafigue Project and CI as a whole.

The government of CI's successful National Development Programmes (PND's) and the associated funding by external intergovernmental organisations (IGO's) and private organisations is having a significant impact on CI's GDP growth and on the quality/capacity of in-country enabling infrastructure which supports cross border trade and the development of heavy industry (i.e. mining), both on a national and transnational basis.

[156] Should have been signed by 15 December 2021.

CI's installed and planned hydropower dams, thermal gas fired power stations, and offshore gas/oil fields have enabled growth and stable/low energy power prices for a number of years. Whilst there were concerns that gas/oil would run out within the next 10 or so years, recent offshore oil/gas finds suggest that the country's energy independence can be maintained, subject to oil/gas field development/commercialisation timelines. It is noteworthy that coal fired power plants and LNG facilities planned to address growing energy demand and perceived energy short falls, have not materialised and periods of low rainfall, without sufficient spinning energy reserve capacity, created power rationing in CI in 2021.

CI has ambitious solar and hydro power plans as part of their CO_2 reduction commitments; however, both have low capacity factors. Thus, more gas fired thermal power stations will be required, particularly if the planned coal fired power stations and LNG terminal do not materialise. It should be noted that an LNG terminal would have made CI subject to globally traded energy prices/shocks to some degree, moderated only by in-country production. Further, it is unclear how the CI gas price, is linked to internationally traded gas prices. Whilst oil is exported, a liquification facility would be required to export CI gas.

There is nothing to suggest that another low rainfall year could not happen in CI, with the attendant in-country power rationing. However, ECG (the Issuer's Electrical consultant) believe that heavy industry would be least likely to suffer outages (>98% availability) and that power quality on the 225 kV network is good.

Notwithstanding this, power pricing and power availability is still an operational risk factor and effort will be required by the Issuer to understand CI's future energy supply scenarios and constraints. The Issuer will also need to consider all contractual issues, associated with any in-country power rationing and the associated valued chain disruptions.

The installation of solar in-country or at the mine[157] by the Issuer, may provide additional tools to minimise 'load shedding/power rationing' risk, and the Issuer could structure a deal with CI Energies that would provide favourable fixed term tariffs. Benefits associated with CO_2 offsets, should also be considered. Low gas/hydro power pricing is likely the biggest hurdle for solar adoption.

Whilst there are large cities/towns in the area (Bouake being the second largest city in CI), there is limited to no heavy industries locally to support the mine and without other mines in the area, it is unlikely that OEMS/mine service providers will establish in the area. Thus, for the foreseeable future, the mine will likely be serviced from Abidjan and abroad. This is a similar situation to the Issuers' other mines, and no issues are foreseen.

There is some concerns with respect to the availability of skilled local labour and the Issuer will need to implement a labour sourcing/training development plan upon completion of the feasibility study. Said plan should also factor in the Issuer's other in-country operations.

From a physiography perspective, PE 58 is considered low risk from a construction and operational perspective, and there is sufficient space on PE 58 for all the required mine infrastructure, including but not limited to: waste rock dumps, tailings facilities, water harvest and storage dams, mine accommodation and plant.

[157] From an 'insolation' point of view, there may be better locations

The highest risk for construction and operations, is the supply of water. Notwithstanding this, the use of a non-perennial water course on PE 58 to harvest water falling in the wet season, supplemented with ground water is sufficient to maintain operations for a 100-year Annual Return Interval (ARI) dry and wet season event (Knight Piesold).

25.3 History

Endeavour has integrated the historical exploration data into their database management system and validated and verified the data wherever possible. However, it is highlighted that much of the pre-Endeavour era historical exploration data is not accompanied by comprehensive QAQC sampling checks. The QP is satisfied that the historical data has been collected, collated, and managed in an appropriate manner and is not biased or unreliable. Data used for the 2021 MRE and subsequent MREs is considered appropriate for the purposes of estimating Mineral Resources.

25.4 Geological Setting and Mineralisation

It is considered that the geology at Lafigué is reasonably well understood, with extensive exploration drilling, along with a dedicated structural study of the deposit interpreting the main mineralisation-controlling structures and their dominant trends having been completed. The key geological risk is associated with the level of understanding of the mineralization controls at a local scale, and the potential local variations in the thickness and geometry of the mineralisation as a result.

25.5 Deposit Type

The Lafigué deposit resembles a typical shear zone-hosted deposit located within the north-south-trending Oumé-Fetekro greenstone belt. The deposit is hosted by a Birimian-age complex of bimodal metavolcanics and meta-volcanoclastic rocks intruded by a series of felsic intrusions.

25.6 Exploration

Exploration works across the Project area have been broadly appropriate for the style of mineralisation present and has included a wide range of approaches including district-scale geophysical surveys, license-scale soil geochemistry surveys and more localised geological investigations, including mapping and sampling. Six targets have been highlighted as warranting more detailed exploration work.

25.7 Drilling

Overall, it is considered that the drilling procedures since 2017, including collar and downhole surveys, logging and sampling generally conform to industry best practise and provide a sound basis for the 2022 Mineral Resource estimate. Much less information is available regarding the drilling procedures associated with the historical drilling (pre-2013). It is noted that the spatial accuracy and core recovery (pre-2010) from these holes presents a risk when compared to drillholes completed using industry standard operating procedures during the more recent drilling campaigns. This data is typically concentrated along the northern periphery of the deposit (up-dip portions), where the risk is mitigated to some extent by the addition of numerous close-spaced holes completed between 2017 and 2019.

25.8 Sample Preparation, Analysis and Security

Overall, it is considered the majority of sample preparation, analyses and security protocols conform to industry best practise. The absence of QAQC sample results for the 1997 and 2002 drilling campaigns is noted and as such, assay data from these drilling campaigns present a risk in terms of accuracy and precision of the associated assay grades. The results of the 2010 to 2022 QAQC programmes are summarised as follows:

- A total of 11 458 coarse and fine blank samples were analysed between 2010 and 2022 using conventional fire assay analysis. No material issues with contamination were noted.
- A total of 14 648 pulp and field duplicates were inserted into the sample stream between 2010 and 2022, generally returning reasonable correlations between original and duplicate samples.
- The results for a range of 15 different certified reference materials submitted for analysis between 2010 and 2022 are generally acceptable. with some minor CRM mislabelling (by manufacturer) issues identified.

The QAQC analyses presented are generally of sufficient quality to support the 2022 Mineral Resource estimate.

25.9 Data Verification

25.9.1 Geology and Resources

Most aspects of data and database validation carried out for the Project are considered in line with industry best practise. Given the reliance on historical (pre-2010) drilling in some areas of the deposit, and the lack of verification surveys or QAQC sample data for these drillholes, it is considered that this data does pose a risk to the stated Mineral Resources, however this drilling comprises a relatively small component of the overall database supporting the MRE (<8% of total RC + DD drillholes used for the MRE) and is often supported by relatively close spaced younger drilling.

25.10 Mineral Processing and Metallurgical Testing

The summary metallurgical testwork outcomes are discussed below. These lead into the implications for process plant design, which are elaborated in Section 17.

- Core samples from the various ore lithologies, weathered states and mineralisation styles were selected to extend over appropriate minable widths, include likely dilution and be representative of expected life of mine (LOM) gold grade ranges, with sufficient sample to make up average grade composites. Oxide and transitional mineralisation make up less than 6% of the resource, so metallurgical testing focussed primarily on the fresh ores.
- Comminution samples aimed mainly to cover the typical lithological distribution and alteration around the contact zones where the gold mineralisation typically occurs. Examples of individual host lithologies and adjoining country rock types (likely dilution) were also sampled for comminution testing.
- Sample head assays indicated that there are few deleterious elements for gold leaching with low levels of base metals and arsenic. The high gravity gold content resulted in a reasonable degree of gold assay variability. Silver grades were generally very low, with a few exceptions where an isolated higher assay occurred. The silver is not considered to add significant value to the resource metal content.

- The comminution testwork programme produced characteristic data for the fresh ore lithological examples selected. The fresh ore is very competent with a high breakage energy requirement for the coarse particles. Minor variability between samples was noted, most likely due to differences in alteration near the mineralised contact zone. The felsic lithology (granodiorite) displayed consistently higher abrasion indices.
- A mineralogical investigation indicated that much of the gold occurs as free grains. There is some locking of fine gold in pyrite and pyrrhotite, but this is likely relatively minor given the generally high overall gold extraction. Silver telluride association with gold mineralisation was also noted and this, along with pyrrhotite association, may be responsible for some of the lower gold leach extractions and slower kinetics observed. These elements did not result in any notable impact on reagent consumption or oxygen demand.
- Cyanidation tests were conducted on the fresh master composites following gravity gold recovery at different grind sizes to evaluate the effect of grind size on gold extraction. Leaching was rapid for all grinds, following high gravity gold recoveries, with the bulk of the gold dissolution occurring within four to eight hours. The ore appeared to be relatively insensitive to grind, with only a 1% difference in gold extraction over the size range tested. A P_{80} grind size of 106 µm was selected for design and further testing following an economic evaluation of optimum grind size. This grind had similar gold extraction and lower operating costs compared to the finer grind sizes tested. Subsequent review and additional grind size comparative work indicated that there are economic benefits in finer grinding of some samples, but 106 µm was retained as the target grind for the process design basis. In practise, intensive cyanidation of the gravity concentrates may minimise the impact of the mineral associations causing slow leaching, since liberation is not the problem. Cyanide consumption was 15% higher on average at the finer P_{80} grind size of 75 µm, indicating the increase in reactivity with fineness.
- Leach optimisation testing on the master composite samples indicated that high gold extractions were achieved with air only sparging (no high purity oxygen required), relatively low cyanide dosing and high fresh ore slurry densities (up to 55% solids w/w). These conditions were used for bulk leaching to produce larger slurry volumes for physical characterisation of slurry rheology, carbon adsorption, cyanide detoxification and dewatering testwork.
- Variability leach tests were conducted on 25 fresh samples and six oxide samples in the 2019 testwork and 40 fresh ore samples in the 2021 programme. These samples were also used to make up master composites for further testing. Gold extractions from most samples were very high (>95%) with a few exceptions (five samples) that had slightly lower extractions. Increased cyanide addition (maintaining a higher free cyanide excess concentration since consumption remained low) and extra leach residence time (36 h) improved extractions to expected levels. These conditions were accepted as improving the flowsheet robustness and the base design conditions were modified to suit.
- Slurry rheology results for the oxide ores indicated that, although one sample contained viscous clay, the balance presented no material handling issues. All oxide/transition ore blends with the fresh feed will need to be managed to avoid excess fines presentation to the HPGR, but this will also have benefits for downstream operation if the oxides are slow settling and form viscous slurries. The fresh ores display low viscosities up to high operating densities and fast settling rates.
- Thickening testwork indicated reasonably high flux rates for the samples tested with moderate flocculant consumption to generate high thickened underflows (>60% solids) and good overflow clarity.
- On the basis of treating Lafigué fresh and oxide ores via gravity and direct cyanidation, an overall gold recovery of 96.5% is recommended. This recovery is based on the median of the variability testwork results after allowing for the likely soluble gold loss equivalent.

- Cyanide consumption will be 0.17 kg/t, including a free cyanide excess loss of 100 ppm NaCN.
- Lime demand to maintain pH will typically be low for the fresh ores. Oxide and transition ores in the feed blend will increase the lime consumption.

The testwork conducted has addressed all the potential risks that arose and the high degree of consistency in testwork results between the samples selected, suggests that risks relating to sample representivity are very low. A few nominal risks relating to the metallurgical testing outcomes are noted for engineering design and plant operations.

- The very competent fresh ore will require additional installed power on the crushers.
- Adding excess oxide/soft transitional ore (above the recommended maxima) to the feed blend may result in material handling issues in the dry plant but will definitely reduce the HPGR throughput. Without competent material to open the gap, the passage of material will be restricted to the minimum gap setting.
- Excess moisture (>8% w/w) in the HPGR feed can breakdown the protective autogenous layer and result in increased HPGR contact part wear rates.
- The agglomerates in the HPGR product are assumed to be readily broken down with water. The need for mechanical de-agglomeration is rare, but this has not been tested.
- There may be some variability in BWi, with one extreme value having been measured, but testwork has shown that gold extraction is relatively insensitive to grind. Finer grinding is more likely, given constraints on mining ore delivery rates.
- Efficient gravity recovery must be practised, with at least one concentrator operating at all times. Coarser gold particles reporting downstream are likely to be slow leaching and result in lower overall recoveries.
- Maintenance of (200 to 250) g/m^3 free NaCN in the CIL tanks will be key to achieving faster leach kinetics and high gold extractions. This requirement will be counter-intuitive given the low cyanide consumption experienced but is a feature of this ore and its mineralisation styles.
- Operations should be alert to slower leaching ores in the event that these are more prevalent locally and become the predominant feed source for a brief period. These ores do not typically have much lower extractions, but the slow leaching affects the carbon profile, increasing locked gold inventory and solution losses.
- During extended dry seasons when the stored water is not readily replenished, dilution of the tailings will need to be reduced, resulting in increased cyanide discharge concentrations. Since Endeavour is not a signatory to the cyanide code, this temporary deviation is not a problem, and all impacted water is contained within the process and tailings system.

25.11 Mineral Resource Estimates

SRK considers that the geological model developed for the 2022 MRE is a reasonable representation of the in-situ mineralisation based on the available supporting data. The Mineral Resource classification categories attributed to the mineralised packages estimated in the 2022 model, reflect SRK's confidence in the quality of the supporting data, as well as the level of understanding of both geological and grade continuity of the deposit, where these are reduced towards the centre of the deposit and along down-dip extensions of the mineralisation.

It is noted that when comparing the SRK 2022 geological model with Endeavour's 2020 model, the Mineral Resource Estimate has proven to be sensitive to the modelling approach, in particular, the degree to which lower grade mineralisation above the specified modelling grade threshold (0.3 g/t) is incorporated into the model, and the degree to which the resultant volumes are interpreted to represent the style and geometry of the in-situ mineralisation.

25.12 Mineral Reserves Estimates

25.12.1 Hydrogeology

The open pit hydrogeology study currently has limited characterisation and insufficient monitoring data, resulting in some degree of uncertainty in the subsequent groundwater modelling analysis and, thereon, the design of the water management plan for the open pit operation.

SRK considers that the pit hydrogeological characterisation and dewatering design has been undertaken to a 'pre-feasibility study level' of development accordingly. However, whilst there are hydrological risks, these are not considered significant relative to the geotechnical risks. Notwithstanding this, further work is required to better define the geological structural model and the associated hydrogeological conditions, and this work should be done during detailed design/FEED and prior to start of mining.

25.12.2 Geotechnical

SRK considers that BG has implemented a diligent review of the pre-existing geotechnical data, identifying gaps and defining confidence levels within the various models feeding into the Geotechnical Model and updating the slope stability analyses. Whilst the analysis undertaken is appropriate, the number of boreholes used to define the updated rock mass conditions and Rock Mass Classification values (4 No.) could be considered a lower bound and may not provide confidence in the spatial distribution of the geotechnical properties within the pit. In addition, drilling orientation bias has resulted in a very limited structural data set (especially for the hangingwall).

Other than foliation, no additional discontinuity sets have been defined, which could impact the achievability of the proposed inter-ramp angles within the hangingwall. As recognised by BG, however, this can be mitigated by additional geotechnical data collection and verification of the proposed design criteria.

Bench crest loss will be prevalent in the footwall within the design domains affected by the presence of foliation, and a 3D fault model will be critical to understanding the role the identified shear zones will have on any other footwall shears. It should also be noted that BG has used lower bound rock mass strength values (defined from Golder geotechnical logging of boreholes GTLF01 to GTLF07), within their analyses and additional data collection may show upside with regards to rock mass strength.

The pit geotechnical design criteria illustrated were incorporated during the pit optimisation and design process. Geotechnical data was limited to within the boundaries of the PFS pit design, and only the Oxide zone for the smaller satellite pits. With the updated Mineral Resource and the subsequent extension of the indicated mineral Resource, the updated pit optimisations extended outside the PFS boundary to constrain the Geotech zones provided.

25.12.3 Mineral Reserves

Whilst artisanal workers have been active on the Lafigué deposit, it has not historically been mined on a commercial basis. SRK have made assumptions to mitigate the extent to which the deposit was deleted by artisanal workers, but there is a risk of the depletion being more than anticipated.

The mineralogy at Lafigué might be visually discerned between higher and lower grades from quartz vein versus rock/shear-hosted mineralisation. Still, this relationship did not prove to be consistent within the drill core. In addition, the ore/waste contacts will be challenging to visually distinguish as the deposit incorporates diffuse packages of mineralisation, where the grade slowly drops off. Good grade and ore control practises will be required; that is, supported by internal and external training on dig polygon design and adherence to acceptable dilution levels.

It will be essential to have advanced ore control to limit losses and dilution, with the mineralisation occurring in variable thicknesses and shallow dipping veins. Losses and dilution will largely depend on the size of the ore loading unit impacting the operators' ability to maintain an adequate SMU where required. The current mine plan assumes that the ore loading unit can maintain a 5 m x 5 m x 2.5 m SMU where needed. An additional 5% dilution was applied due to uncertainty and should mitigate any impact related to a larger SMU.

Optimisation results showed that the Lafigué deposit does not benefit from a pushback-phased approach, with limited improvement in NPV for pit shells larger than USD 900/oz Au. Additional pushbacks might benefit operations for better control of activities (drilling and loading) and waste strip requirements. However, there are no clear pushbacks from the optimisation result, and these interim pushbacks will have to be iteratively designed to ensure an adequate balance between LG and HG ore coming from the pits. It is important to note that any delayed stripping will result in periods of only LG ore being mined with a direct impact on the ounces produced.

The DFS has identified an economically mineable pit at Lafigué, that has the capability of supplying the process plant, 4.0 Mt/a (db) of ROM feed for 13 years.

25.13 Mining Methods

In addition to opportunities identified, inherent risks related to the deposit, historical development, and the level of technical development undertaken to support the feasibility study are summarised in Sections 25.13.1 and 25.13.2 following.

25.13.1 Interpretations and Conclusions

An owner mining equipment and cost model was developed based on the equipment proposed in the preliminary contractor submission. The equipment model indicated that 23 Komatsu HD-1500 dump truck with 5 x PC3000 for waste mining is required, and 8 Komatsu 785 dump trucks with 1 x PC2000 for ore mining.

It should be noted that the preferred excavator size for the mine, based on the orebody and selectivity requirements, is in the range of the Komatsu PC2000. Therefore, a larger ore excavator will result in an increased SMU; however, the impact would not significantly change the results of the DFS, due to the additional modifying factors already applied.

The main finding throughout the various schedule scenarios was that during the second to third year of production, while mining the main pit pushback 2, the grades decrease while waste stripping continues to access the next higher-grade lens. This resulted in lower ounces being produced during this time, which was consistent throughout all scenarios.

Various sequence changes were assessed, but any change in sequence resulted in a delay in accessing the higher grades. The only way to overcome this was by increasing the pre-strip tonnes or increasing the total tonnes mined during the first two years.

The decrease in the pre-strip volumes, delayed access to the higher grades, with a gold production decrease still occurring. To overcome this decrease, an additional 6 Mt of total material mined is required to access the HG material earlier and improve the ounces produced.

25.13.2 Risks and Opportunities

In addition to opportunities identified, inherent risks related to the deposit, historical development, and the level of technical development undertaken to support the feasibility study are summarised below:

- Resource variation due to the highest classification being Indicated.
- Underestimation of artisanal depletion directly impacts the first year of production.
- Shallow dipping veins with variable thickness can lead to the underestimation of losses and dilution.
- Geotech domain D6 has very shallow dipping foliation planes requiring depressurisation. Therefore, the initial ore ramp switchback was not extended through this section and adequate Geotech safety berms were added to ensure the safety of lower ramps.
- Geotech drilling does not extend to the boundary of Pit B's eastern highwall. Therefore, the decreased slopes might be increased if the data supports it.
- The Lafigué deposit will have an ultimate depth of 338 m, slope angles are steeper than in the PFS, and good highwall control will be necessary.
- Not exposing higher grade areas or maintaining adequate waste stripping, will decrease production ounces for extended periods.
- An appropriate water management plan is required to ensure pit slope stability.
- Transition zone extending deeper than anticipated in areas resulting in increased waste mining.
- Current flitch heights of 2.5 m will result in lower productivity but were used to fit into the Geotechnical bench designs. An alternative bench height of 18 m will allow a flitch height of 3 m. Using flitch heights of 3.33 m to fit into 20 m benches is problematic for planning software.

25.14 Recovery Methods

The plant as designed/specified, will be able to meet the mine plan schedule as proposed (SRK, 2022) and the gold recoveries and plant operating costs as developed, are in accordance with the requirements of a DFS and the associated accuracy provision.

Outside of changes in unit input costs, no risks are foreseen in the design or in the operation of the plant, or in the operating costs estimates developed.

25.15 Project Infrastructure

25.15.1 Geotechnical

25.15.1.1 TSF

The embankment cut off trench will need to key into competent and low permeability ground. The expected typical depth of the TSF cut off trenches is approximately 1.5 m. If this is done, no issues are foreseen.

25.15.1.2 Water Storage Dam

The surface of the WSD area has been extensively excavated by the artisanal miners and it is unclear whether this has comprised solely; shallow excavation into the alluvial soils, or more extensive mining and/or the use of explosives. This activity may have significantly increased the permeability of the ground and reduced the water holding capacity of the basin. This should be assessed during the initial filling of the WSD (prior to commissioning).

Rock outcrops are present in places in both the embankment and basin areas. The alignment of the embankment may need to be adjusted on site to avoid notable rock outcrops (this has been completed as part of the early works design). Subject to the number of joints and permeability of rock outcrops in the basin area, some rock may need to be capped with a low permeability soil layer. Capping of the rock will require the removal of loose blocks, the capping with general fill to provide a surface of suitable grade on which compaction plant can operate and a 300 mm thick low permeability layer. The cost estimates for the WSD include consideration of these findings.

The embankment cut off trench will need to key into competent and low permeability ground. The expected typical depth of the WSD cut off trenches is approximately 3.0 m deep. This depth takes some account of the ground disturbance as a result of artisanal mining.

25.15.1.3 Water Harvest Dam

Near surface material is similar to the TSF material, where high plasticity clay materials are predominant with some areas of more granular material.

It is expected that the existing ground conditions have sufficiently low permeability, to retain water for this period without excessive seepage.

Areas of poor ground can be expected in the valley floor and some material will need to be removed and replaced during the construction of the embankment.

The embankment cut off trench will need to key into competent and low permeability ground. The expected typical depth of the WHD cut off trenches is approximately 1.5 m.

25.15.1.4 Plant

The Plant terrace will predominantly be located in cut, which averages approximately 2.5 m, with much of the more compressible surface soil removed.

The ground conditions are considered suitable to support ground bearing foundations and thus, piling or similar approaches are not expected to be required. However, some settlement reduction measures are likely to be required.

The calculated values of settlement indicated are generally lower than the allowable settlement values, with the exception of total settlement for the main stockpile and differential settlement for the HGPR. The settlement of a number of other structures are borderline. These structures require more detailed consideration with respect to allowable settlement values.

High plasticity clay soils are present at the site which have the potential to shrink and swell with seasonal changes in moisture content. It is recommended that foundations are founded at a minimum of 1 m depth.

There is the potential for rock to be present at of close to foundation level. Care is needed to avoid structures 'straddling' rock/hard spots as this will create peak pressure points on the foundation and can lead to an increase in differential settlement. Hard spots should be removed to below foundation level and backfilled with structural fill (depth to be determined).

25.15.1.5 Airstrip

The compatibility of the airstrip pavement design with the in-situ foundation conditions and available construction materials has been confirmed through geotechnical investigation and associated laboratory testing. The testing indicated that the in-situ material will generally suitable as bulk fill and subgrade material for the construction of the airstrip, and that a higher specification base course for the runway sheeting will be sourced available from local borrow areas.

It is noted that the in-situ gravel material may achieve a California Bearing Ratio (CBR) value of 40%. Typically, a CBR value of greater than 80% would be recommended for airstrip pavements, but is subject to material availability and the acceptance of additional maintenance requirements. In this instance, the in-situ materials were designated marginal for use as pavement material, accepting that the pavement may degrade and require increased and more frequent maintenance.

25.15.1.6 Construction Materials and Aggregates

Local off-site quarry materials met the specification requirements (Australian Standards) for coarse aggregate with the exception of the Los Angeles (L.A) abrasion test, which was borderline. The borderline test results do not discount the use of the materials, and it is recommended that strength testing of concrete mixes is undertaken to confirm that concrete mixes using the coarse aggregates meet the required strength specifications.

Selected ferricrete, laterite and gravel colluvium are expected to be suitable for sub-base and basecourse for unsealed roads, and structural fill. The ferricrete and laterite are preferable, as they tend to be of intermediate and not high plasticity.

25.15.2 Roads and Airstrip

The DFS design of the site access roads and haul roads presented herein is suitable for the requirements of the study. No significant risks are foreseen.

The DFS design and geotechnical investigation of the site airstrip presented herein is suitable for the requirements of the study. No significant risks are foreseen.

An air strip pavement inspection and maintenance plan will be required.

25.15.3 Mine Services Area

The Mine Services Area (MSA) as described in Section 18, is fit for purpose and has been costed by the Mining Contractor, as part of the tender process. If an owner mining operation were reverted to, capital costs may differ.

25.15.4 Emulsion Plant and Explosives Storage

The use of an equivalent emulsion facility from another Endeavour CI operation for layouts and costing is considered reasonable, and aligned with the requirements of a DFS. For the explosive services tendering stage, consideration should be given to:

- determining whether an emulsion facility should be established on site, or whether the material should be imported directly; and,
- if established on site, co-locating close to the MSA area for the purpose of sharing Site services/facilities.

Whilst not material to the study cost estimate, the services/utilities required to support the emulsion and explosive facility should be jointly defined in greater detail with the explosive contractor during the tendering process.

25.15.5 Power Supply

Whilst power supply quality in West Africa generally has been questioned in the past, this particular connection at 225 kV is considered to be very reliable. There are no other customers other than the network operator themselves at 225kV and the network is a ring system that has redundancy. Even with hydroelectric dams becoming low due to drought, the high voltage customers are normally the last to be shed from the grid and history has shown, that the major mining loads have not suffered severely in such cases. The mining loads are an attractive load to CIE being a high load factor, with mining companies normally being the best contributors to the tariff system.

No other risks identified.

25.15.6 Tailings and Water Management

25.15.6.1 Overview

The DFS design of the TSF, geotechnical investigation and tailings physical and geochemical testing undertake is suitable for advancement to detailed design with no additional work required, subject to any amendments required by relevant authorities during the permitting process.

The TSF is designed to accommodate a total of 41 Mt (db) of tailings based on the Sc12I mining schedule. It is noted that the current mining schedule (Sc13k) requires an additional 6.7 Mt of tailings, the impact of overall costs for the study is not considered to be material. It is estimated that the TSF can be expanded to approximately 80 Mt (db) before impacting other site infrastructure, subject to embankment stability checks. As such, the additional tonnage in the current mining schedule will not prohibit the current TSF location, nor warrant additional siting studies.

The level of detail and information/data utilised in the water balance modelling of the infrastructure is reasonable and in-line with the requirements of a DFS.

25.15.6.2 Tailings Storage Facility Risks

Tailings Beach Slope

The design is based on an average tailings beach slope of 0.67% (150H:1V); however, the beach slope is heavily dependent on the grind size and the ore blend. Thus small changes in plant performance or design, ore type, or the ore blend have the potential to change the tailings beach slope.

If the measured beach slope is steeper than the design slope, the tailings rate of rise against the TSF embankment will be faster than expected, and the Stage 1 TSF will reach its tailings storage capacity earlier than the design. If this were to become an issue, the response would be to move Stage 2 construction of the TSF forward. Commencing the construction one or two months earlier would not have a significant impact on the operation or construction schedule, as the construction would still occur predominantly in the dry season. It should be noted Stage 1 capacity is 36 months, which provides a high level of flexibility for the construction schedule if required. In addition, the deposition line could be extended to the eastern valley to provide additional tailings storage capacity without impacting the operation significantly. It should be noted that for steeper beach slopes, the potential tailings storage would be reduced, but the storm water storage capacity would increase accordingly.

If the measured tailings beach slope is flatter than the design slope, the capacity of the Stage 1 TSF to store tailings would increase. The overall TSF stormwater storage capacity will not be affected unless Stage 2 construction is deferred beyond the original construction schedule.

Achieved Densities

The staged TSF embankment crest elevations are based on the ore blend and throughput used for the water balance modelling. Changes in these characteristics and/or throughput will result in changes in the achieved densities in the TSF. Similar to the variations in tailings beach slope, this may result in an adjusted construction schedule for the first raise, either earlier or later than the design timing. It is recommended that monitoring of throughput, ore blend, rate of rise and achieved densities be undertaken so that suitable planning and staging of the future embankment construction can occur.

Life of Mine Planning

Any changes to the LoM plan or throughput will impact upon the tailings management requirements for the site. Any significant increases in throughput may result in lower tailing densities being achieved within the TSF, thus increasing construction costs. Any decrease to the total tonnage may require reconsideration of the proposed closure plan, as the closure spillway may become prohibitively deep. In addition to the impacts on the TSF design, any changes to the operating throughput and percent solids of the tailings may impact water demands.

Availability of Mine Waste

Design of the TSF is based on structural fill material being sourced from the open pit mining operations for Stage 1 and construction of future raises. If waste is not readily available during the Stage 1 construction, additional borrow areas will be required in proximity to the TSF. Although this is possible, the capital cost will increase. Utilising a civil earthworks fleet to win material from the Open Pit footprints may prove to be uneconomical due to the long haul distances. In this scenario, material may be sourced from within the TSF basin area, which may offset some of the increased costs by providing additional capacity within the TSF (thus reducing the embankment fill volumes).

Likewise, suitable low permeability fill material may be stockpiled by the mining operation at locations in close proximity to the TSF embankment, for use by civil contractors in future stages. This may reduce civil earthworks rates during future construction raises.

Tailings Solids and Supernatant Geochemistry

Geochemical testing was carried out on tailings solids and supernatant solution. Further geochemical and solids testing of the tailings should be continued at points throughout the life of the facility (nominally within the first year of operation and then every two years thereafter) to ensure that initial testing remains valid. Measurements will need to continue as part of ongoing operations to ensure information is available on the geochemical and physical behaviour of the tailings. This testing should be included in the standard operational cost estimates.

25.15.6.3 Water Management Risks

Life of Mine Planning

Any changes to the LoM plan or throughput will impact water demands for the process. Any significant increases in throughput and percent solids of the tailings may impact water demands, and thus the required abstraction rates from the WHD to the WSD. A contingency on the pumping rates has been considered to account for operability; however, throughput increases during operation have not been considered.

Water Supply Risks

The WHD capacity and abstraction rates to the WSD have been designed with capacity to supply sufficient water for the process under 1 in 100 Year ARI dry conditions. The runoff reporting to the WHD (based on its catchment area) was estimated for all years of the Dabakala rainfall record (1922 to 2000). On average, the WHD intercepted 3.9% of all runoff estimated to report to the WHD location over the course of each year (maximum 8.1% for the driest year on record). As such, if the runoff coefficients upstream of the WHD are considerably lower than that estimated, it is considered that there is significant contingency within the design that process requirements should still be met; however, the impact on downstream catchments would increase and further assessment should be completed. The initial filling of the WSD and WHD should be monitored pre-commissioning to calibrate runoff coefficients and basin permeabilities, and thus verify water balance modelling outcomes.

Sediment Generated by Artisanal Mining Works

Significant artisanal mining works have been noted within the stream bed upstream of the WHD location. Stream flows in these areas may collect a significant amount of sediment, which may impact the clarity available in the WHD reservoir (for abstraction to the WSD). This should be assessed by the environmental consultant to determine the requirement for additional source control (over and above that listed in Section 18.2.9) upstream of the WHD reservoir to reduce this sediment load. This may comprise a series of rockfill check dams within the main stream bed.

25.15.6.4 Tailings Storage Facility Opportunity

The current TSF design includes a decant tower system used to abstract water from the TSF for use in the process circuit. A decant tower system will require significant earthworks and need to be relocated as the pond migrates during operation. It is proposed that a floating turret system may be viable for the project and could lead to significant savings to ongoing costs and should be investigated in the designed design phase. It is recommended that the performance of the decant turret system that is currently planned to be commissioned in early 2022 at Endeavour's Hounde gold mine be reviewed by Endeavour for its suitability at Lafigué.

It may be possible to defer the TSF chimney drain construction to Stage 2 to reduce capital cost in Stage 1. It is also noted that a reduction in Zone F supply costs could be reduced if a suitable on-site quarry or mine waste stockpile can be established prior to construction of the Stage 2 TSF embankment.

25.15.7 Balance of Infrastructure

For the balance of infrastructure provided by Lycopodium, the level of engineering and cost development is in alignment with the requirements of the DFS and no risks are foreseen that would materially impact the validity of the study.

25.16 Market Studies and Contracts

25.16.1 Market Studies

The commodity prices detailed in the column titled 'Modelling' in Table 25-1 have been used for the modelling of resources, reserves, operating costs and revenue.

Gold pricing used for resource and reserve modelling (reserves: USD 1300/ozt; resources: USD1500/ozt) are reasonable and in alignment with industry norms stated in Section 19.1.2. Based on a mean long-term nominal and real gold price of USD 1746/ozt and USD 1668/ozt respectively, the use of a gold price of USD 1500/ozt for revenue modelling is considered conservative.

Silver is not declared as a resource or reserve in Endeavour's current financial models and a silver price of USD 15/ozt is used for internal budgeting purposes only.

Table 25-1 illustrates a range of values that should be considered in any sensitivity analysis, namely, the long term price (LTP) worst case scenario (WORST-SCN), moderate scenario (MOD-SCN) and an optimistic scenario (OPT-SCN). Points to note:

- Silver is not currently modelled in any of the Issuer's NI 43-101 Technical Reports, for the properties that it holds exploitation rights for.
- The column titled 'Possible Duration' indicates the period/duration where the WORST-SCN and OPT-SCN may be applicable.

Importantly, market forecasts cannot adequately consider global trade rebalancing, the impact of disruptive technologies; war (physical and trade), economic recession, high interest rates, sudden legislative changes (national/transnational) and political instability.

Table 25-1: Endeavour Assumptions for Modelling and Sensitivity Analysis

Parameter	Units	Modelling	Sensitivity Analysis				
			LTP (Nominal)	WORST-SCN	MOD-SCN	OPT-SCN	Possible Duration[158]
Gold Resources	USD/ozt	1500	1459	1300	1600	1700	N/A
Gold Reserves	USD/ozt	1300	1356	1300	1400	1500	N/A
Gold Revenue	USD/ozt	1500	1746[159]	1500	1675	1850	1 year
Silver Revenue	USD/ozt	15	22.5	15	22	24	1 year
Diesel Price	USD/L	0.91	0.91	1.28	0.91	0.79	2 to 3 years
Steel Price	%	N/A	-30	10	-20	-30	1 year

[158] Abnormal spikes, either up or down
[159] Median nominal LTP from Table 19-3

The principal risk relating to 'market studies', pertains to the commodity and raw material input prices used, which may impact operating and construction costs. Conversely, there may be opportunities, should the gold price increase above the current LTP forecast for a prolonged duration.

There are concerns around the current geopolitical conflicts and inflation, and the potential impact that this may have over the short-medium term on input prices.

Since 2018, CI has managed to artificially control the diesel price at around 0.89 to 0.92 USD/L, whilst the Brent crude price over this period varied between USD 29 and 114/bbl.

25.16.2 Contracts

The tender and budget quotation methodology used to develop costs for the DFS is in accordance with standard DFS requirements. However, there are concerns that the basis of quotation/tender is not optimised with respect to the provision of services between SML/the contractor and other contractors. Potential savings may be realised in both costs and labour numbers if optimised.

The appropriate use of benchmarking/factoring to derive costs is likely to deliver costs in accordance with the requirements of a DFS, however missing out a commercial stage in the development of a project makes the next stage of negotiation more complicated and business optimisation opportunities could be missed. It may also impact the development of labour numbers, which has a spill over effect on infrastructure sizing and the basis of contracts.

Until such time as the Lafigué mining convention is signed, the tax basis for the outsourced service providers is not known. This has possible implications for how commercial contracts are set up, specifically with respect to the provision and charging basis of facilities and services.

In moving forward into the project execution phase, the mine business model needs to be optimised/finalised with respect to the provision of facilities and services between parties and how each party is to be charged. Consideration also needs to be given to local procurement/business development.

Given the fast track into operations, it is a risk that the outsourced services contracts and their basis has not been more fully developed in the DFS, specifically with respect to local development and associated labour and social obligations. There is time to address this, but it needs to be addressed as soon as practical.

SML do have the option of utilising contractors providing services to its other CI mine, and this may provide operational economies of scale for the contractor, and the potential for further cost reductions.

25.17 Environmental Studies, Permitting and Social or Community Impact

The proposed Lafigué Project will generate adverse Environmental and Social (E&S) impacts, the most significant of which will be related to physical and economic displacement, contribution to reduced biodiversity value and possible contamination of water resources. The Study Area is already characterised by extensive ecological degradation due to anthropogenic activities, such as ASM and agriculture. The socio-economic baseline also reveals inadequacies in provision of socio-economic infrastructure and services. The effective implementation of the ESMP will not only aid in avoiding or reducing the severity of identified adverse impacts but will also present an opportunity for more sustainable mining practise and investment in social benefits. No fatal flaw has been identified with respect to E&S considerations at this stage. To this end, it can be determined that benefits of the Project outweigh the adverse impacts (on provision the ESMP is effectively implemented). Without the Project the ore resource would continue to be exploited via ASM mining practises, and a large part of the benefits of a well-capitalised mining project would not be realised.

The subsections below provide a more detailed interpretation of the specific environmental and social aspects in relation to the project.

25.17.1 Impact of Site Layout between the ESIA and the DFS

The PFS infrastructure layout used for the ESIA to assess the impacts associated with it, is different to the layouts proposed for the DFS (i.e., Figure 20-3 versus Figure 20-4), given that the DFS considers new; engineering, geological, mining, and cost information This subsequently led to an optimisation of the Site layout, and new design criteria for key infrastructure features. The updated layout needs to be discussed with authorities and the ESIA updated, to reflect the layout as it is going to be constructed.

Key differences related to the environmental and social impacts of the differing layouts are detailed in Table 25-2.

Table 25-2: Identified Changes in the Project's Layout Plan (ESIA Versus DFS)

Infrastructure Item	Change	Impact
Airstrip	Significant change. It is now east of Lafigué as opposed to southeast.	Possible change in magnitude of nuisance impacts (noise, dust, visual).
Plant and processing facility	Same location.	There shouldn't be changes in noise or dust impact.
Waste Rock Dump (WRD)	Consolidation of two WRDs into one.	Positive for the residents of Lafigué as there will be less vehicle movements close to them and less visual intrusion. Good geochemical signature, so potential of very little changed groundwater impact.
	Relocation of WRD to the east.	Possible change in extent of future contamination plume.
Tailings Storage Facility (TSF)	Increase in TSF size and change to wall design.	More topsoil to be cleared. Same location so minimal change in impact based on location. Possible change in extent of future contamination plume.
Water storage dam	Minimal changes.	No significant change in impacts.
Exploration camp	Same location.	No change.
Fence line	Fence line moved further north, near Lafigué to encompass new airstrip location.	Still within mining boundary.
Water harvest dam	Changes. This was not shown on previous layout in ESIA.	Possible wetland, aquatic and surface water impacts
Solar plant	New addition. This was not shown on previous layout in ESIA.	Minimal impacts expected, near to processing plant.
Accommodation	Only a slight movement.	Minimal impacts expected.
Haul roads	Significantly different.	Within the main footprint area, so no significant change in impact.
Site access roads	Significantly different. More roads closer to Lafigué.	Possible additional or increased magnitude of nuisance impacts.
Explosives magazine	Was not shown on previous layout in ESIA.	River crossing, possible wetland, aquatic and surface water impacts.
Water pipeline corridor	Whole route not shown on previous layout in ESIA.	Various river crossings, possible wetland, aquatic and surface water impacts
Security post	Changed location.	No significant change in impacts.
Transmission line	Slight change in route.	No significant change in impacts.

None of the changes to the layout or plans are significant enough to justify a wholescale re-assessment of the ESIA, but can be dealt with in terms of an ESIA update, the contents and the format of which needs to be agreed with the authorities. All of the proposed changes are roughly within the previously impacted footprint.

25.17.2 Air Quality

Insufficient survey methodology and no evidence of a detailed emissions inventory and dispersion model to inform the quantification (intensity and extent) of impacts associated with fugitive dust and volatiles emissions was found. An understanding of the extent and intensity of the impact need to form the basis of quantifying and prescribing management measures for the Environmental and Social Management Plan (ESMP).

25.17.3 Noise

Insufficient survey methodology and no evidence of a detailed noise source inventory and dispersion model to inform the quantification (intensity and extent) of impacts associated with increased and cumulative ambient noise levels was found. An understanding of the extent and intensity of the impact needs to form the basis of quantifying and prescribing management measures for the ESMP.

25.17.4 Surface Water

No impacts of the dams on the river are discussed or the systems downstream, only two samples were collected, which is likely to not be representative of the hydrology of the site in the ESIA. Endeavour has since established a surface water monitoring programme which is conducted on a monthly basis. It is recommended that a detailed surface water monitoring plan be implemented as soon as possible, and continued through all phases of the project.

25.17.5 Groundwater

A groundwater model was developed in May 2021 at a pre-feasibility level, for pit water management during operation. The study did not investigate post-closure rebound rates and decant predictions.

In addition, no mine-related impacts on the groundwater environment were conducted. Pit dewatering will lower the water table, which could affect nearby private boreholes and streams. Such impacts need to be quantified considering radius of influence and impact duration. The proposed TSF could also leach and contaminate the aquifer. The contamination plumes originating from the mine infrastructure needs to be predicted during and after mine closure, and proper mitigation measures need to be put in place.

During the pre-feasibility study, only five percussion boreholes were drilled within the pit footprint area. These, however, are not sufficient for impact assessment. Monitoring boreholes need to be sited taking the source-pathway-receptor dynamics of the Lafigué site.

25.17.5.1 Groundwater Recommendations

For baseline reference, regular monitoring of surface and groundwater conditions needs to continue, before mine construction.

The following hydrogeological activities are recommended for a groundwater impact assessment and mitigation planning:

- Ongoing monitoring of the TSF, WSD and WHD.
- Monitoring instrumentation shall be installed during operation, as required by the Operating Manual of each structure (to be provided prior to commissioning), including routine auditing.
- Monitoring and auditing costs should be included within the standard operating cost estimate.
- TSF monitoring boreholes be installed during the early stages of construction to facilitate the collection of baseline readings.

25.17.5.2 Aquifer Characterisation and Monitoring Borehole Drilling

The boreholes need to be sited strategically considering mine infrastructure and geological structures. To site boreholes along water-bearing fractures, geophysical surveying needs to be done first e.g. ground geophysical surveys, particularly Electrical Resistivity Tomography (ERT) survey and ground magnetic survey. ERT is a preferred method to identify fracture zones that are preferential pathways. Available air-borne magnetic survey map will be needed to delineate dykes which often control groundwater movement. Aeromagnetic maps are a good starting point, however, magnetic survey identifies structures with only magnetic anomalies, such as dykes. Water has no magnetic property and water-bearing fractures are better interpreted using resistivity surveys. This information should be integrated with the magnetic data as well as geological information, to site boreholes for rock permeability assessments.

25.17.5.3 Hydrocensus

This is needed to understand the extent of groundwater users and surface/groundwater interaction and is important for the future liability study. The background water quality consisting of full-suite analysis is needed, particularly focussing on high-risk monitoring points.

25.17.5.4 Borehole Drilling

Dewatering and aquifer characterisation boreholes should be drilled at fracture areas, following the geophysical results. The PFS hydrogeological study recommended the drilling of nine dewatering boreholes – consisting of three in-pit and six out-pit. However, the previous model was developed with limited hydrogeological, structural and geological data, and needs to be updated to further refine the number, location and design of the dewatering boreholes.

25.17.5.5 Aquifer Testing

All the dewatering boreholes should be aquifer tested.

Any borehole with a blow yield of 0.5 L/s or less should be used as part of the monitoring network, but not dewatering. The permeability of these boreholes is too low to be used for pumping. Such boreholes only need to be slug tested.

Any borehole with yield of more than 0.5 L/s should be subjected to pump tests. Initially a step test should be done for two hours, each step being 30 minutes long. This should be followed by a 24-hour constant rate test.

Packer test should be done in at least six exploration holes to profile the permeability of the geology though depth. A packer test is required to profile the permeability along each fracture zone and its significance for groundwater ingress.

25.17.5.6 Conceptual Model Updating

The conceptual model needs to be updated taking the source-pathway-receptor dynamics of the aquifer system. The groundwater information obtained from the above activities should be integrated with the geological structural model to update the hydrogeological model.

25.17.5.7 Numerical Modelling

The existing numerical model needs to be updated considering site specific hydrogeological conditions. A more reliable and accurate inflow rates will then be obtained. The model will also be able to estimate the potential environmental impacts with acceptable accuracy.

25.17.6 Biodiversity

25.17.6.1 Freshwater

No wetland assessment was completed, this is a risk as it is considered sensitive environment by the CI Environmental Code and for this reason, an EIA is undertaken in some cases. Additionally, only one aquatic wet season sampling site was surveyed which may not be representative of the system as a whole. Additionally, there is no upstream and downstream impacts to act as a baseline for future impacts of the mine.

Now that the ASM miners have been mostly removed off site and the water courses are receiving water again, it is recommended that a wetland assessment is undertaken.

25.17.6.2 Flora

There is no habitat delineation indicating sensitive areas which is a requirement for the biodiversity management plan. Furthermore, new species have been added to the IUCN Red list, which are present on site, and therefore it would be prudent to identify the locations of these trees and any other sensitive habitat.

25.17.7 Social

25.17.7.1 Engagement in Terms of Final Land Use Plan

The rehabilitation plan should incorporate an engagement plan specifying how communities should be involved and their preferences recorded in terms of the final land use plan. The rehabilitated lands will be handed over to government at closure. This will go through lot of discussion with relevant stakeholders to align with them on what this land will be used for. The recommendations will be part of the closure plan that will be submitted to national authorities for validation.

25.17.7.2 Loss of Livelihood/ Project Social Impacts

Given the importance of agricultural practises in the area, the loss of livelihood based on agriculture should be considered during all the phases of the project, and related mitigation measures should ensure the maintenance of existing means of agricultural practises, or the development of alternative options.

For impacted communities, a LRP will be implemented over three years.[160] After year three, the plan will be audited to ensure that the LRP has impacted the community positively, and if there are gaps, ensure that these are addressed.

25.17.7.3 Stakeholder Engagement

The ESIA states that approximately 26% of the 81 people consulted have expressed the need for power supply to the surrounding communities. The report does not clearly state who has to supply the power. The mine bearing this cost is likely to create precedent, and this option would not be seen as sustainable. Such a precedent has created issues with mining companies elsewhere.

One should be aware of the fact that villages will increase in size with time, and there will be a requirement to provide electricity to new residents. Table 25-3 illustrates how important the supply of power is to the community.

The community levy (0.5% of company revenue)[161] paid by SML, may be used to fund one or more social/community initiatives.

Table 25-3: Number of Engagements per Topics Covered

Topics Covered	Number of Interventions	Percentage (%)
Identification of people with their crops and land in the Study Area	9	11.1
Creation of conditions for the viabilityof the ecosystem	10	12.3
Creation of the conditions for the viability of the ecosystem	3	3.70
Delimitation of the area concerned by the mining lease	11	13.6
Local hiring	8	9.87
The reconversion of young indigenous gold miners	3	3.7
Connection to the power grid	21	**25.9**
Electrification of non-electrified villages in the localities concerned by the project	6	7.4
LMCI/SML compliance with its obligations to communities	10	12.3
TOTAL	**81**	**100**

In line with good international best practise, additional public consultation is recommended within the directly affected communities to provide feedback on the ESIA, as well as how their views and comments on the Project from the initial engagement were considered in the ESIA. Ongoing stakeholder engagement will be required throughout the LoM as per the ESMP, particularly with respect to meaningful engagement with affected communities.

[160] Expected to start Q1 2023.

[161] Discussed in Section 4 and 11 of the Technical Report and DFS respectively.

25.17.8 Soils

There is no evidence or indication of a Soil, Land Use and Land Capability Assessment being undertaken. No land use was mapped and assessed, which is important considering the reliance on agriculture in the area. No baseline soil samples were collected and thus soil fertility and a current baseline aren't known. This makes determining project impacts difficult.

25.17.9 Impact Assessment

The following impact assessments have not been done:

- The impact assessment has not assessed the updated layout of the site.
- No surface water impact assessment that investigates impacts of the dams.
- No heritage impact assessment.
- The impact assessment does not look at residual impacts, following implementation of prescribed mitigation measures.

25.17.10 Permitting

As per the environmental authorisation No. 00044/MINEDD/ANDE dated 18 February 2021, the following applies:

- Article 2 - The present authorization is granted to LMCI under the conditions that the company adheres to the recommendations formulated in the ESMP.
- Article 5 - In cases where ANDE observes a non-alignment/adherence with the environmental prescriptions formulated, it has the right to bring these to the attention of LMCI for corrective measures within 15 days. After expiration of the said 15 days, the following actions can be undertaken:
  - ANDE will implement the corrective measures and LMCI will bear the cost.
  - ANDE can legally suspend the development of the activities up until the corrective measures are undertaken.
  - ANDE can Definitely remove from LMCI, the environmental authorization.
- Article 6 - Any modification to the initial scope of the validated ESIA must be brought to the attention of ANDE.
- Article 7 - LMCI is held responsible for any environmental damage taking place outside the scope of the ESIA. LMCI will be subject to payment of a fine and will support all rehabilitation costs in line with the regulatory requirements in force.
- Article 10 - LMCI is required to inform ANDE about the start of the activities in order to enable ANDE to conduct environmental follow up as prescribed in the ESMP. LMCI is required to produce bi-annual reports on the implementation of the ESMP that will be addressed to ANDE.

25.18 Capital and Operating Costs

25.18.1 Mining CAPEX & OPEX

The QP responsible for the development of the CAPEX and OPEX estimates for mining, is of the opinion that based on the assumptions stated, the estimates are fair and in alignment with the estimate accuracy provision requirements for a DFS. If the tax assumptions made by the mining contractor are not realised in the Lafigué Mining Convention, then the capital and operating costs may change.

25.18.2 Plant and Infrastructure CAPEX

The QP considers that the CAPEX cost estimate as presented is to a DFS standard and no further work is required before moving to the front-end engineering design phase.

25.18.3 Process and G&A OPEX

Site laboratory costs are notably high, and it is recommended to tender this contract on the open market, since only one offer has been considered. Further, alternate business models should be considered including Endeavour funding and fitting out the facilities to the service providers specifications. This activity can be undertaken in the FEED phase, under the existing EPCM contract structure and thus no additional costs are required.

Labour operating practises need to be aligned with in-country regulatory requirements during the FEED Phase. This alignment process is a corporate cost and is not borne by the Project.

25.19 Economic Analysis

The economic model represents the culmination of all the key input assumptions outlined in the respective sections of the Report. Applying a long-term gold price of USD 1500/oz on a flat line basis to these assumptions, the Project delivers robust results over its 13-year mine life of mine, delivering an after-tax NPV5% (post tax) of USD 477 M on a 100% basis at a LoM AISC of USD 871/oz with a post-tax IRR and payback of 21% and 4.2 years respectively.

The sensitivity analysis shows that there is significant financial upside to the Project if gold prices were to stay at, or above, the long-term real price identified in Section 19. Furthermore, the operational sensitivity of the Project is in line with expectations, with relatively low sensitivity to capital and operating costs, but high sensitivity to movements in LoM head grade and gold price.

As the economic model relies on inputs from each of the disciplines outlined in the previous sections, there is a risk that each of the risks of the preceding sections could have a compounding impact on the results of the economic model. The following risks should be noted:

- Mining costs
  - The mining contractor has assumed that like SML, they will be VAT exempt during the construction phase. This is not a given and would need to be agreed in the Lafigué Mining Convention.
  - The mining contractor has assumed that Mobile equipment can be bought into CI on a temporary admission permit and after three years, duties will be paid on the amortised amount. Again, this is not a given, and would likely need to be agreed in the Lafigué Mining Convention and/or with the Customs Authority.

- Fuel Costs
  - Since Q1 2018 to Q2 2022, the diesel price to the Issuer's mines in CI has been range bound between (0.83 and 0.94) USD/L, whilst over the same period, the Brent Crude price has varied between (29 and 114) USD/bbl. (Section 19 of this Report). It is unclear whether this level of control can be maintained over the long-term, and thus some prices changes could occur.
  - The Project has used a long-term brent crude price of USD 73/bbl. and a corresponding diesel price of USD 0.91/L. It is important to note that there is virtually no correlation between the Brent Crude price in CI and the diesel price.
  - Assuming no diesel price controls in CI, the Issuer has estimated a free-floating diesel price range of USD 0.79/L (USD 60/bbl.) and USD 1.28/L (USD 98/bbl.). This would represent an overall OPEX decrease and increase of approximately -2% and +5% respectively (Figure 22-2).
- Labour
  - Owner's team labour costs at less than USD 10 M/a, are a relatively small component of the overall average annual mine OPEX[162], circa USD 170 M/a over years 1 to 12. Thus, minor changes in labour numbers and rates, are not going to have a significant impact on NPV.
- Taxes and Duties

  It has been assumed that the exoneration on the Business Patente Tax will continue after year three of 'Production'. If this is not agreed in the Lafigué Mining Convention, the associated implication on project economics is estimated to be an after-tax NPV impact of USD 22 M.
- Closure costs

  The closure cost estimate is conceptual in nature; is based on the PFS layouts/Mine Plan; and excludes the cost of labour retrenchment[163]. SML will need to update closure costs for the DFS as soon as practical. This revised value, will form the basis of the Closure Bond. The quantum of the change is in the process of being developed. It is noteworthy that only 20 per cent of the annual closure cost payment is paid into an escrow account, with the remainder held as a bank guarantee bond. The cost of the bank guarantee bond has not been incorporated in the financial model, on the basis that the terms will only be defined when the bond is in place.
- Sustaining Capital Costs
  - The sustaining capital costs for mining are based on tendered rates, with sustaining capital costs built in. No further opinion is offered in this area.
  - For Plant and Infrastructure, the sustaining capital costs applied in the financial model were not updated for the latest capital estimate and mine plan. Further, the life of the facility has increased from 11 to 13 years, and the Plant and infrastructure direct costs that are applicable to SML are of the order of USD 200 M. Applying factors by discipline cost (0 to 5%) and moderating for the short life of mine, results in a weighted average LoM sustaining capital cost of USD 2.8 M/a , as opposed to USD 1.3 M/a applied (down from the USD 1.5 M/a over 11 years allowed for in the financial model). It is considered that this delta of USD 1.5 M/a is within the overall estimate accuracy provisions of the DFS, and no further comment is provided.

[162] Mining, Process and G&A costs only

[163] For the DFS labour numbers, labour retrenchments costs are estimated to be USD 2.1 M.

Risk to the financial model include;

- Not meeting the construction schedule and/or not meeting the production ramp up schedule. The consequential impact on the financial model has not been defined.
- Either SML or the mining contractor not achieving their targeted assumptions with respect to taxes/duties. The Lafigué Mining Convention should be signed before the appointment of the mining contract, thus bringing clarity to the tendering process.
- Sustaining capital and closure will have some impact, but the impact is considered to fall within the accuracy provisions of the DFS financial results.

25.20 Adjacent Properties

Based on currently available information (exclusively on Turaco Gold's Eburnea Project), adjacent properties to the Lafigué Project do display evidence of gold occurrences as highlighted from AC and RC drilling. Gold mineralisation has been intercepted in Birimian volcano-sediments of the Oumé Fetekro greenstone belt, associated with zones of quartz veining, close to margins of dykes (Bouaké North Project) or within carbonate-silica altered fine-grained sandstone of the Birimian Comoé basin (Satama Project). As of the 'Effective Date' of this Report, none of these occurrences have been sufficiently drilled to define any Mineral Resources reported in the public domain.

25.21 Other Relevant Data and Information

25.21.1 Human Resources

Owner's team labour costs were based on another Endeavour mine in CI, with different shift working practises to that proposed for the Lafigue Mine. Once shift rosters are defined, the Owner's team labour cost estimate should be updated.

The legal framework presented solely relates to working hours; this needs to be expanded to cover all legal requirements and guidelines (government and intra-government).

Two shift roster systems have been proposed, both of which are in accordance with CI labour regulations. Further work is required to define whether there are any other shift rosters suitable, specifically taking into consideration where workers are to be sourced from, and their associated living/sleeping conditions.

The sourcing of skilled and unskilled Nationals, and the local/regional/national institutional capacity for training and development needs further development. This goes to what infrastructure and facilities are provided on/off site.

In summary, for both the Owner's team and contractors' operational staff, consideration should be given to:

- Using Endeavour's in-country and West African operations for training and staff selection.
- Defining local populations demographics (age, sex, religion, languages, tribal/race associations, and skills level (including functional literacy)).
- Defining community and government expectations.
- By functional role, defining where persons are likely to be recruited from.
- Defining which expatriates roles are to be phased out, how and by when.
- Defining cultural/religious considerations, and how this may inform facility design and operation.

- Defining which roles are likely held by women or disables persons (informs facility design).
- Defining HR conditions of employment for SML and contractors employees (required for setting up service level agreements (SLAs)/contracts).
- Identifying changes/updates required in existing HR/IR plans, to address the results of a local/regional base line HR assessment.

There is still sufficient time to address the aforementioned points. Further detail is provided in 'Recommendations', Section 24.5)

25.21.2 Project schedule

The execution schedule for the Project has been developed based on the project specific scope using logic driven activity links with a clearly identified critical path. The project schedule has been reviewed and benchmarked against similar recent projects, and is considered realistic and appropriate for a DFS.

26. RECOMMENDATIONS

26.1 Property Description and Location

The permitting/stakeholder register needs to be finalised and aligned to the Project development schedule. Further outstanding permitting and agreement items with government need to be resolved. The costs for this work is borne by Endeavour and not SML and thus, no costs need to be allocated to the Project.

As per Article 9 of the Mining Agreement, SML needs to notify the Minister of Mines, Petroleum and Energy that the LoM/Production schedule has changed. No costs are assignable to the Project for this activity.

As per Article 6 of the Environmental authorisation No. 00044/MINEDD/ANDE, SML needs to notify ANDE that the original scope of the ESIA has changed slightly, with minor consequential changes to the Closure costs required. It is notable that PR 329 is being renewed under the third exceptional renewal request, which is valid for a further two-year period. Whilst there should be no reason why this is not granted, it is incumbent on Endeavour to delineate further resources on this permit within the term or release the permit. Exploration is a corporate cost, and no costs are borne by the Project.

26.2 Accessibility, Climate, Local Resources, Infrastructure and Physiography

Recommendations for Section 5 are discussed more fully below.

- Climate

  The existing weather station on PE 58 should be upgraded and installed in compliance with World Meteorological Organisation (WMO) guidelines (Budget USD (10 000 to 20 000)). Specific focus should be on:

  - Rainfall/Rainfall intensity and evaporation measurement - Purpose: site seasonal water/salt balances and understanding extreme rainfall events
  - Insolation readings - Purpose potential for the installation of a photovoltaic farm at the mine or in the region.
  - Accurate wind direction and wind speed readings for dust and noise dispersion modelling.
  - Dry and coincident wet bulb temperature measurement.

  This upgrade is not currently budgeted for in the DFS or FEED Phase.

- Rail

  - As soon as practical initiate discussions with SITARAIL and others (fuel suppliers) around the possibility of providing the requisite rail infrastructure at Katiola to support the mine's operational logistics requirements. Whilst not an imperative, rail may provide some cost saving and reduce the CO2 footprint of the mine's logistics function. Rail could also be used for transporting non-resident nationals to their hometowns, as opposed to transport by road or air. This is a corporate function, and costs if any, should not be borne by Project.

- Power

  At least one and ideally two years of solar data should be collected[164] before any PV based solar system is considered for the mine or the area. Discussions should be initiated with CI Energies around the integration of any PV system with the grid. This activity does not form part of the DFS and hence no costs have been allocated for this exercise. There are some concerns with the cost effectiveness of solar, with the expansion of hydro and gas generation in CI, and appropriate pricing mechanisms/incentives will need to be agreed.

  Importantly, the provision of solar power should be seen as a means to minimise price escalation, minimise power disruptions (agreement with CI to minimise load shedding) and as a carbon reduction initiative, rather than as a supplementary power initiative for the mine.

  There are concerns that CI's base load capacity projections will not meet plans, and thus the Issuer (corporate cost and function) needs to undertake scenario planning exercises in this space, considering current petroleum reserves and new petroleum finds, along with the timelines required for development/commercialisation.

- Communications

  Discussions should be initiated with CI Energies and MTN/Orange for direct connection with RNHD's fibre backbone. With respect to CI Energies, it may be possible to install another OPGW on the transmission line from Dabakala to the mine, thereby allowing direct connection with RNHD's fibre connection at Dabakala. A budget has been allowed in the DFS for a Microwave link and any costs associated with a fibre line would likely be incorporated into a service level agreement between the mine and the service provider. Thus, no additional budgeting is required for the Project.

26.3 History

Prior to commencement of mining activities, it is recommended that close-spaced, advanced grade control drilling is conducted on an ongoing basis to refine more accurate estimates of historical depletion by artisanal mining activities. This aspect is covered in more detail in Section 14.

26.4 Geological Setting and Mineralisation

During the preparation of the MRE, SRK noted that some discrepancies between the logging of intrusive and extrusive forms of each rock type resulted in some localised inconsistencies in the lithological wireframes and recommended that these intervals be relogged and refined, for use in future iterations of the lithology modelling.

Further work reviewing the variety of controls on mineralisation towards Lafigué Centre would be beneficial in understanding the change in style and geometry of the mineralized bodies in this area, where mineralisation is typically thinner and less laterally continuous than to the north and east of the deposit. This will be most viable when the mineralized structures are exposed during the early years of mining, when more detailed pit mapping will be possible. The costs for this work, is discussed more fully in Section 14 of this Report.

26.5 Deposit Type

The QP considers that the deposit type is broadly understood, and thus there are no recommendations for Section 8.

[164] See weather station upgrades

26.6 Exploration

Although several assays returned from both RC and DD drilling samples from the exploration targets shown in Figure 9-3 have returned intercepts >1 g/t Au, the continuity of the mineralization remains unproven and further work will be necessary to develop some of these targets.

A complementary exploration drilling programme of 10 000 m (USD 1.5 M) should be implemented to define the potential of targets; WA01, WA03, WA08, Target 4, Target 9-11 and Central Area. This drilling programme forms part of Endeavour's Exploration Budget (corporate cost), and thus not costs are borne by the Lafigue Project.

26.7 Drilling

It is recommended that drilling activities be conducted on the six exploration targets highlighted for more detailed exploration work in Section 9. The same drilling methodologies and procedures used by LMCI/Endeavour at the project to date, should continue to be used for the future drill programmes. Endeavour considers exploration drilling costs a corporate cost, and as such, no costs are assignable to the Project.

26.8 Sample Preparation, Analysis and Security

Sample preparation, analyses and security are generally carried out to the required standard and as such, there are no recommendations to be implemented at this time.

26.9 Data Verification

26.9.1 Geology and Resources

No further data verification steps are planned or recommended at this time, however data should continue to be verified on an ongoing basis as it is collected.

26.10 Mineral Processing and Metallurgical Testing

Given the consistently high gold extractions from this ore and low cyanide and lime requirement, it is recommended to proceed with detailed process plant design and engineering.

No further testing is recommended for the Project. Gold assay variability is inherently high in high gravity gold orebodies and further testing will not improve consistency of outcomes or provide improved recovery data.

A grade control regime including sulphur assays in the ore zones is recommended for the mine. Spikes in the sulphur grade are clear indication of slower leaching character, and the need for lead nitrate and/or higher free cyanide levels. Bottle roll leach testing of incoming ore composites with leach profiles should also be routine with additional testing being triggered by increased sulphur grades. Costs associated with these activities are included in the mine's operating cost expenditure, and as such, do not form part of any forward work activity.

Process guarantees are always conditional and vendor dependant. This makes application of any penalties difficult and there is more merit in doing appropriate due diligence in advance of contracting a vendor to be confident that reputational damage is of greater concern to the vendor than any penalties that may be levied.

26.11 Mineral Resource Estimates

SRK recommend the following actions be undertaken to improve the confidence in the geological model and quality of the Mineral Resource estimates in future:

- The addition of some close-spaced exploration drilling or pre-production RC grade control drilling will be useful to investigate grade continuity in more detail. This is particularly the case in areas of the model where the continuity of the mineralisation wireframes is significantly supported by relatively low-grade composites, and where the continuity of mineralised structures reduces significantly in the Lafigué Centre area. Pre-production grade control drilling is planned at a 10 m x 10 m spacing for the first six months of mining in two of the open pits prior to the start of production. The total pre-production grade control cost is estimated at USD 2.13 M based on drilling and external assaying costs. Labour is not included in this budget. To complete the initial grade control models in advance of the start of mining, drilling is planned to start by May 2023. This will provide four months for drilling with two rigs, with sufficient assay turnaround time (one to two months) for generating grade control models, and additional time for engineering and mine planning before January 2024.
- Although Lafigué has not been mined on a commercial scale, SRK note that there has been significant artisanal mining activity at the site. SRK has depleted the declared Mineral Resources using elevation data obtained by a drone survey conducted by Endeavour the Client on 17 August 2021, and to a depth considered broadly appropriate for the average depth of artisanal workings at the time of the SRK site visit. However, in the absence of a detailed survey of these workings, this presents a potentially significant risk to the early stages of the mine plan. Where possible, a detailed survey and/or mapping of any workings not destroyed by the bulldozing of the site is recommended to be completed on an ongoing basis as overburden stripping and mining commences during the earliest periods of the mine life.

26.12 Mineral Reserve Estimates

26.12.1 Geotechnical

Infill geotechnical drilling in DS8 and DS9 to verify the required design criteria within the Fresh rock. This work is to be done before mining starts, and the costs are already included in the estimate.

26.12.2 Mining

Mining recommendations include:

- Interim pushback designs should be investigated, but this will be an iterative process and needs to be integrated into the LoM production schedule. This can be done inhouse or allocated to a consultant (a budget of USD 25 000 should be allowed).
- Dilution and losses are to be re-evaluated when the mining fleet is confirmed. This can be done inhouse or allocated to a consultant (a budget of USD 5 000 should be allowed).

26.13 Mining Methods

26.13.1 Hydrological

26.13.1.1 Updated Evaluation of the Structural Geology Model

Further evaluation of the structural geological model should be undertaken before additional testwork is undertaken, so that structural controls on groundwater behaviour are fully assessed and associated risks are identified accordingly. This should include a review of the structural interpretation performed by Bastion (2021), including the two major shear zones identified, and a review of the hydrogeological characterisation and implications.

26.13.1.2 Additional Hydrogeological Testwork

Further test work should be undertaken to improve the hydrogeological characterisation of the site. Existing borehole infrastructure should be used where possible but include two additional boreholes. Techniques such as packer testing should be employed to determine hydraulic conductivity across the larger geological structures. In addition, pumping tests should be undertaken in all groundwater monitoring wells that have been installed (where not dry). Some investigation of the hydrological characteristics and behaviour of the saprolite is also recommended aligned with the geotechnical slope stability assessment. VWPs should be installed before the above testwork, particularly pumping tests, so that VWP sensors can be used as groundwater pressure monitoring points during testwork. The costs for said testwork, is discussed more fully in the Section 20 of this Report.

26.13.1.3 Update of Numerical Groundwater Model

The groundwater modelling should be updated to accommodate the results of the further characterisation studies (as above). Sensitivity analysis should be performed to understand likely maximum and minimum inflows to pits. The design of pit water management measures should be reviewed according to a more informed risk-based analysis of predicted groundwater inflows and behaviour. Pore pressure profiles should be similarly reviewed and cross-checked by the geotechnical team to ensure there are no implications for slope stability risks. The costs for said testwork, is discussed more fully in the environmental section of the Report.

26.13.1.4 Effectiveness of Horizontal Drain Holes

Before any detailed design of a horizontal drain programme, the effectiveness of drain holes on reducing pore pressure should be tested with a pilot programme targeting VWP installations such that piezometer responses can be monitored. If it is not feasible to monitor the impact of drain holes using the existing monitoring network, then additional VWPs should be installed both directly above and laterally offset by 20 m and 50 m from the drain holes.

26.13.1.5 Review of Surface Water Management Designs

Given the potential requirement for access routes, surface water management proposals should be reviewed following a final design and infrastructure layout development.

26.13.1.6 Pore Pressure Monitoring

A robust network of VWP must be installed and maintained throughout the mine life. Ideally, there should be at least one active piezometer string behind the slope for each design section. The purpose of the monitoring program is:

- To ensure that general depressurization of the slopes is occurring, pore pressures are equivalent to or lower than the pressures used for the slope designs.
- To monitor pore pressures across any primary or secondary faults or dykes identified as a higher risk for slope stability.
- To assess the effect of recharge from rainfall events on pore pressures in the longer term.

A detailed VWP design should be provided once critical geotechnical sectors are identified.

26.13.1.7 Management of Dewatering Programme

The operation of a successful water management system for the Lafigué pit complex will require a designated team on-site to plan the dewatering programme, track its progress and maintain its functionality as the pit expands.

26.13.1.8 Cost Estimates for Recommendations

Provisional cost estimates for the above recommendations are included in Table 26-1. Table 26-1 is in part superseded by the costs allowed for in Section 20 of this Report.

Table 26-1: Provisional Cost Estimates for Pit Hydrogeology Recommendations

Recommendation	Description	Cost Estimate (USD (k))	Comment	Timing
Updated Evaluation of the Structural Geology Model	Consultant Study	15		FEED/detailed design stage
Additional Hydrogeological Testwork	Hydrogeological drilling and testing fieldwork, including supervision and analysis of results	50	Drilling contractor costs not included	FEED/detailed design stage
Update of Numerical Groundwater Model	Consultant study	30		FEED/detailed design stage
Effectiveness of Horizontal Drain Holes	Pilot programme	Operational cost		During operations
Review of Surface Water Management Designs	Consultant study	15		FEED/detailed design stage
VWP Installation for Pressure Monitoring	Drilling and VWMP installation	60	Drilling contractor costs not included	FEED/detailed design stage
Management of Dewatering Programme	Technical support during operations	Operational cost		During operations

26.13.2 Geotechnical

BG recommend the following work programme to ensure that the inherent uncertainties and variabilities in the geotechnical model are covered:

- Development of a Ground Control Management Plan (GCMP). This should be updated on an annual basis and will fall under operating cost.
- Development of a 3D large structure model through the acquisition of mapping data. This will be an ongoing operational process and will fall under operating cost.
- Re-logging of the core to update the fabric model in addition to the mapping of benches when exposed. This can be done during operation using mapping data and will fall under operating cost.
- Targeted pore pressure monitoring of the north-eastern stacked footwall slopes of Stages PFS stage four to eight. This will be done during operation and will fall under operating cost.

26.13.3 Mining

Mining recommendations are as noted below:

- Advanced-grade control drilling is required to improve confidence in the short-term plan by increasing Resource classification to Measured. An advance grade control programme is already planned, and USD 450 k has been budgeted and included in the estimate.
- A medium-term plan of at least three years should be regularly updated to ensure waste stripping is according to plan, tonnage and spatially. This should be done by the operations team rather than a consultant.

26.14 Recovery Methods

The QP for Section 17 considers that the Plant has been developed in accordance with CIM best practise guidelines for mineral processing, and the level of technical development undertaken over the course of the DFS, is suitable to progress directly to the Front-End Engineering Design (FEED) Phase. On this basis, no recommendations/forward work programme activities are applicable to Section 17.

26.15 Project Infrastructure

26.15.1 Geotechnical

26.15.1.1 TSF

A more detailed topographical survey is required to map the steeper rock outcrop areas. Alternatively, it is recommended that the rock outcrops areas be drilled and blasted prior to construction to provide a source of rock material for the project. Provisionally the ground below the topsoil layer over approximately 10% of the TSF, may be unsuitable to scarify and re-compact to form a smooth HDPE subgrade. A soil capping layer sourced from local borrow will be required. The cost estimates for the TSF include consideration of these findings.

26.15.1.2 Plant

During detailed design, consideration will need to be given to measures to reduce foundation settlement to meet structural and operational requirements. These include:

- Undertaking settlement calculations using more detailed loading of the different stages of construction.
- The opportunity to preload some structures, where practicable. For example, the CIL tanks can be preloaded with water (hydraulic testing undertaken for a longer period).
- Replacing some of the nearer surface ground below specific foundations with compacted rockfill to increase the stress-strain modulus and reduce settlement.

The cost of this work is within the FEED phase EPCM budget.

26.15.1.3 Aggregate Assessment

More detailed testing of the Koundougou sand should be undertaken to confirm suitability. Costs for this work is included in the FEED phase EPCM budget.

26.15.2 Earthworks and Site Preparation

Whilst provisional topsoil volumes have been defined, the storage locations have not. This needs to be addressed early in the FEED Phase and is covered in the EPCM budget.

26.15.3 Roads and Airstrip

The DFS design of the site access roads and haul roads presented herein is suitable for advancement to detailed design and construction with no additional work/costs required.

The DFS design and geotechnical investigation of the site airstrip presented herein is suitable for advancement to detailed design and construction with no additional work required, subject to any amendments required by relevant authorities during the permitting process.

26.15.4 Mine Services Area (MSA)

During the next stage of FEED phase negotiations, more work is required to establish the battery limits and associated responsibilities between the mine owner and the mining contractor.

26.15.5 Emulsion Plant and Explosives Storage

During FEED phase negotiations, Endeavour/SML will discuss in detail, the approach to the supply of explosive's and emulsion, and the associated facilities and services required. The cost for this activity falls within owner's project development man-hour costs (pre-production Capex), with the facility requirements fully detailed during tendering stage.

26.15.6 Power

The conceptual designs have been completed and approved by CI-ENERGIES. Detailed design can commence immediately, with costs accounted for within the EPCM budget.

26.15.7 Tailings and Water Management

The TSF is designed to accommodate a total of 41 Mt (db) of tailings based on the Sc12I mining schedule. It is noted that the current mining schedule (Sc13k) requires an additional 6.7 Mt of tailings; however, the additional tonnage occurs during the final years of operation. The current mining schedule can be incorporated into the detailed design of the TSF at no additional cost over and above the engineering costs included in the study estimate.

The detailed design of the TSF shall include incorporation of the Global Industry Standard on Tailings Management (GISTM, 2020). As this document is heavily focussed on governance and operation of TSFs, it is not expected that this update will significantly impact the design of the TSF presented within this document. This update can be completed during the detailed design at no additional cost over and above the engineering costs included in the study estimate.

Ongoing monitoring of the TSF, WSD and WHD monitoring instrumentation shall be undertaken during operation as required by Operating Manual of each structure (to be provided prior to commissioning), including routine auditing. Monitoring and auditing costs should be included within the standard operating cost estimate. It is recommended that TSF monitoring bores are installed during the early stages of construction to facilitate collection of baseline readings.

The dataset utilised for the baseline climatic assessment is sufficient for the study; however, installing an automated weather station at the project site will allow the data to be calibrated to the site during the project. An allowance for a weather station has been included in the project capital cost estimate.

The DFS design and geotechnical investigation of the WHD and WSD presented herein is suitable for advancement to detailed design with no additional work required, subject to any amendments required by relevant authorities during the permitting process.

The water balance modelling should be updated as part of the detailed design to reflect the current mining schedule and any modified process parameters. This update can be completed during the detailed design at no additional cost over and above the engineering costs included in the study estimate.

Any modifications to the WSD design resulting from the aforementioned water balance update can be incorporated into the detailed design of the WSD at no additional cost over and above the engineering costs included in the study estimate.

Any modifications to the WHD design resulting from optimisation studies completed during these latter stages of the study and the aforementioned water balance update can be incorporated into the detailed design of the WHD at no additional cost over and above the engineering costs included in the study estimate.

The DFS design of the SCSs presented herein is suitable for advancement to detailed design with no additional work required, subject to any amendments required by relevant authorities during the permitting process.

Any modifications to the SCS can be incorporated into the detailed design of the SCS at no additional cost over and above the engineering costs included in the study estimate.

26.15.8 Waste Rock Management

The WRDs have been developed to a DFS level of design and no further work or costs are required to move to the FEED phase, over and above what is currently allowed for in the EPCM budgets.

26.15.9 Balance of Infrastructure

The balance of infrastructure developed and costed by Lycopodium is at a DFS level of development and no further work is required to proceed to the FEED phase. All FEED phase activities are covered in the EPCM budget.

26.16 Market Studies and Contracts

26.16.1 Marketing

Endeavour will continue to use consensus market data (CMD) for commodity price assumptions while benchmarking against peers. This data is largely sourced from corporate subscription services (no cost to SML), and no further work or costs are required in this area.

26.16.2 Contracts

The hybrid business model to be employed at the SML mine needs to be further developed/refined, specifically with respect to how services and facilities are to be utilised and shared between mine stakeholders and the associated charging basis. In setting up said business model, consideration will need to be given to:

- the tax provisions agreed in the Lafigué mining convention when signed;
- human resources requirements, local labour sourcing and development, and the employment of woman;
- social development requirements, specifically local sourcing/procurement, and the development of local businesses;
- environmental requirements and standards;
- the size, local capacity, and strengths and weaknesses of each contractor;
- minimising the duplication of roles across the mine, where there is no good rationale for doing so; and
- leveraging the Group's buying power, to negotiated better terms based on economies of scale in-country.

The costs for aforementioned activities is based on utilising existing Endeavour (no charge), and SML employees (covered within 'Owner's costs') and capitalised as pre-production costs.

26.17 Environmental Studies, Permitting and Social or Community Impact

The 2021 ESIA report contained an assessment of the biophysical and socio-economic environment which enabled a characterisation of the Lafigué Study Area. However, further work is recommended to better ascertain the existing environment and ecosystem services which will be affected by the development. This will also allow for a refinement of the prescribed mitigation and management measures in an effort to ensure that irreversible adverse consequence to the environment and affected communities are avoided, in line with the principles of sustainable development.

Table 25-3, following, details the gaps identified in the current Environmental and Social (E&S) studies and recommended actions to address these gaps. It also outlines when the proposed forward work programme activities should be undertaken. Phases are as defined below:

- Prior to FEED Phase (PFP);
- FEED Phase (FP); and
- Operational Phase (OP).

Table 26-2: Forward Work Programme

Aspect	Identified Gap and Justification	Recommended Action	Timeframe	Cost Estimate	Imperative/ Phasing (NOT Working to IFC Requirements)
Terrestrial Biodiversity Studies	No habitat delineation was completed for sensitive habitats which is required in the BMP.	Undertake an additional flora site assessment to complete a habitat delineation of sensitive habitats and to locate the endangered species newly listed in the IUCN Red List.	3 Months (1 wet season supplement survey if IFC compliance is not required)	USD 22 000 (1 survey)	FP
Freshwater ecosystem Assessment	A wetland delineation was not undertaken due to ASM. Only one site assessed due to ASM for the aquatic sampling with no mine related impacts assessed. This is an important consideration to assess the potential changes from the dam, as well as the potential changes to the flow regime during the dewatering phase.	Conduct a wetland delineation. A freshwater ecosystem study is recommended which would include aquatic and wetland areas as well as an Ecological Water Requirement study.	3 Months (1 wet season supplement survey if IFC compliance is not required)	USD 25 000	FP
Critical Habitat Assessment	Potential presence of several threatened species may trigger natural and/or critical habitat with associated reputational, license to operate, access to funding and financial risks.	Undertake a critical habitat assessment.	2 months	USD 30 000	FP
Air Quality	No evidence of a detailed emissions inventory and dispersion model to inform the quantification (intensity and extent) of impacts associated with fugitive dust and volatiles emissions was found. An understanding of the extent and intensity of the impact need to form the basis of quantifying and prescribing management measures for the Environmental and Social Management Plan (ESMP).	Undertaken an Air Quality Impact Assessment (incl. construction and operational dispersion models) to update the Impact Assessment and mitigation measures.	2 months	USD 12 000	FP

Table 26-2: Forward Work Programme

Aspect	Identified Gap and Justification	Recommended Action	Timeframe	Cost Estimate	Imperative/ Phasing (NOT Working to IFC Requirements)
Noise	A Noise Study was conducted . However, no evidence of a detailed noise source inventory and dispersion model to inform the quantification (intensity and extent) of impacts associated with increased and cumulative ambient noise levels was found. An understanding of the extent and intensity of the impact need to form the basis of quantifying and prescribing management measures for the ESMP.	Undertake a Noise Impact Assessment (incl. construction and operational day-time and night-time dispersion models) to update the Impact Assessment and mitigation measures.	3 months	USD 12 000	FP
Ground vibrations	Blasting will be required according to the ESIA. A detailed Blasting and Vibrations Assessment is required.	Undertake a baseline structure profile analysis within a 500 m buffer of the mining area which will be referenced to address any grievances relating to ground vibrations and conduct a blasting assessment.	6 weeks	USD 4000 (structure profile), USD$ 6500 (blasting assessment)	FP
Land Use	Given the importance of agricultural practises in the area, the loss of livelihood based on agriculture should be considered during all phases of the project and related mitigation measures should ensure the maintenance of existing means or development of alternative options. Currently there is no soil studies, no baseline or fertility studies.	Undertake a soil and land use assessment.	8 months	USD 9000 (first season); USD 6000 (second season)	FP
Hydrogeology	A groundwater model was developed in May 2021 on a pre-feasibility level for pit water management during operation. The study did not investigate post-closure rebound rates and decant predictions.	The following hydrogeological activities are recommended for a groundwater impact assessment and mitigation planning:	6 months	USD 50 000	FP

Table 26-2: Forward Work Programme

Aspect	Identified Gap and Justification	Recommended Action	Timeframe	Cost Estimate	Imperative/ Phasing (NOT Working to IFC Requirements)
	In addition, no mine-related impacts on the groundwater environment were conducted. Pit dewatering will lower the water table, which could affect nearby private boreholes and streams. Such impacts need to be quantified considering radius of influence and impact duration. The proposed TSF and waste rocks could also leach and contaminate the aquifer. The contamination plumes originating from the mine infrastructure needs to be predicted during and after mine closure, and proper mitigation measures need to be put in place. During the pre-feasibility study, only five percussion boreholes were drilled within the pit footprint area. These, however, are not sufficient for impact assessment. Monitoring boreholes need to be sited taking the source-pathway-receptor dynamics of the Lafigué site. Regular monitoring of surface and groundwater conditions needs to be conducted starting before mine construction for baseline reference. This is also assist in ascertaining existing impacts from historic ASM activity in the permit areas.	Aquifer characterisation and monitoring borehole drilling. The boreholes need to be sited strategically considering mine infrastructure and geological structures. To site borehole along water-bearing fractures, geophysical surveying needs to first be done. Ground geophysical survey, particularly Electrical Resistivity Tomography (ERT) survey and ground magnetic survey. ERT is a preferred method to identify fracture zones that are preferential pathways. Available air-borne magnetic survey map will be needed to delineate dykes which often control groundwater movement. Aeromagnetic maps are good starting point, however, magnetic survey identifies structures with only magnetic anomalies, such as dykes. Water has no magnetic property and water-bearing fractures are better interpreted using resistivity surveys. This information should be integrated with the magnetic data as well as geological information to site boreholes for rock permeability assessments. Hydrocensus: This is needed to understand the extent of groundwater users and surface/groundwater interaction and is important for future liability study. The background water quality consisting of full-suite analysis is needed, particularly focussing on high-risk monitoring points. Borehole Drilling: Dewatering and aquifer characterisation boreholes should be drilled at fracture areas, following the geophysical results. The PFS hydrogeological study recommended for the drilling of 9 dewatering boreholes – consisting of 3 in-pit and 6 out-pit. However, the previous model that was developed with limited hydrogeological, structural and geological data and needs to be updated, to further refine the number, location and design of the dewatering boreholes.		Cost excludes: Laboratory cost for water quality analysis USD 200 per sample Borehole drilling cost approximately USD 20 000 per borehole	

Table 26-2: Forward Work Programme

Aspect	Identified Gap and Justification	Recommended Action	Timeframe	Cost Estimate	Imperative/ Phasing (NOT Working to IFC Requirements)
		Aquifer Testing All the dewatering boreholes should be aquifer tested. Any borehole with blow yield of 0.5 L/s or less should be used as part of the monitoring network, but not dewatering. The permeability of these boreholes is too low to be used for pumping. Such boreholes only need to be slug tested. Any borehole with yield of more than 0.5 L/s should be subjected to pump tests. Initially a step test should be done for 2 hours, each step being 30 minutes long. This should be followed by a 24-hour constant rate test Packer test should be done in at least 6 exploration holes to profile the permeability of the geology though depth. A packer test is required to profile the permeability along each fracture zone and its significance for groundwater ingress. Conceptual model updating: The conceptual model needs to be updated taking the source-pathway-receptor dynamics of the aquifer system. The groundwater information obtained from the above activities should be integrated with the geological structural model to update the hydrogeological model. Numerical Modelling: The existing numerical model needs to be updated considering site specific hydrogeological conditions. A more reliable and accurate inflow rates will then be obtained. The model will also be able to estimate the potential environmental impacts with acceptable accuracy		Aquifer test contractor cost approximately USD 10 000 per borehole	

Table 26-2: Forward Work Programme

Aspect	Identified Gap and Justification	Recommended Action	Timeframe	Cost Estimate	Imperative/ Phasing (NOT Working to IFC Requirements)
Social/Stakeholder Engagement	Public consultation was conducted as part of the regulated ESIA process following the compilation of the draft ESIA. In line with good international practise an additional public consultation exercise is encouraged to illustrate how the views and concerns of stakeholders were considered in the final approved ESIA.	Undertake a public consultation exercise in the Dabakala Department to provide feedback on the stakeholder considerations in the final ESIA. During this exercise, information on the layout changes can also be shared.	2 weeks	USD 5 000	PFP
	Population influx due to speculative job seekers and/or opportunities for compensation is expected within the PE 58 permit and broader local area, especially given the presence of ASM. The development of an Influx Management Plan is a useful tool to manage influx into the local area as well as the knock-on effects on public infrastructure and services.	Develop an Influx Management Plan in conjunction with local authorities to manage the impact of the expected population influx into the local area.	6 weeks	USD 15 000	PFP
ESIA/ ESMP Report	Update of ESIA document with additional information	Provided the gaps identified above the ESIA and ESMP should be updated to incorporate the outcomes of this additional work.	3 months, Note: following completion of the above recommendations	USD 20 000	OP

26.18 Capital and Operating Costs

26.18.1 Mining

Mine capital and operating costs should be updated once the Lafigué Mining Convention is signed and the tax basis for the mining contractor finalised. This is considered a corporate cost and not borne by the Project.

26.18.2 CAPEX Balance of Facilities

The QP considers that the capital cost estimate, outside of mining, is in alignment with requirements of a DFS, and no further project development work is required.

26.18.3 OPEX Balance of Facilities

The QP considers that the operating cost estimate, outside of mining, is in alignment with requirements of a DFS, and no further project development work is required.

26.19 Economic Analysis

Noting the economic model is simply a snapshot in time, as at the Base Date, it is recommended that the economic model be updated on a regular basis. The updates should consider any significant operational updates and/or market updates. This work would be considered under normal operating procedures for a mining company, and as such, would not require any additional budget allocation.

Importantly, the mining model needs to be updated once the terms and conditions of the Lafigué Mining Convention are defined. This is a corporate cost, and the cost of this update is not borne by the Project.

26.20 Adjacent Properties

Any joint venture/toll treatment potential with the holders of the Permits adjacent to PR 329 as described herein, are outside of the scope of this Report. Further, no Mineral Resource Statements have prepared for the properties reviewed.

Notwithstanding this, developments on other properties should be followed and any opportunities arising should be investigated. No budget is required for this exercise.

26.21 Other Relevant Data and Information

26.21.1 Human Resources

In order to move seamlessly from the study phase into project execution and operations, it is recommended that a thorough local/regional base line assessment be undertaken and company policies and plans be updated, to ensure that the learnings from the baseline assessment are incorporated into; the HR strategy, facility design, operating basis and service level agreements. Whist a number of the requirements outlined in Table 26-3 have been undertaken, further work is still required.

The cost for the activities outlined in Table 26-3 are either covered by corporate budgets and/or within the Owner's team man-hour budget presented in the 'Report's' CAPEX estimate.

Table 26-3: HR/IR Recommendations

<table>
<tr><th>Functional Requirement</th><th>Functional Requirement Description</th></tr>
<tr><td>Base Line Assessment</td><td>As part of a base line assessment SML should where applicable identify and state:
• relevant guidelines/laws and statutes to be adhered to; including consideration of IFC, Performance Standards 2, 4, 7 & 8 (2012), direct linkage to ILO requirements to which the country is a signatory;
• government legislation/policies (current/proposed), including localisation policy and commitments made in any agreements signed;
• key external stakeholders:
– national, provincial & district - labour/education/mining/social security/health, …),
– tribal/community leaders,
– organised labour bodies, NGO's and financial institution (employee salaries)
• legal requirements/expectations of each stakeholder in a matrix format; services and facilities that may be required by employees and their dependants (nationals and expatriates, education (schools, trade schools, tertiary institutions), medical services (clinics, hospitals,…), shops, accommodation (residential and hotels) and other.
• in combination with contractors, the owner's team and consultants should prepare a detailed analysis of the skills required to construct & operate the proposed mine and comment on where they are going to come from (locally (villages and towns), nationally & internationally) and how they are going to be retained/attracted (training, poaching and timelines);
• define conditions of employment applicable, benchmark against other companies competing for the same resources;
• define community development employment obligations;
• define HR Impact on local communities & region;
• define local/provincial/national cultural/educational issues that may impact the development of an HR Management Plan, Specific focus given to; tribal structures; key role players; ethnic groups; culture; customs/religion; language; woman; people with disabilities; youth; migrant communities; indigenous people and elderly; and
• labour productivity (benchmark), including historic data, including reasons for industrial unrest and outcomes.</td></tr>
<tr><td>Company Policies and Plans</td><td></td></tr>
<tr><td>Human Resources and Industrial Relations Policies</td><td>In accordance with the baseline assessment, define relevant Human Resources (HR)/Industrial Relations (IR) policies. For IR Consider: the basis for employment of all people on the mine, irrespective of employer; industry best practise, collective bargaining, grievance/communication mechanisms, agreement of shared conditions of employment/services between persons working on the mine and sharing of HR data between SML/contractors, in accordance with company objectives.</td></tr>
<tr><td>General Employment Conditions</td><td>Define general employment conditions, including; working hours (day work and shift work, including points of clock in/out), leave cycles, public holidays; general employee benefits, general operations policies including police/security clearance certificates; security searches; polygraphs; drugs and alcohol testing; induction, annual & exit medicals; STD/HIV testing and treatments.
In the determination of working hours and rosters for residents and non-residents, consideration must be given to:
• whether non-local nationals should be employed on a residential married status;
• where local people employed live, and whether travel time or other conditions impact the employees ability to function safely and efficiently during working hours; and
• the impact of nationals and expatriates on rostered leave, and the impact on staffing numbers.
Consideration shall also be given to:
• Occupational hygiene facilities provided at the mine, including; change rooms and a laundry service.
• Meal requirements (consideration of local, religious requirements)</td></tr>
<tr><td>Performance Management and Compensation</td><td>Define basis for rewarding employees, with respect to company and personal performance KPIs and staff retention strategy. Define basis for staff churn by level of practise. Should cover all persons employed on the mine, whether directly or indirectly employed.</td></tr>
</table>

Table 26-3: HR/IR Recommendations

Functional Requirement	Functional Requirement Description
Labour Sourcing and Recruitment	In accordance with the base line assessment, identify where persons are going to be sourced from and type (skills, nationality, local, regional, tribal), impact of competition between local/regional employers, technologies and skill sets required, local recruitment practises/protocols and entities to be used (private, government, advertising); localisation requirements, likely age, sex, literacy rates and experience level of persons employed. Focus will be on both construction and operations. The output should be a detailed plan with defined costs (CAPEX and OPEX).
Human Resource Development and Training	For mine onboarding and operations, define how training is to be undertaken, facilities and organisational resources required on/off site. Define training budget and development basis. The output should be a detailed plan, including: • internal/externals persons required; • whether other company mines in CI or West Africa can be used • facilities internal/external to the site required; and • training schedule and costs.
Retrenchment Closure Planning	Define retrenchment/closure planning basis and potential impact on the local community. Also consider termination of service level agreements on mine and treatment of service providers employees. The output should be a preliminary plan, refined over the Life of Mine.
Stakeholders/Stakeholder Reporting Requirements	Define internal and external stakeholders and associated reporting requirements, the output should be a detailed costed plan.
HR/IR Information Management	Define HR management systems to be employed from execution through to operations, including system structure (head office; regional office; mine ; contractors); Cost (CAPEX/OPEX); interface with other systems (security/access control; time and attendance; payroll; medical; information management; training, performance management, incident reporting, banking, etc.).
HR/IR organisational Structure	Outline HR/IR function, structure, facilities and resources required to deliver mandate, considering shared corporate services if applicable. Output forms input to the Operations Management Plan.
Company Labour Costs	Define company labour cost, specifically with respect to developing a total cost to company (TCTC). Labour costs will be built up by band/level of practise and by nationality. Consideration will be given to; statutory payments; bonuses; share options, insurances (death/disability), company self-insurance requirements, pensions, medical, travel (local, regional and international) and visas, family benefits, staff churn, shift allowances, accommodation, schooling, vehicle allowances, hardship allowances, medical testing, and relocation, recruitment and termination allowances.
Company Labour Ratios	In accordance with the detailed manning lists presented by year in the operations management plan, define ratios/metrics and compare against regulatory requirements, locals/international benchmarks and company policies. Ratios will consider persons employed by sex, by band and by nationality as a whole, and against operating metrics per ROM t, per unit of product produced and/or other. The manning lists will show: • the phasing out of expats and the training and transfer of roles from expatriates to nationals; • nationals split by: local (live at site) and non-locals (live elsewhere, ideally defined by key locations); • employment of woman; • employment of special interest groups as defined by government or other guidelines/standards; and • the onboarding of apprentices.
Risk Analysis	A full risk/opportunity assessment must be undertaken for construction and operations and be reported accordingly.

26.21.2 Project Implementation

Project planning is a key component of the overall project management process and the successful delivery of a project on time and within budget. An important requirement during the early stages of the project implementation phase will be to establish a Project Master Schedule (Level 3 minimum detail) for monitoring and controlling project activities against an approved baseline.

To minimise the overall project implementation schedule, it is recommended that critical path activities and long lead time procurement packages (such as Ball Mill and HPGR equipment) be committed to as early as possible, ahead of full project funding approval.

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28. DATE AND SIGNATURE PAGE

This 'NI 43-101 Technical Report, for the Issuer's 'Lafigué Project, Côte d'Ivoire, with an 'Effective Date', and a Publication date of '11/30/2022', was prepared and signed by the following Authors.

Name: Degree and Professional Association: Company: Signature: Date:	Alex Veresezan MSc (Mining Engineering), PEng Endeavour Mining plc (Endeavour) (signed and sealed) 11/30/2022
Name: Degree and Professional Association: Company: Signature: Date:	David Morgan MSc (Irrigation Engineering), MIEAust CPEng, MAusIMM Knight Piesold Pty Limited (KP) (signed and sealed) 11/30/2022
Name: Degree and Professional Association: Company: Signature: Date:	Geoff Bailey BEng (Electrical Power), FIEAust, CPEng, NPER-3, REPQ ECG Engineering Pty Ltd (ECG) (signed and sealed) 11/30/2022
Name: Degree and Professional Association: Company: Signature: Date:	Silvia Bottero MSc (Geological Sciences) Exploration Mining plc (Endeavour) (signed and sealed) 11/30/2022
Name: Degree and Professional Association: Company: Signature: Date:	Stuart Thomson MEng (Chemical & Materials Engineering), FSAIMM Endeavour Mining plc (Endeavour) (signed and sealed) 11/30/2022
Name: Degree and Professional Association: Company: Signature: Date:	David Taylor BEng (Mech), CPEng, FIE(Aust) Lycopodium Minerals Pty Ltd (Lycopodium) (signed and sealed) 11/30/2022
Name: Degree and Professional Association: Company: Signature: Date:	Abraham Buys NHD (Extraction Metallurgy), FAusIMM Lycopodium Minerals Pty Ltd (Lycopodium) (signed and sealed) 11/30/2022

Name:	Lucy Roberts
Degree and Professional Association:	BSc (Exploration Geology), MSc (Mineral Resources) PhD (Applied Geostatistics), AusIMM(CP)
Company:	SRK Consulting (UK) Ltd (SRK)
Signature:	(signed and sealed)
Date:	11/30/2022
Name:	Francois Taljaard
Degree and Professional Association:	PR Eng, BEng (Hons) (Mining and Technology Management), SAIMM, MIMMM
Company:	SRK Consulting (UK) Ltd (SRK)
Signature:	(signed and sealed)
Date:	11/30/2022
Name:	Graham Trusler
Degree and Professional Association:	MSc, Pr Eng, MIChE, MSAIChE
Company:	Digby Wells and Associates Pty Ltd (DWA)
Signature:	(signed and sealed)
Date:	11/30/2022

CERTIFICATE OF QUALIFIED PERSON

I Alex Veresezan, M.Sc., P.Eng. do hereby certify that:

1. I am a Group Manager - Mining Contracts at Endeavour Gold Corporation, Mourant Governance Services (Cayman) Limited Corporate Centre 94 Solaris Avenue, Camana Bay, PO Box 1348 Grand Cayman KY1-1108, Cayman Islands.
2. I am a co-author of the NI 43-101 Technical Report titled 'Lafigué Project, Definitive Feasibility Study'. with an 'Effective Date' of 1 June 2022.
3. I graduated with a degree from the University of Petrosani, Petrosani, Romania, with a M.Sc. in Mining Engineering (1993).
4. I am a Registered Member in good standing of the Professional Engineer Ontario (Membership Number: 100078587).
5. I have worked as a mining engineer for a total of 29 years since graduation, eight years of which have been as a mining consulting engineer with TeraTech, P&E Mining Consultants, ERM/CSA Global Consultants, and RPM Global.
6. I have read the definition of 'qualified person' as set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a 'qualified person' for the purposes of NI 43-101.
7. For this Technical Report, I am responsible for the preparation of Sections/Subsections: 1.20.1, 12.9.5, 12.9.6, 18.7.2, 18.7.3, 18.10.3, 18.10.4, 21.2.8.2, 21.3.1, 21.3.2.1, 21.3.3.1, 21.4.3.1, 21.6.1, 25.15.3, 25.15.4, 25.18.1, 26.15.4, 26.15.5, 26.18.1.
8. I have visited the Site (PE 58). Site visit dates are as indicated below:
   - Three days from 16 May 2022 to 19 May 2022 to conduct a recognition of the site and facilities proposed for construction to support the Mining Services as well as Site Orientation for various Mining Contractors tendering the works.
9. I have not had prior involvement with the property that is the subject of this Technical Report.
10. I am not independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
12. As of the 'effective date' stated in the Technical Report, to the best of my knowledge, information and belief, the Sections of the Technical Report that I am responsible for, contain all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

CERTIFICATE OF QUALIFIED PERSON

Dated: 30 November 2022

'Signed and Sealed'

Alex Veresezan

M.Sc., P.Eng.

Group Manager - Mining Contracts,

Endeavour Mining plc

CERTIFICATE OF QUALIFIED PERSON

I, David J T Morgan, MSc, MIEAust CPEng, MAusIMM, do hereby certify that:

1. I am the Managing Director at Knight Piesold Pty Limited, Level 1, 184 Adelaide Terrace, East Perth, Western Australia 6055.
2. I am a co-author of the NI 43-101 Technical Report titled 'Lafigué Project, Definitive Feasibility Study' effective date of 1 June 2022.
3. I graduated with a degree from the University of Southampton, UK, with a MSc, Irrigation Engineering (1981).
4. I am a member in good standing of the Australasian Institute of Mining and Metallurgy (Australasia, 202216) and a Chartered Professional Engineer and member of the Institution of Engineers Australia (Australia, 974219).
5. I have worked as a Civil Engineer for a total of 41 years since graduation, 25 of which has been as a Project Director. Relevant experience includes:
   - Akyem Gold project, Project Director.
   - Geita Gold Mine, Project Director.
   - Ahafo Gold Project., Project Director.
6. I have read the definition of 'qualified person' as set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a 'qualified person' for the purposes of NI 43-101.
7. For this Technical Report, I am responsible for the preparation of Sections/Subsections: 1.17.1.2, 1.17.1.3, 1.17.2.4, 1.17.4, 1.17.7.1, 12.9.1, 12.9.2, 12.9.3, 12.9.4, 12.9.8, 12.9.9, 18.2.2, 18.2.3, 18.2.9, 18.3.2.2, 18.3.2.4, 18.3.2.5, 18.5, 18.8.1, 18.10.1, 18.10.2, 18.10.6, 25.15.1, 25.15.2, 25.15.6, 26.15.1, 26.15.3, 26.15.7.
8. I have visited Site (PE 58/PR 329). Site visit dates are as indicated below:
   - two days from 2 July 2021 to 3 July 2021, to review the TSF infrastructure location and site conditions.
9. I have not had prior involvement with the property that is the subject of this Technical Report.
10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
12. As of the 'effective date' stated in the Technical Report, to the best of my knowledge, information and belief, the Sections of the Technical Report that I am responsible for, contain all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

CERTIFICATE OF QUALIFIED PERSON

Dated: 30 November 2022

'Signed and Sealed'

David Morgan

MSc, MIEAust CPEng, MAusIMM,

Managing Director, Knight Piesold Pty Limited

CERTIFICATE OF QUALIFIED PERSON

I Geoff Bailey, FIEAust, CPEng, NPER-3, REPQ do hereby certify that:

1. I am the Principal Engineer/Director at ECG Engineering Pty Ltd, 2-10 Adams Drive, Welshpool, Western Australia 6106.
2. I am a co-author of the NI 43-101 Technical Report titled 'Lafigué Project, Definitive Feasibility Study effective date of 1 June 2022.
3. I graduated with a degree from Curtin University Perth Western Australia, with a Bachelor of Engineering Degree in Electrical (Power) (1978). I am a Fellow Member of Engineers Australia (FIEAust 378695), Chartered Professional Engineer (CPEng), National Professional Engineers Register (NPER-3) and Registered Professional Engineering Queensland (REPQ 06035).
4. I have worked as an Electrical Engineer for over 40 years, the majority of these in the mining industry with considerable experience in Africa.
5. I have read the definition of 'qualified person' as set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a 'qualified person' for the purposes of NI 43-101.
6. For this Technical Report, I am responsible for the preparation of Sections/Subsections: 1.17.3.1, 12.9.7, 18.4.2, 18.4.4.3, 18.4.4.4, 18.10.5, 25.15.5, 26.15.6.
7. I have visited Site (PE 58 and immediate area) on two occasions in 2022, to review the alignment options for the transmission line. Site visit dates are as noted below.
   - Two days from 13 May 2022 to 15 May 2022; and
   - Two days from 8 October 2022 to 10 October 2022.
8. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.
9. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
10. As of the 'effective date' stated in the Technical Report, to the best of my knowledge, information and belief, the Sections of the Technical Report that I am responsible for, contain all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

CERTIFICATE OF QUALIFIED PERSON

Dated: 30 November 2022

'Signed and Sealed'

Mr. Geoff Bailey (FIEAust, CPEng, NPER-3, REPQ)

Principal Engineer/Director

ECG Engineering Pty Ltd

CERTIFICATE OF QUALIFIED PERSON

I Silvia Bottero, M.Sc. (Geological Sciences) do hereby certify that:

1. I am VP Exploration CI with Endeavour Mining plc, with an office at Hotel Palm Club, Abidjan, Côte d'Ivoire, 08 BP 872100.
2. I am a co-author of the NI 43-101 Technical Report titled 'Lafigué Project, Definitive Feasibility Study' effective date of 1 June 2022.
3. I graduated with a degree from the 'Università degli Studi di Genova', in Italy, with a M.Sc. in Geological Sciences (2000).
4. I am a registered Professional Natural Scientist of the SACNASP (South African Council of Natural Scientific Professions, in South Africa). My membership number is #400139/13:
5. I have worked as a Hydrogeologist, Senior Geologist, Exploration Manager or VP Exploration continuously since my graduation from university. I have been employed since my graduation in 2000. I have gained relevant experience on deposits and projects similar to the Fetekro Project. Relevant experience includes:
   - work on greenfield and brownfield exploration programmes, for projects hosted in Birimian greenstone belt formations in west Africa;
   - work in near-mining programmes to extend the mining life of open pit projects in Cote d'Ivoire;
   - participation and supervision of several exploration programmes for different projects, with experience ranging from grassroots regional exploration to project generation and resource infill drilling;
   - design, supervision and implementation of exploration programmes;
   - review, audits and interpretation of geoscientific data; and
   - participation in the preparation of sections of NI 43-101 compliant Technical Reports.
6. I have read the definition of 'qualified person' as set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a 'qualified person' for the purposes of NI 43-101.
7. For this Technical Report, I am responsible for the preparation of Sections/Subsections: [1.5, 1.6, 1.7, 1.8, 1.9, 1.10, 1.11.1, 1.22, 6, 7, 8, 9, 10, 11, 12.2, 12.14, 23, 25.3, 25.4, 25.5, 25.6, 25.7, 25.8, 25.9, 25.20, 26.3, 26.4, 26.5, 26.6, 26.7, 26.8, 26.9, 26.20.]
8. I have visited the Site (PR 329 and PE 58) regularly, since 2014. My most recent visit was for three days from 7 February 2022 to 10 February 2022.
9. I have had prior involvement with the property that is the subject of this Technical Report.
   - From 2014 to 2020, I was the Exploration Manager leading the exploration programme on PR 329, later PE 58.

10. I am not independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
12. As of the 'effective date' stated in the Technical Report, to the best of my knowledge, information and belief, the Sections of the Technical Report that I am responsible for, contain all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated: 30 November 2022

'Signed and Sealed'

Silvia BOTTERO Pr. Sci. Nat.

Endeavour Mining plc, VP Exploration CI

CERTIFICATE OF QUALIFIED PERSON

I Stuart Thomson, MEng (Chemical & Materials Engineering) do hereby certify that:

1. I am the Group Studies Manager at Endeavour Mining plc, 5 Young Street, London, United Kingdom, W8 5EH.
2. I am a co-author of the Technical Report titled ''Lafigué Project, Definitive Feasibility Study, with an 'Effective Date' of 1 June 2022.
3. I graduated from the University of Auckland, New Zealand, with a Masters Degree in Chemical and Materials Engineering (1991).
4. I am a Fellow of the South African Institute of Mining and Metallurgy (FSAIMM). My membership number is: 702632
5. I have worked as a process engineer/consultant, study manager, project director and operations manager for a total of 26 years since graduation, 14 of which has been as a study manager/project director and VP Operations, both for EPCM companies and the owner's team. Experience covers; mineral processing and hydrometallurgical and pyrometallurgical applications and estimating (CAPEX and OPEX), across a broad spectrum of commodities (base metals, precious metals, coal and industrial minerals). Relevant gold experience includes:
   - Anglo Gold, Sadiola PFS/DFS (Mali), refractory gold study - Study Manager.
   - Mwana Africa, Zani-Kodo CS (DRC), refractory gold study - Process Consultant/Study Manager.
   - Golden Star Resources, Bogoso DFS (Ghana), free milling ore plant upgrade - Study Manager.
   - Kilo Gold (DRC), refractory gold deposit - VP Operations.
6. I have read the definition of 'qualified person' as set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a 'qualified person' for the purposes of NI 43-101.
7. For this Technical Report, I am responsible for the preparation of Sections/Subsections: 1.1, 1.2, 1.3, 1.4, 1.11.2, 1.18, 1.21, 1.23.1, 1.24, 1.25, 2, 3, 4, 5, 12.1, 12.3, 12.4, 12.10, 12.13, 12.15.1, 18.3.3.2, 18.4.1, 18.7.7, 18.7.9, 19, 22, 24.1, 25.1, 25.2, 25.16, 25.19, 25.21.1, 26.1, 26.2, 26.16, 26.19, 26.21.1.
8. I have visited the Site (PE 58). Site/Area visit dates are as indicated below:
   - One full day on Site on 25 October 2021, for a general orientation of the Lafigué Site, and a half day on 24 October 2021 visiting CI Energie's Switch Yard in Dabakala.
9. I have not had prior involvement with the property that is the subject of this Technical Report.
10. I am not independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
12. As of the effective date, to the best of my knowledge, information and belief, the Sections of the Technical Report that I am responsible for, contain all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated: 30 November 2022

'Signed and Sealed'

Stuart Thomson

M.Eng, FSAIMM

Group Studies Manager, Endeavour Mining plc

CERTIFICATE OF QUALIFIED PERSON

I, David Taylor, Bachelor of Engineering (Mech) do hereby certify that:

1. I am a Senior Consultant at Lycopodium Minerals Pty Ltd, Level 5, 1 Adelaide Terrace, East Perth, WA 6004, Australia.
2. I am a co-author of the NI 43-101 Technical Report titled ‘Lafigué Project, Definitive Feasibility Study effective date of 1 June 2022.
3. I graduated with a degree from Monash University, Australia with a Degree in Mechanical Engineering (with Honours) (1977).
4. I am a Chartered Professional Engineer with Engineers Australia, and a Fellow of the Institution of Engineers, Australia FIE(Aust). My FIE (Aust) membership number is 118869.
5. I have worked as a Mechanical Design Engineer for a total of 45 years since graduation, 25 of which has been as a Principal Design Engineer or Design Manager. Relevant experience includes:
   - Senior design involvement in 30 gold projects, including; new, expanded, refurbished or relocated plants.
   - Conceptual, PFS, DFS, bankable FS and project detail design.
6. I have read the definition of ‘qualified person’ as set out in National Instrument 43-101 (‘NI 43-101’) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a ‘qualified person’ for the purposes of NI 43-101.
7. For this Technical Report, I am responsible for the preparation of Sections/Subsections: 1.17.1.1, 1.17.2.1, 1.17.2.2, 1.17.2.3, 1.17.2.5, 1.17.3.2, 1.17.3.3, 1.17.5, 1.17.6, 1.20.2, 1.23.2, 12.9.10, 12.12.1, 12.15.2, 18.1, 18.2.1, 18.2.4, 18.2.5, 18.2.6, 18.2.7, 18.2.8, 18.2.10, 18.2.11, 18.3.1, 18.3.2.1, 18.3.2.3, 18.3.3.1, 18.4.3, 18.4.4.1, 18.4.4.2, 18.4.5, 18.6.1, 18.6.2, 18.6.3, 18.6.4, 18.6.5, 18.6.6, 18.6.7, 18.7.1, 18.7.4, 18.7.5, 18.7.6, 18.7.8, 18.10.8, 21.1, 21.2.1, 21.2.2, 21.2.3, 21.2.4, 21.2.5, 21.2.6, 21.2.9, 21.4.1, 21.4.2, 21.4.3.2, 21.4.3.3, 21.4.4, 21.6.2, 24.2, 25.15.7, 25.18.2, 25.21.2, 26.15.2, 26.15.9, 26.18.2, 26.21.2.
8. I have not visited the Site (PE 58).
9. I have had prior involvement with the property that is the subject of this Technical Report.
   - Design Manager for the Lafigue DFS.
10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
12. As of the ‘effective date’ stated in the Technical Report, to the best of my knowledge, information and belief, the Sections of the Technical Report that I am responsible for, contain all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

CERTIFICATE OF QUALIFIED PERSON

Dated: 30 November 2022

'Signed and Sealed'

David Taylor

B.Eng(Mech), CP Eng, FIE(Aust),

Senior Consultant, Lycopodium Minerals Pty Ltd

CERTIFICATE OF QUALIFIED PERSON

I Abraham Buys, NHD. (Extraction Metallurgy) do hereby certify that:

1. I am Group Manager - Process at Lycopodium Minerals Pty Ltd, Level 5/1 Adelaide Terrace, East Perth, 6004 Western Australia, Australia.
2. I am a co-author of the Technical Report titled 'Lafigué Project, Definitive Feasibility Study', effective date of 1 June 2022.
3. I graduated with a diploma from Technikon Witwatersrand, South Africa, with an HND (Higher National Diploma) in Extraction Metallurgy (1992).
4. I am a fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM). My membership number is: 227721.
5. I have worked as a metallurgist, operations manager, and process engineer/manager for a total of thirty years since my graduation, over 16 years of which have been as a Process Engineer/Manager. Relevant experience includes:
   - Managed, interpreted, and reported the results from numerous testwork programs on gold ores.
   - Involved in the process design of treatment plants for over 16 years.
6. I have read the definition of 'qualified person' as set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a 'qualified person' for the purposes of NI 43-101.
7. For this Technical Report, I am responsible for the preparation of Sections/Subsections: 1.12, 1.16, 12.5, 12.8, 12.12.2, 13, 17, 18.3.3.3, 21.2.7, 21.2.8.1, 21.2.8.3, 21.3.2, 21.3.2.2, 21.3.2.3, 21.3.2.4, 21.3.3.2, 21.3.3.3, 21.3.3.4, 21.3.3.5, 21.3.3.6, 21.3.3.7, 21.3.3.8, 21.6.3, 25.10, 25.14, 25.18.3, 26.10, 26.14, 26.18.3.
8. I have not visited the Site (PE 58).
9. I have not had prior involvement with the property that is the subject of this Technical Report.
10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
12. As of the 'effective date' stated in the Technical Report, to the best of my knowledge, information and belief, the Sections of the Technical Report that I am responsible for, contain all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

CERTIFICATE OF QUALIFIED PERSON

Dated: 30 November 2022

'Signed and Sealed'

Abraham Buys

FAusIMM,

Group Manager - Process, Lycopodium Minerals Pty Ltd

CERTIFICATE OF QUALIFIED PERSON

I, Lucy Sarah Roberts BSc MSc PhD, AusIMM(CP), do hereby certify:

1. I am a Principal Consultant (Resource Geology) with SRK Consulting (UK) Limited at 5th Floor, Churchill House, 17 Churchill Way, Cardiff, CF10 2HH, UK
2. I am a co-author of the NI 43-101 Technical Report titled 'Lafigué Project, Definitive Feasibility Study' effective date of 1 June 2022.
3. I graduated with a 2:1 degree (BSc, 2000) from Cardiff University, Wales, UK in Exploration Geology and a Masters degree with Distinction (MSc, 2001) in Mineral Resources. I am also a graduate of James Cook University, with a PhD in Applied Geostatistics (2005).
4. I am a member in good standing of the 'Australasian Institute of Mining and Metallurgy' and a Chartered Professional (Geology). My membership number is 211381.
5. I have practiced my profession continuously since graduation. I have been employed by SRK Consulting (UK) Ltd since April 2006.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
7. For this Technical Report, I am responsible for the preparation of Sections/Subsections: 1.13, 14, 25.11, 26.11.
8. I visited the Site (PE 58/PR 329). Site visit dates are as indicated below:
   - three days from 14 May 2021 to 17 May 2021.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
11. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.
12. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

CERTIFICATE OF QUALIFIED PERSON

Dated: 30 November 2022

'Signed and Sealed'

Dr Lucy Roberts

Principal Consult (Resource Geology)

SRK Consulting (UK) Limited

CERTIFICATE OF QUALIFIED PERSON

I Francois Taljaard, Pr.Eng, BEng (Hons), SAIMM, MIMMM do hereby certify that:

1. I am a Principal Consultant (Mining Engineer) at SRK Consulting (UK) Limited at 5th Floor, Churchill House, 17 Churchill Way, Cardiff, CF10 2HH, UK.
2. I am a co-author of the NI 43-101 Technical Report titled 'Lafigué Project, Definitive Feasibility Study', effective date of 1 June 2022.
3. I graduated with a degree from the University of Pretoria, South Africa, with a Bachelor of Engineering in Mining (BEng, 2010) and a Bachelor of Engineering Honours in Technology Management (BEng (Hons),2017).
4. I am a Professional Engineer (Pr.Eng) and a Member of the Engineering Council of South Africa which is a 'Recognised Professional Organisation' (RPO). My Licence number is: 20150469.
5. I have practised my profession continuously since graduation. I have been employed by SRK Consulting (UK) Ltd since October 2019. I have over ten years' experience that is relevant to the style of mineralisation and type of deposit described in the Report, and to the activity for which I am accepting responsibility.
6. I have read the definition of 'qualified person' as set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a 'qualified person' for the purposes of NI 43-101.
7. For this Technical Report, I am responsible for the preparation of Sections/Subsections: 1.14, 1.15, 1.17.7.2, 12.6, 12.7, 15, 16, 18.8.2, 18.10.7, 25.12, 25.13, 26.12, 26.13, 26.15.8.
8. I have not visited the Site (PE 58).
9. I have not had prior involvement with the property that is the subject of this Technical Report.
10. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
11. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.
12. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

CERTIFICATE OF QUALIFIED PERSON

Dated: 30 November 2022

'Signed and Sealed'

Francois Taljaard

Pr.Eng, Principal Consultant (Mining Engineering).

SRK Consulting (UK) Limited.

CERTIFICATE OF QUALIFIED PERSON

I Graham Errol Trusler, MSc, Pr Eng, MIChE, MSAIChE do hereby certify that:

1. I am CEO of Digby Wells and Associates Pty Ltd (DWA), Turnberry Office Park, Digby Wells House, 48 Grosvenor Road, Bryanston, 2191, South Africa.
2. I am a co-author of the NI 43-101 Technical Report titled 'Lafigué Project, Definitive Feasibility Study', effective date of 1 June 2022.
3. I graduated from the University of KwaZulu-Natal, South Africa, with a Masters Degree in Chemical Engineering (1988). Relevant further qualifications include:
   - B.Comm, Economics and Business Economics
4. I am registered as a Professional Engineer (920088) with the Engineering Council of South Africa. I am also registered as a:
   - Member of the Institution of Chemical Engineers (SAIChE) since1994;
   - Chartered Chemical Engineer with the Institution of Chemical Engineers;
   - Fellow of the Water Institute of South Africa; and a
   - Lifetime member of the American Society of Mining and Reclamation.
5. I have worked as an engineer for a total of 30 years since graduation, 29 of which has been as a consultant and expert within the mining industry in metallurgical production, research and environmental and social management. Relevant experience includes:
   - Overseeing numerous ESIA and EMP developments across Africa and in particular west-Africa.
   - Social and Environmental Management Plans and coordination of projects that includes the oversight of and leading teams involved in aquatic ecology, biodiversity, social, GIS, soils, wetlands, rehabilitation, environmental legal requirements, ESG, closure plans and cost assessments, surface and groundwater and serving on environmental oversight committees.
6. I have read the definition of 'qualified person' as set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a 'qualified person' for the purposes of NI 43-101.
7. For this Technical Report, I am responsible for the preparation of Sections/subsections; 1.19, 12.11, 20, 25.17, 26.17.
8. I have not visited the Site (PE 58).
9. I have not had prior involvement with the property that is the subject of this Technical Report.
10. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
11. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.
12. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated: 30 November 2022

'Signed and Sealed'

Graham Trusler

Msc, Pr Eng, MICHE, MSAICHE

CEO

Digby Wells and Associates Pty Ltd (DWA),

29. APPENDIX A

TECHNICAL REPORT FOR THE LAFIGUE GOLD PROJECT, REPUBLIC OF CÔTE D'IVOIRE

Prepared For

La Mancha Côte d'Ivoire

Date Issued: 30 September 2022

Report Prepared by

srk consulting

SRK Consulting (UK) Limited

UK31113

SRK Legal Entity:	SRK Consulting (UK) Limited
SRK Address:	5th Floor Churchill House 17 Churchill Way Cardiff, CF10 2HH Wales, United Kingdom.
Date:	September, 2022
Project Number:	UK31113
SRK Project Director:	Dr Tim Lucks
SRK Project Manager:	Dr James Davey
Client Legal Entity:	La Mancha Côte d’ivoire
Client Address:	La Mancha Côte d’ivoire Immeuble Palm Club Angle de la rue du Lycée Technique Et du Boulevard Latrille 08 BP 872 Abidjan 08 Abidjan République de Côte d’ivoire

Table of Contents: Executive Summary

SRK Consulting (UK) Limited
5th Floor Churchill House
17 Churchill Way
Cardiff CF10 2HH
Wales, United Kingdom
E-mail: enquiries@srk.co.uk
URL: www.srk.com
Tel: + 44 (0) 2920 348 150

EXECUTIVE SUMMARY

TECHNICAL REPORT FOR THE LAFIGUE GOLD PROJECT, REPUBLIC OF CÔTE D'IVOIRE

1 INTRODUCTION

SRK Consulting (UK) Limited ("SRK") is an associate company of the international group holding company, SRK Consulting (Global) Limited (the "SRK Group"). SRK has been requested by La Mancha Côte d'Ivoire ("LMCI", hereinafter also referred to as the "Company" or the "Client"), a subsidiary of Endeavour Mining Corporation ("Endeavour") to prepare the Mineral Resource and Ore Reserve estimates for the Lafigué Gold Project (hereinafter also referred to as the "Project"), located in Côte d'Ivoire.

In September 2020, SRK received a scope of work for the 2021 Mineral Resource and Ore Reserve Estimates. This scope included an authored Mineral Resource Estimate ("MRE") for the Lafigué Gold Project, Côte d'Ivoire ("Lafigué", the "deposit" or the "Project"). This report details the work undertaken in preparing the Mineral Resource Estimate Update for the Lafigué deposit, effective as at 15 May 2022.

The international reporting code used for the reporting of Mineral Resource and Mineral Reserve statements herein is the "CIM Definition Standards on Mineral Resources and Reserves" prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014 (the "CIM Definition Standards") which are incorporated by reference into National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI-43-101".

This report serves as an independent report prepared by Dr James Davey, a Consultant Resource Geologist, under the management of Dr Lucy Roberts (MAusIMM (CP)), a Principal Resource Geologist and Dr Tim Lucks (MAusIMM (CP)), Managing Director and Principal Consultant at SRK.

SRK has relied upon the information provided by LMCI in its review of the data quality used for the MRE; however, SRK takes full responsibility for the Mineral Resource Statement presented herein.

2 PROPERTY, ACCESS, AND HISTORY

The Lafigué Gold Project, within the wider Fétékro Exploration Licence, is located some 500 km north of the economic capital of Côte d'Ivoire, Abidjan. The site is adjacent to Lafigué village which is approximately 80 km northeast of the town of Bouake. The Project area is located approximately 15 km from the nearest sealed road, which is the B412, running approximately east-west between Katiola in the west (50 km), to Dabakala in the East (30 km).

Exploration across the Project area commenced in the 1930’s when Bureau Minier of the France d’Outre-mer (“BUMIFORM”) conducted geological mapping. The Bureau de Recherches Géologiques et Minières (“BRGM”) and Société pour le Développement Minier de la Côte d’Ivoire (“SODEMI”) continued exploration intermittently throughout the 20th Century, including airborne geophysical surveys, stream sediment and soil geochemistry surveys and some limited exploration drilling. Between 1999 and 2002, Compagnie Minière Or (“COMINOR”) delineated some mineralisation at Lafigué through a series of rotary air blast, reverse circulation and diamond core drilling programmes. Exploration works were then terminated until 2010 due to the civil war affecting Côte d'Ivoire. LMCI conducted exploration drilling at Lafigué in 2014 before acquisition of the Project by Endeavour Mining in 2016.

3 LICENCE STATUS

The Lafigué Gold Project is located within the wider Fétékro Exploration Licence (PR 329), which was allocated to SODEMI by decree N°2013-410 on 6 June 2013. The Permit covered an area of 335.5 km² before being reduced by 25% to 249.5 km² on 11 July 2017. A further renewal on 06 June 2019 was not accompanied by a reduction in area. The permit was recently transferred to LMCI by Ministerial order N° 00174/MMG/DGMG on 18 December 2020. Mining license PE58 was granted to LMCI by Decree N° 2021-538 on 22 September 2021. The transfer of PE58 to the operating company, Société de Mines de Lafigué (SML), was granted by Arrêté n° 018/MMPE/DGMG on 12 January 2022.

In accordance with the Mining Code in Côte d’Ivoire, the Fétékro Exploration Licence (PR 329) was issued by a presidential decree, with a validity of an initial three years, where it may be renewed for two consecutive periods of three years each, followed by a final exceptional renewal for a two-year period, provided the titleholder complies with the rights and obligations set by the Mining Legislation.

The Mining Code gives the exploration permit holder the exclusive right to explore for the minerals requested, on the surface and in the subsurface, within the boundaries of the permit.

4 GEOLOGY AND MINERALISATION

Lafiguè is located towards the northern end of the Birimian-age Oumé-Fetekro greenstone belt, a N-S-trending meta-volcano-sedimentary belt comprised primarily of bimodal metavolcanics and clastic metasedimentary rocks.

Lafiguè has been interpreted to lie within a compressive relay domain (or transpressive restraining bend), bound by two NNE-trending sinistral shear corridors, formed at an angle to regional NW-SE directed shortening during the D2 and D3 regional deformation events (Ciancaleoni, 2018). On the deposit scale, gold mineralisation is controlled by a series of ENE-trending shear zones dipping at 10-40° to the SSE.

Mineralisation is often hosted by quartz-carbonate-tourmaline-pyrite-pyrrhotite-gold veins as well as the associated biotite-tourmaline-sericite-chlorite-carbonate alteration zones, where these veins typically exploit the gently dipping brittle-ductile reverse shear zones. Gold is also hosted within broader zones of altered, stacked shear zones in the hanging wall (and to a lesser degree, the footwall) of the main lithological contacts.

In total, the Lafigué mineralisation span a strike length of approximately 2 km, trending ENE and dipping moderately to the SSE to a maximum depth of approximately 440 m below the surface in Lafigué North (approx. down-dip extension of 700-900 m). Mineralisation continuity reduces towards Lafigué Centre and to the south and west. The deposit remains open at depth along some parts of its strike length.

5 DRILLING AND SAMPLING

Drilling at Lafigué since 2017 has comprised of six separate campaigns aiming to delineate the full down-dip and along-strike extent of mineralisation, as well as increase confidence in the geological and grade continuity through infill drilling. Drilling conducting during Endeavour's ownership of the Project, including reverse circulation and diamond core drilling, has been carried out under the supervision of technically qualified personnel applying standard industry approaches. For all drillholes completed since 2017, collar surveys were conducted using a differential GPS and downhole surveys were completed in each drillhole using a Reflex-EZ track ± EZ-Gyro.

The majority of DD and RC holes at Lafigué have been completed on a 20-40 m by 50 m grid, with some areas of closer drilling towards the up-dip portions of the deposit and wider spaced drillholes in down-dip areas. The majority of drillholes dip at 50° or 60° towards an azimuth of either 000° or 335°. Mineralisation typically dips at approximately 20° towards the S/SSE, resulting in drilling intersection angles of 90-110°.

Field duplicates, blank samples and certified reference materials were inserted into the regular sample stream as part of the QAQC programmes during the 2014-2022 drilling campaigns. Overall, SRK considers the majority of sample preparation, analyses and security protocols to conform to industry best practice. SRK notes the absence of QAQC sample results for the 1997 and 2002 drilling campaigns and as such, assay data from these drilling campaigns present a risk in terms of accuracy and precision of the associated assay grades.

6 DENSITY

The density database used by SRK includes a total of 2,214 measurements (with logged lithology and weathering attributes) taken between 2014 and 2021. Density determinations were carried out using drillcore samples representing the full range of lithologies and weathering intensities present at the Project. The average dry bulk density values applied during the MRE are listed in Table ES 1.

Table ES 1: Average dry bulk density values used for the MRE

Lithology	Number of Measurements	Mean Value (g/cm^3)
Laterite	1	2.00
Saprolite	9	1.66
Saprock	17	2.51
Fresh - Mafic	1,205	2.86
Fresh - Felsic	628	2.72

7 GEOLOGICAL MODELLING

SRK produced a simplified lithology model, based on a refined lithology logging field, as well as a weathering model constructed using surfaces based on weathering/material type logging completed by on site geologists.

SRK selected a nominal modelling cut-off grade of 0.30 g/t Au for the modelling of Au mineralisation, using an indicator interpolant with a probability value of 0.4. The indicator interpolant was guided by a structural trend based on a series of surfaces interpreted to be the primary controls on the geometry and distribution of mineralisation (ie. lithological contacts and associated shear zones). Additionally, a series of vein wireframes were produced based on interval selections in order to accurately model thinner mineralisation domains towards the west of the deposit where mineralisation continuity is reduced.

8 MINERAL RESOURCE

SRK carried out the following steps to produce the MRE:

- database compilation and review;
- construction of wireframe geological models in Leapfrog Geo 2021.1 software;
- statistical analysis and definition of domains;
- geostatistical analysis (variography) within estimation domains;
- block modelling and grade interpolation using Leapfrog Edge software;
- model validation;
- Mineral Resource classification;
- consideration of reasonable prospects for eventual economic extraction (RPEEE); and
- reporting of Mineral Resources.

The 2022 Mineral Resource statement for the Lafigué gold deposit is shown in Table ES 2.

Table ES 2: Mineral Resource statement for the Lafigué Gold Project, effective of 15 May 2022*

Mineral Resource Classification Category	Material Type	Tonnes (Mt)	Grade	Metal Content
			Au (g/t)	Au (koz)
Indicated	Oxide	0.7	1.55	36
	Transition	1.7	1.71	94
	Fresh	43.8	2.06	2,896
	Total	**46.3**	**2.03**	**3,027**
Inferred	Oxide	0.1	1.22	4
	Transition	0.1	2.05	4
	Fresh	1.4	2.11	94
	Total	**1.5**	**2.05**	**102**

**In reporting the Mineral Resource Statement, SRK notes the following:*

- *The reported Mineral Resources are depleted to a drone survey provided to SRK by Endeavour. The survey was conducted on 17 August 2021 and only accounts for artisanal open pit development at surface. SRK understands that there are further artisanal mining workings underground, but these cannot be captured by the drone survey. To account for this, outside of (and below, where necessary) the artisanal open pit workings, to a depth of 5m below the pre-mining topography, the Mineral Resource grades have been reduced to zero. The pre-mining topography was supplied to SRK by LMCI/Endeavour. In the absence of any underground survey, and to reflect the uncertainty for these areas, SRK has not depleted the tonnages.*
- *SRK is aware of ongoing artisanal mining at the Project, and as such, highlights the risk associated with more extensive depletion due to ongoing artisanal mining activity than is accounted for in this Mineral Resource Statement.*
- *The reported Mineral Resources have an effective date of 15 May 2022. The Competent Person for the declaration of Mineral Resources is Dr Lucy Roberts, MAusIMM(CP), of SRK Consulting (UK) Ltd. The Mineral Resource estimate was authored by Dr James Davey, also of SRK;*
- *Technical and economic assumptions were agreed between SRK and LMCI/Endeavour for mining factors (mining and selling costs, mining recovery and dilution, pit slope angles) and processing factors (gold recovery, processing costs), which were used for optimisation. These factors were developed as part of the ongoing Feasibility Study for the Lafigué project, as stated below:*
  - *Ore mining cost: 2.12 (US$/$t_{ore}$)*
  - *Waste mining cost: 2.65 (US$/$t_{rock}$)*
  - *Processing cost: Oxide/Transition: 7.47 (US$/$t_{ore}$); Fresh: 9.13 (US$/t_{ore})*
  - *Selling cost: 71.8 (US$/oz Au)*
  - *Mining recovery: 98%*
  - *Mining dilution: 9%*
  - *Processing recovery: Oxide = 94.87%; Transition = 94.92%; Fresh = 95.08%*
  - *Average slope angles: 33-51°, dependent on geotechnical domain*
  - *G&A cost: 5.60 (US$/$t_{ore}$)*
  - *Discount rate: 5%*
- *SRK considers there to be reasonable prospects for eventual economic extraction by constraining the Mineral Resources within an optimised open pit shell using a gold price of USD 1,500/oz.*
- *Mineral Resources are reported within the optimised pit shell using cut-off of grades of 0.4 g/t Au (oxide); 0.5 g/t Au (transition) and 0.5 g/t Au (fresh), which are the marginal cut-off grades for CIL processing determined during the pit optimisation.*
- *Mineral Resources are reported as in-situ and undiluted, with no mining recovery applied in the Statement. All tonnages are reported on a dry basis.*
- *Mineral Resources are not Ore Reserves and do not have demonstrated economic viability, nor have any mining modifying factors been applied;*
- *Tonnages are reported in metric units, grades in grams per tonne (g/t), and the contained metal in kilo troy ounces. Tonnages, grades, and contained metal totals are rounded appropriately. 1 troy ounce is assumed to be the equivalent of 31.1034 g*
- *Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content. Where these occur, SRK does not consider these to be material.*

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SRK Consulting (UK) Limited
5th Floor Churchill House
17 Churchill Way
Cardiff CF10 2HH
Wales, United Kingdom
E-mail: enquiries@srk.co.uk
URL: www.srk.com
Tel: + 44 (0) 2920 348 150

TECHNICAL REPORT FOR THE LAFIGUE GOLD PROJECT, REPUBLIC OF CÔTE D'IVOIRE

1 INTRODUCTION

1.1 Issuer and Terms of Reference

SRK Consulting (UK) Limited ("SRK") is an associate company of the international group holding company, SRK Consulting (Global) Limited (the "SRK Group"). SRK has been requested by La Mancha Côte d'Ivoire ("LMCI", hereinafter also referred to as the "Company" or the "Client"), a subsidiary of Endeavour Mining Corporation ("Endeavour") to prepare the Mineral Resource and Ore Reserve estimates for the Lafigué Gold Project (hereinafter also referred to as the "Project"), located in Côte d'Ivoire.

In September 2020, SRK received a scope of work for the 2021 Mineral Resource and Ore Reserve Estimates. This scope included an authored Mineral Resource Estimate ("MRE") for the Lafigué Gold Project, Côte d'Ivoire ("Lafigué", the "deposit" or the "Project"). This report details the work undertaken in preparing the Mineral Resource Estimate for the Lafigué deposit, effective as of 15 May 2022.

This report serves as an independent report prepared by Dr James Davey, a Consultant Resource Geologist, under the management of Dr Lucy Roberts, a Principal Resource Geologist and Dr Tim Lucks, Managing Director and Principal Consultant at SRK.

The Mineral Resource Statement presented herein has an effective date of 15 May 2022 and is signed off by Dr Lucy Roberts (MAusIMM (CP)), who acts as the Qualified Person.

SRK has relied upon the information provided by LMCI in its review of the data quality used for the MRE. SRK takes full responsibility for the Mineral Resource Statement presented herein.

1.2 Sources of Information

SRK's report and study is based upon information provided by the Company, along with access to key personnel from the Project technical team on-site. The key sources of information for this report, including information relating to the data quality, data collection procedures and protocols, are as follows:

- Database files:
  - Exploration drilling and sampling database (collar, survey and assay);
  - drillhole logging database (lithology, weathering, oxidation, structure, mineralisation);
  - density database;
  - quality control sample database; and
  - topographic survey mesh (Lidar).

- Maps, plans and sections:
  - Local interpreted geological map covering the deposit area; and
  - schematic interpretive cross-sections denoting the geology and mineralisation of the Deposit, produced by Endeavour geologists.
- Reports:
- Structural geology report prepared by Dr Laurent Ciancaleoni of Arethuse Geology.
- Site Visit

1.3 Capability and Independence of Consultant

This report was prepared on behalf of SRK by the persons whose qualifications and experience are set out in Table 1-1 below.

SRK is an independent consulting engineering organisation, wholly owned by its employees, that has been active in the mining and natural resources industries for nearly 40 years. The group operates globally and currently employs approximately 1,500 professionals in 48 offices worldwide. SRK has a demonstrated track record in undertaking independent assessments of resources and reserves, project evaluations and audits and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies and financial institutions worldwide.

This technical report has been prepared based on a technical and economic review by a team of consultants sourced from SRK's Group office in the United Kingdom.

Neither SRK nor any of its employees and associates employed in the preparation of this report has any material present or contingent interest in the outcome of this report or in any of the Assets being assessed. Nor do they have any pecuniary or other interest that could be reasonably regarded as being capable of affecting their independence or that of SRK. SRK will be paid a fee for the preparation of this report in accordance with normal consulting practice.

The individuals who have provided input to this report, and who are listed below, have extensive experience in the mining industry and are members in good standing of appropriate professional institutions.

Table 1-1: Professional Qualifications of SRK Consulting (UK) Staff

Name	Professional Qualifications and Affiliations	Discipline and Role
Tim Lucks	Member and Chartered Professional with the Australasian Institute of Mining and Metallurgy (MAusIMM(CP)); BSc (Hons) Geology; PhD Mineral Deposit Geology	Project director and reviewer
Lucy Roberts	Member and Chartered Professional with the Australasian Institute of Mining and Metallurgy (MAusIMM(CP)); BSc (Hons) Geology; MSc (Distinction) Mineral Resources; PhD Applied Geostatistics	Geology and Mineral Resources (Qualified Person for Mineral Resource statement)
James Davey	Fellow of the Geological Society of London (FGS); MESci (Hons) Exploration and Resource Geology; PhD Economic Geology	Project manager and geology / Mineral Resources
Francois Taljaard	Member of Southern African Institute of Mining and Metallurgy (MSAIMM); Professional Engineer with Engineering Council of South Africa (Pr.Eng); BEng Industrial Engineering; BEng Mining Engineering	Mining engineering

1.4 Scope of Work, Materiality, Limitations and Exclusions

1.4.1 General

SRK has independently assessed the Lafigué Gold Project by reviewing pertinent data, including that relating to resources, reserves, equipment and manpower requirements, environmental, rehabilitation and abandonment issues and the future plans relating to productivity and production including projected costs and revenues. All opinions, findings and conclusions expressed in this report are those of SRK.

SRK's opinion contained herein is effective as of 15 May 2022 with regards to the Mineral Resource and Mineral Reserve Statements and the review of the mine plan. SRK's opinion is based on information provided by the Company throughout the course of SRK's investigations, which, in turn, reflects various technical conditions at the time of writing. These conditions can change significantly over relatively short periods of time. The achievability of the technical-economic plans is neither warranted nor guaranteed by SRK.

This report contains technical information which may have been used in subsequent calculations to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding which consequently introduces margins of error. Where these occur, SRK does not consider them to be material to the purpose or use of this report.

1.4.2 Compliance and reporting standard

The international reporting code used for the reporting of Mineral Resource and Mineral Reserve statements herein is the "CIM Definition Standards on Mineral Resources and Reserves" prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014 (the "CIM Definition Standards") which are incorporated by reference into National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI-43-101". Furthermore, the Mineral Resource and Mineral Reserves as reported herein have also been prepared in accordance with the "CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" prepared by the CIM Mineral Resource and Mineral Reserve Committee and adopted by the CIM Council on November 29, 2019 (the "CIM Best Practice Guidelines").

The CIM Definition Standards are aligned with the Committee for Mineral Reserves International Reporting Standards (“CRIRSCO”) reporting template. Accordingly, SRK considers the CIM Definition Standards to be an internationally recognised reporting standard which is adopted world-wide for market-related reporting and financial investment.

1.4.3 Limitations

SRK has no reason to believe that any material facts have been withheld and the Company believes it has provided all material information.

The achievability of the projections of technical-economic parameters as included in this Report are neither warranted nor guaranteed by SRK. The projections as presented and discussed herein have been proposed by the Company’s management and adjusted where appropriate by SRK and cannot be assured; they are necessarily based on economic assumptions, many of which are beyond the control of the Company. Future cashflows and profits derived from such forecasts are inherently uncertain and actual results may be significantly more or less favourable.

Unless otherwise expressly stated all the opinions and conclusions expressed in this Report are those of SRK.

1.4.4 Reliance on information

SRK believes that its opinion must be considered as a whole and that selecting portions of the analysis or factors considered by it, without considering all factors and analyses together, could create a misleading view of the process underlying the opinions presented in the Report.

SRK’s assessment of Mineral Resources and Mineral Reserves, and technical-economic forecasts are based on information provided by the Company throughout the course of SRK’s investigations, which in turn reflect various technical-economic conditions prevailing at the date of this Report. In particular, the Mineral Resources and Mineral Reserves, and the technical-economic models are based on expectations regarding the commodity prices and exchange rates prevailing at the date of this report. These projections can change significantly over relatively short periods. Should these change materially the projections could be materially different. Furthermore, SRK has no obligation or undertaking to advise any person of any change in circumstances which comes to its attention after the date of this Report or to review, revise or update the Report or opinion.

1.4.5 Declaration

SRK will receive a fee for the preparation of this report in accordance with normal professional consulting practice. This fee is not contingent on the outcome of any applications made by the Company and SRK will receive no other benefit for the preparation of this report. SRK does not have any pecuniary or other interests that could reasonably be regarded as capable of affecting its ability to provide an unbiased opinion in relation to the Mineral Resources and Mineral Reserves, and the projections and assumptions included in the various technical studies completed by the Company, opined upon by SRK and reported herein.

Neither SRK, the SRK professional staff responsible for authoring this Report, nor any Directors of SRK, have at the date of this report, nor have had within the previous two years, any shareholding in the Company, the Assets, or advisors of the Company. Consequently SRK, the SRK Qualified Persons and the Directors of SRK considers themselves to be independent of the Company.

In this Report, SRK provides assurances to the Board of Directors of the Company that the technical-economic models, including production profiles, operating expenditures, and capital expenditures, of the Assets as provided to SRK by the Company and reviewed and where appropriate modified by SRK is reasonable, given the information currently available.

1.4.6 Copyright

Copyright of all text and other matter in this document, including the manner of presentation, is the exclusive property of SRK. It is an offence to publish this document or any part of the document under a different cover, or to reproduce and/or use, without written consent, any technical procedure and/or technique contained in this document. The intellectual property reflected in the contents resides with SRK and shall not be used for any activity that does not involve SRK, without the written consent of SRK.

1.5 Inherent Risks

Mining and processing are carried out in an environment where not all events are predictable. Whilst an effective management team can identify the known risks and take measures to manage and mitigate these risks, there is still the possibility for unexpected and unpredictable events to occur. It is not possible therefore to totally remove all risks or state with certainty that an event that may have a material impact on the operation of a mine will not occur. Similar considerations apply to the marketing of the minerals.

1.6 Site Visits and Inspections

SRK Qualified Person, Dr Lucy Roberts, Principal Consultant (Resource Geology), and Dr James Davey, Consultant (Resource Geology), visited the site between 14 May and 16 May 2021. The visit involved a tour of the Project area; verification of a selection of drillhole collar positions; a review of selected drillcore and RC chip samples; discussion on the geological and mineralisation interpretation; and reviewing some quality assurance/quality control (“QA/QC”) procedures employed by the Company.

2 RELIANCE ON OTHER EXPERTS

SRK’s opinion, effective as of 15 May 2022, is based on information provided to SRK by the Company throughout the course of SRK’s investigations as described. These in turn reflect various technical and economic conditions at the time of writing.

This report includes technical information, which requires subsequent calculations to derive sub-totals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.

SRK was reliant upon information and data provided by the Company; however, SRK has, where possible, verified data provided independently and has undertaken a site visit to review physical evidence for the Project.

The main technical reports utilised for reference in producing this Technical Report are as follows:

- Lafigué Gold Project Pre-Feasibility Study National Instrument 43-101 Technical Report (2167-GREP-002), 2021, Lycopodium; and
- Ciancaleoni, L. 2018. Structural geology of oriented cores from selected targets of the Fétékro project (LMCI, Côte d'Ivoire), Arethuse Geology Sarl.

3 PROJECT DESCRIPTION AND LOCATION

3.1 Location

The location of the Lafigué Project, within the wider Fétékro Exploration Licence, is shown in Figure 3-1. It is located some 500 km north of the economic capital of Côte d'Ivoire, Abidjan. The site is adjacent to Lafigué village which is approximately 80 km northeast of the town of Bouake.

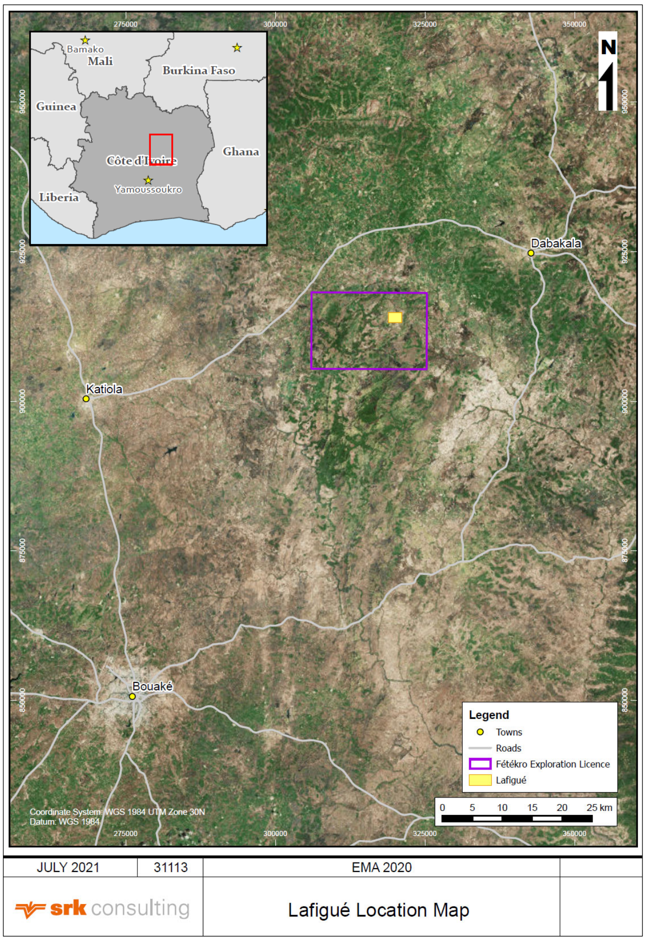


Figure 3-1: Map showing the location of the Lafiguè Project within the wider Fètèkro Exploration Licence area located between the towns of Katiola and Dabakala

3.2 Licences, Permits and Ownership

The Lafigué Gold Project is located within the wider Fétékro Exploration Licence (PR 329), which was allocated to Société pour le Développement Minier de la Côte d'Ivoire ("SODEMI") by decree N°2013-410 on 6 June 2013. The Permit covered an area of 335.5 km² before being reduced by 25% to 249.5 km² on 11 July 2017 as shown in Figure 3-1, with corner points at the locations specified in Table 3-1. A further renewal on 06 June 2019 was not accompanied by a reduction in area. The permit was recently transferred to LMCI by Ministerial order N° 00174/MMG/DGMG on 18 December 2020. Mining license PE58 was granted to LMCI by Decree N° 2021-538 on 22 September 2021. The transfer of PE58 to the operating company, Société de Mines de Lafigué (SML), was granted by Arrêté n° 018/MMPE/DGMG on 12 January 2022.

Table 3-1: Boundary Coordinates for Exploration Permit PR 329

Corner Point	Latitude	Longitude
A	8º 18' 12.00"	4º 45' 40.00"
B	8º 18' 12.00"	4º 35' 10.00"
C	8º 11' 12.00"	4º 35' 10.00"
D	8º 11' 12.00"	4º 45' 40.00"

The initial Fétékro Exploration permit (PR 329) was issued by a presidential decree, with a validity of an initial three years. Then, in accordance with the 2014 Mining Code in Côte d'Ivoire, PR 329 was renewed for two consecutive periods of three years each. Typically, at each renewal stage, at least 25 per cent of the original area must be relinquished, however the titleholder may retain the right over the full license area through the payment of an 'option fee'.

The Mining Code gives the exploration permit holder the exclusive right to explore for the minerals requested, on the surface and in the subsurface, within the boundaries of the permit.

3.3 Environmental Studies and Permits

Environmental baseline studies began at Lafigué in November 2019, with the publication of an Environmental and Social Impact Assessment ("ESIA") in September 2020. Arrêté Environmental approved the ESIA for commercial gold mining operations in February 2021.

3.4 Royalties

In Côte d'Ivoire, mining permits are subject to a 10% carried ownership interest to the benefit of the state, and the permit holder pays production royalties as summarised in Table 3-2.

Table 3-2: Côte d'Ivoire Government Royalty Rates

Royalty	Gold Price
3%	Up to US$1,000
3.5%	US$1,000 – US$1,300
4%	US$1,300 – US$1,600
5%	US$1,600 – US$2,000
6%	Greater than US$2,000

Additionally, a 0.5% social contribution is applied to total revenue.

4 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

Much of the content in the following Sections is based on or reproduced from previous technical and MRE reports including:

- Lafigué Gold Project Pre-Feasibility Study National Instrument 43-101 Technical Report (2167-GREP-002), 2021, Lycopodium.

4.1 Physiography

Lafiguè lies within the Bandama Valley, defined by a series of small northeast-trending hills to the northwest and southeast of the Project area. The immediate prospect area is defined by two hills, separated by a central, north-south-trending valley, with topography ranging in elevation from approximately 290-430 masl. The Project area is characterised by savannah forest with grass, with nearby cattle farming, as well as cotton, cashew, peanut, rice and corn plantations.

4.2 Climate

Lafiguè is located in central Côte d'Ivoire and as such, experiences a transitional tropical climatic regime, between arid and semi-arid sub-Saharan areas to the north and humid tropical areas to the south (Gulf of Guinea). Rainfall averages 800 mm annually, including average monthly rainfall of approximately 120 mm in the wet season (June to September), reducing to <40 mm in the dry season (November to February) (Figure 4-1). The dry season is characterised by the Harmattan winds, blowing from north to south, originating in the Sahara Desert. Temperatures range from a low of 19°C to a high of 29°C during the wet season, increasing to lows of 24°C, highs of 34°C during the dry season, with an average annual temperature of approximately 28°C.

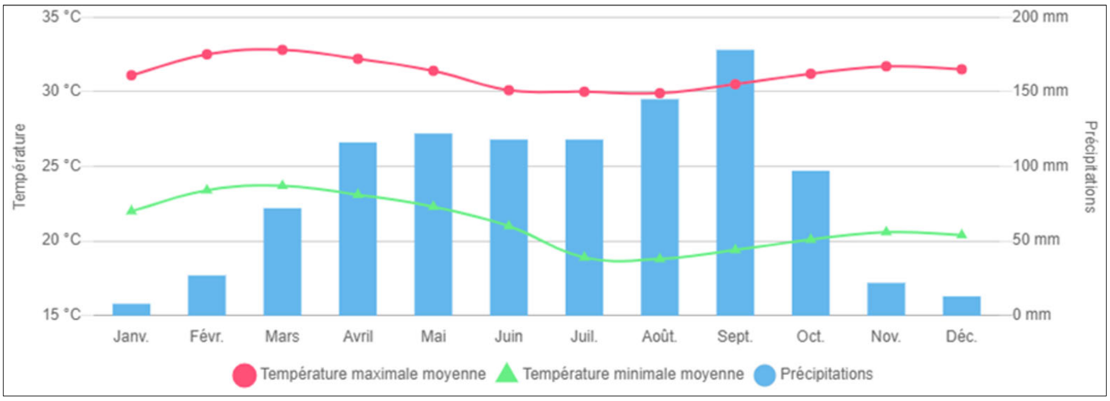


Figure 4-1: Average temperatures and precipitation for the nearby city of Katiola

Prevailing winds in the Project area are typically southwesterly, with speeds of 1-4 ms^{-2}. The wider Dabakala region experiences significantly contrasting humidity through the year, ranging from >80% for the majority of April to November, to significantly reduced humidity during the dry season.

4.3 Access

The Project area is located approximately 15 km from the nearest sealed road, which is the B412, running approximately east-west between Katiola in the west (50 km), to Dabakala in the East (30 km) (Figure 3-1). The site is accessed via a 15 km long dirt track from Boniérédougo to the village of Lafiguè, which is adjacent to the deposit.

4.4 Infrastructure and Local Resources

Dabakala is the closest mid-sized town to Lafiguè (population: 56,000, as of 2014 census), with basic services including a hospital, shops and hotels, as well as providing a source of local manpower. Specialised services and equipment are sourced from Yamoussoukro or Abidjan.

Power is envisaged to be supplied via connection with the national grid, where the connecting infrastructure will need to be built during mine construction. Additionally, a 15 km sealed road will be required to link the Project site with the B412 national highway, along with further infrastructure for water supply, air access and mine site processing and accommodation facilities.

5 HISTORY

This section describes the discovery, ownership and early exploration and resource definition history of the Project prior to Endeavour taking an active role in ownership and exploration from 2017. Much of the content in the following Sections is based on or reproduced from previous technical and MRE reports including:

- Lafigué Gold Project Pre-Feasibility Study National Instrument 43-101 Technical Report (2167-GREP-002), 2021, Lycopodium.

5.1 Early Exploration History

The earliest exploration work across the Project area commenced in 1935 when BUMIFORM conducted geological mapping. The BRGM and SODEMI conducted airborne geophysical surveys during the late 1960's and early 1970's, before an exploration, development and operating agreement was set up between SODEMI and GENCOR (through its Ivorian subsidiary, GATRO-CI) in 1996. Through the agreement GATRO-CI completed a series of stream sediment and soil geochemistry surveys, exploration pits and trenches, and a small amount of drilling (14 diamond core drillholes and 37 reverse circulation holes), and defined four main targets, including Lafiguè.

Between 1999 and 2002, COMINOR conducted exploration works including exploration drilling in 2002 comprising of 1,803 m of rotary airblast ("RAB") drilling, 1,281 m of reverse circulation ("RC") drilling and 461 m of diamond core ("DD") drilling, which demonstrated mineralisation was not continuous between Lafigué Center and Lafigué North and that locally, felsic dykes play a role in controlling some mineralisation. Exploration works were then terminated until 2010 due to the civil war affecting Côte d'Ivoire. When COMINOR recommenced exploration works in 2010, a further 11 RC holes (1,109 m) and 4 DD holes (396 m) were drilled to assess the down-dip extents of mineralisation.

LMCI conducted exploration at the Project in 2014, comprising of 23 DD holes (1,864 m) and 54 RC holes (4,634 m), focusing on Lafigué North as well as obtaining structural data to better understand mineralisation controls. The majority of historical boreholes were resurveyed by differential GPS in 2014 by Environnement Technologie Côte d'Ivoire, with the exception of the RAB holes and three RC drillholes completed in 1997, which could not be located (R2087, R2997, R30B97).

A summary of exploration drilling completed at Lafiguè prior to Endeavour Mining Corporation ownership is provided in Table 5-1. Other exploration activities are summarised in Table 5-2.

Table 5-1: Summary of exploration drilling prior to Endeavour ownership

Year	Drilling Type	Number of Drillholes	Metres
1997	DD	14	1,447
	RC	37	1,549
2002	DD	11	461
	RAB	94	1,803
	RC	32	1,281
2010	DD	4	396
	RC	11	1,109
2014	DD	23	1,864
	RC	54	4,638
Total		**280**	**14,548**

Table 5-2: Summary of historical exploration activity at Lafiguè and surrounding areas

Company	Year	Activity	Primary Results
BRGM. Canadian Aero Mineral Surveys Ltd	**1965-1968**	**Regional magnetic airborne survey**	
		Kenting Pty Ltd mission 1973-76	
GATRO CI	**1994**	**Stream sediment geochemistry**	4 gold anomalies:
		2,143 samples taken (1 per 1.2 km²)	Sandérékro, Tibéguélé, Lafigué, Sarakakro
		1,970 samples analysed for gold	
		1,006 ICP analyses	
GATRO CI	**1995**	**Soil geochemistry (Lafiguè anomaly)**	4 gold anomalies:
		200 x 100 m spacing (locally 100 x 50)	Lafigué A: Main anomaly, 1,700 m long and 250 m wide
		1,862 samples Au analyses (detection limit : 5 ppb)	Lafigué B: 2 values >2 g/t Au but not validated. Work stopped in this area
			Lafigué C: Close to a granodiorite, 300 m long and 100 m wide
			Lafigué D: Flat area. Work stopped in this area
		Rock samples	
		35 samples taken and Au Analysed	Lafigué A: 3,4 and 6 g/t Au in rock samples
GATRO CI	**1996**	**LAFIGUE A anomaly**	Mineralisation typically low grade and disseminated (0.4 to 0.8 g.t Au). Highest grade (up to 50 g/t Au) associated with quartz tourmalines veins. Channel samples in the overburden returned an average grade of 3.8 g/t and 1.3 m average length. Some mineralisation was intersected in vertical groove samples (1.3 m @ 3.8 g/t).
		Soil geophysics	
		(magnetics, VLF-EM, PP)	
		Mag. and VLF: 130 km lines with a 100 x 30 m spacing	
		• Induced polarisation: 4 profiles (7.4 km)	
		• Trenches - 26 trenches (3,800 m)	
		2,154 samples taken (horizontal grooving)	
		264 samples (vertical grooving)	
GATRO CI	**1997**	**Diamond core drilling campaign**	Gold associated with weathered schists and mylonites in metavolcanic rocks. The rocks are locally sheared.
		14 drillholes (1,447 m)	Coarse gold associated with quartz veins with sericite and tourmaline.
GATRO CI	**1997**	**LAFIGUE "A" Anomaly**	First calculation of the mineralized quartz potential close to the surface and in the saprolite:
		Exploration pits:	Estimate (using a density of 1.57 g/ cm^3):
		100 x 100 m spacing. Infilled to 50 x 50 m	Tonnage (t): 978 797
		313 pits completed (depths from 0.45 to 8.7 m)	Grade (g/t Au): 2.54

Company	Year	Activity	Primary Results
		1,018 grooved samples	Au metal (t): 2.5
		RC drilling campaign	
		37 RC drillholes (1,549m)	Tested mineralisation continuity in the Centre and South areas
		Mineral processing test	**Results:**
		3 oxidised samples were tested:	1) Gravity recovery:
		• One from the overburden quartz	84-88% recovery
		• One from an in situ quartz vein	86-89% recovery
		• Disseminated (rock-hosted) gold	17-35% recovery
		Initial gravity separation (Knelson concentration), with the rejects cyanide leached	2) Cyanide leaching and gravity recovery: quartz vein: 98-99% disseminated (rock-hosted) gold: 64-83%
COMINOR	**2001**	**Metallurgical Testwork** Inventory of RC 1997 duplicate samples. 40 samples were sent to SGS for analyses. "A" and "Z" samples manufacturing for insertion during the RAB and RC drilling campaign	Mineral processing test (bottle test) of 36 samples from overburden mineralisation (12 holes, 21.35 m)
COMINOR	**2002**	**Lafiguè "A" anomaly drilling:** RAB: 94 holes, 1,803 m, 12 profiles RC: 32 drillholes, 1,281 m, 17 profiles DD: 11 drillholes, 461 m, 8 profiles	RAB Targets: • Continuity between the Centre and the North area, search for extension, "C" anomaly checking. Results: No continuity, 200 m Eastern extension, no positive results on C. • Center area extension, increase the resources. Results: discontinuity on the centre zone linked with a felsic dyke N125°E, North: discontinuity also explained by a felsique dyke N160°E DD and RC Targets: Main mineralisation confirmation, density measurements. Mean density: saprolite: 2.0, Oxidised zone: 2.1, transition zone: 2.5, sulphide zone: 2.8.
COMINOR	**2010**	Drilling campaign:	Targets: - Check the mineralisation extension downdip on the centre area, check the extensions Results: LFDD10 and LFRC10 cf. Table 6: Lafigue best mineralised intercepts (cut off: 0.5g /t, intercepts with an average grade superior at 1 g/t, trenches excluded)
		RC: 11 drillholes, 1,109 m DD: 4 drillholes, 396.30 m	
LMCI	**2014**	**Drilling campaign:** RC: 54 drillholes, 4,634 m DD: 23 drillholes, 1,864 m	Testing of extensions
LMCI	**2015**	DGPS survey of collars	
		LIDAR survey done by AOC	

5.2 Significant Previous Mineral Resource Estimates

The last Mineral Resource estimate conducted prior to Endeavour's ownership of Lafiguè was completed in 2003 by COGEMA based on an updated geological model and density measurements since the previous estimate (2002). The 2003 Mineral Resource Statement was not classified, but was split into North, Centre and South zones, as detailed in Table 5-3. This estimate was not reported publicly or in accordance with any internationally recognised codes or regulations. The 2003 estimate has not been reviewed by SRK and should not be considered a current Mineral Resource estimate.

Table 5-3: COGEMA 2003 preliminary Mineral Resource estimate for Lafiguè

	Oxide Zone			Sulphide Zone			Total		
	Tonnes (kt)	Grade (g/t Au)	Metal (t)	Tonnes (kt)	Grade (g/t Au)	Metal (t)	Tonnes (kt)	Grade (g/t Au)	Metal (t)
North	655	1.81	1.22	299	1.87	0.56	914	1.94	1.78
Centre	550	2.49	1.37	606	3.89	2.36	1,157	3.23	3.73
South	315	1.50	0.47	-	-	-	315	1.50	0.47
OVB*	1,288	2.30	2.96	-	-	-	1,288	2.30	2.96
Total	**2,769**	**2.17**	**6.02**	**905**	**3.22**	**2.91**	**3,674**	**2.43**	**8.94**

*OVB = Overburden
Reported above a 1 g/t Au cut-off grade
Rounding may result in apparent summation differences between tonnes, grade and contained metal content

Subsequent Mineral Resource estimates have been completed by Endeavour on an annual basis between 2017 and 2020, as detailed in Table 5-4 to Table 5-7.

Table 5-4: Endeavour Mineral Resource estimate for the Lafigué deposit – October 2017

Fetekro Lafigué October 2017 Mineral Resource Estimate constrained by $1500 Pit Shell; cut-off 0.5 g/t Au					
Indicated			**Inferred**		
Tonnes (kT)	**Grade (g/t)**	**Au (koz)**	**Tonnes (kT)**	**Grade (g/t)**	**Au (koz)**
4,981	2.34	375	898	2.19	63

Table 5-5: Endeavour Mineral Resource estimate for the Lafigué deposit – October 2018

Fetekro Lafigué October 2018 Mineral Resource Estimate constrained by $1500 Pit Shell; cut-off 0.5 g/t Au					
Indicated			**Inferred**		
Tonnes (kT)	**Grade (g/t)**	**Au (koz)**	**Tonnes (kT)**	**Grade (g/t)**	**Au (koz)**
6,833	2.25	494	3,039	2.25	225

Table 5-6: Endeavour Mineral Resource estimate for the Lafigué deposit – October 2019

Fetekro Lafigué October 2019 Mineral Resource Estimate constrained by $1500 Pit Shell; cut-off 0.5 g/t Au					
Indicated			**Inferred**		
Tonnes (kT)	**Grade (g/t)**	**Au (koz)**	**Tonnes (kT)**	**Grade (g/t)**	**Au (koz)**
14,577	2.54	1,190	867	2.17	60

Table 5-7: Endeavour Mineral Resource estimate for the Lafigué deposit – October 2020

Fetekro Lafigué October 2020 Mineral Resource Estimate constrained by $1500 Pit Shell; cut-off 0.5 g/t Au					
Indicated			**Inferred**		
Tonnes (kT)	**Grade (g/t)**	**Au (koz)**	**Tonnes (kT)**	**Grade (g/t)**	**Au (koz)**
32,030	2.40	2,471	820	2.52	66

In 2021 SRK produced an updated Mineral Resource estimate for the Lafiguè Project, based on additional drilling and a revised modelling approach. The Mineral Resource Statement associated with the study is presented in Table 5-8.

Table 5-8: SRK Mineral Resource estimate for the Lafigué deposit – September 2021

Fetekro Lafigué September 2021 Mineral Resource Estimate constrained by $1500 Pit Shell; cut-off 0.4 g/t Au (oxide); 0.5 g/t Au (transition and fresh)					
Indicated			**Inferred**		
Tonnes (kT)	**Grade (g/t)**	**Au (koz)**	**Tonnes (kT)**	**Grade (g/t)**	**Au (koz)**
44,805	2.02	2,917	3,559	2.36	269

5.3 Significant Previous Mineral Reserve Estimates

A historical Mineral Reserve Estimate was completed for the Lafiguè deposit by GATRO-CI, however SRK has not been supplied with details of this estimate. SRK understands the estimate was for internal use only and was not reported publicly or within any regulatory environment.

5.4 Mine Production History

Lafiguè has not been mined on a commercial scale however there has been significant artisanal mining works, primarily targeting the quartz-tourmaline vein-hosted mineralisation. Since September 2021, Endeavour, alongside the Dabakala Gendarmes, have been undertaking an eviction exercise whereby the majority of artisanal miners have been removed from site.

6 GEOLOGY

6.1 Regional Geology

The majority of known gold resources within the West Africa craton are hosted by the Paleoproterozoic lithologies of the Man-Leo shield (also referred to as the Baoulé-Mossi domain). The gold deposits are typically constrained to NNE-SSW-trending Birimian greenstone belts, formed from calc-alakline or tholeiitic volcanic rocks, with metasedimentary rocks filling adjacent sub-basins (Goldfarb et al., 2017). The greenstone belts themselves most likely represent juvenile oceanic arcs accreted onto continental margin, with adjacent sediments often derived from erosion of arc rocks into back-arc basins which developed into foreland basins during basin closure (Baratoux et al., 2011). Following the emplacement of intrusive and volcanic rocks of the Birimian Supergroup, and the deposition of the clastic sediments of the Tarkwa Supergroup, the region underwent regional greenschist facies metamorphism, with some localised higher-grade metamorphism. This is particularly associated with the largest intrusive centres (John et al., 1999; White et al., 2013).

Lafiguè is located towards the northern end of the Birimian-age Oumé-Fetekro greenstone belt, a N-S-trending meta-volcano-sedimentary belt comprised primarily of bimodal metavolcanics and clastic metasedimentary rocks. The belt is developed along a northeast-trending shear zone and is intruded and surrounded by a series of granite and granodiorite complexes. Other notable gold deposits developed along the Oumé-Fetekro greenstone belt include Agbaou and Bonikro, both located to the south of Lafiguè.

Past field studies (*cf*. Mortimer, 1990; Leake, 1992; Houssou, 2013; Ouattara, 2015) indicate multiple phases of deformation for the Oumé-Fetekro greenstone belt, as summarised:

- **D1** WNW-ESE compression resulting in the formation of NNE-trending upright to isoclinal folds (F1) with a penetrative axial-planar cleavage (S1).
- **D2** WNW-ESE to NW-SE compression produced isoclinal to upright NNE- to NE-trending folds (F2), a penetrative axial-planar cleavage (S2), and moderate- to high-angle reverse shear zones.
- **D3** NW-SE transpression marking a switch from a coaxial deformation regime to a non-coaxial regime and an evolution from ductile to brittle-ductile behaviour. This deformation phase is associated with the formation of a NE-trending spaced crenulation cleavage (S3) and the dissection of the Oumé-Fetekro greenstone belt by N- to NNE-trending sinistral shear zones.
- **D4** E-W shortening occurring at high crustal levels, responsible for the development of ENE-trending (dextral) and WNW-trending (sinistral) brittle strike-slip conjugate faults. This deformation episode is also associated with the formation of localised N-trending upright folds (F4) and associated axial-planar cleavage (S4).

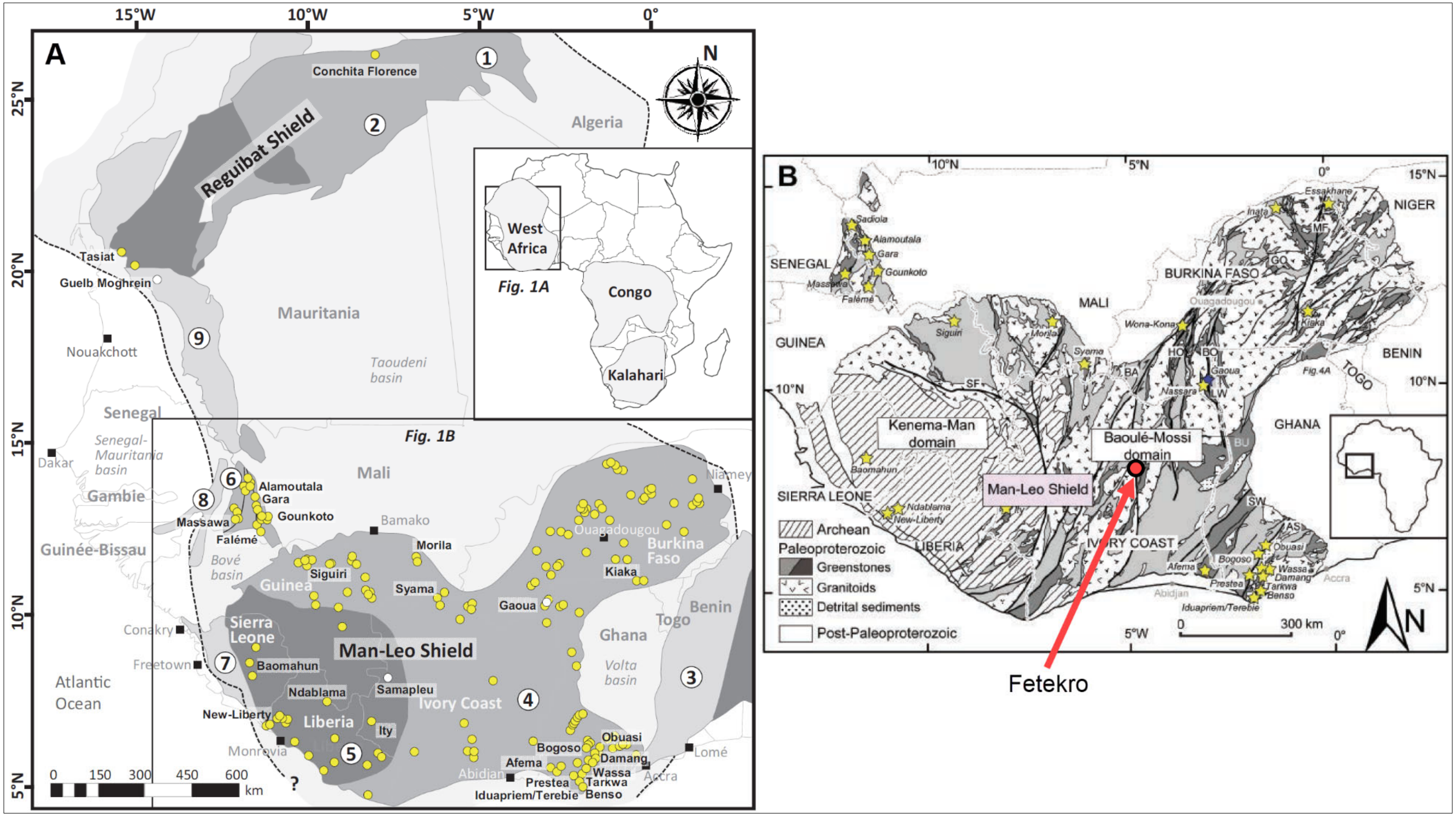


Figure 6-1: **(A) Schematic map of the West African craton (dashed line) showing the distribution of gold deposits (yellow circles) in the context of the various Archean, Proterozoic and Hercynian domains (1 = Eglab; 2 = Yetti; 3 = Daomeyan; 4 = Baoulé-Mossi; 5 = Kenema-Man; 6 = Kédougou-Kénébia Inlier; 7 = Rokelides; 8 = Bassarides; 9 = Mauritanides). (B) Enlargement of the southern West African craton outlining the macro-lithological packages. Gold deposits are denoted by yellow stars. Modified after Goldfarb et al. (2017)**

6.2 Local Geology and Mineralisation

The Lafiguè gold deposit lies within a Birimian age (*ca.* 2.1 Ga), structurally deformed volcanic complex primarily formed of metagabbro-norites and metabasalts, with a lobate granodiorite intrusion towards the west of the deposit. A series of granodiorite and quartz porphyry dykes are spatially associated with the granodioritic intrusive body in the west of the deposit.

Interpretation at Lafiguè is still evolving. The current thinking is that the deposit lies within a compressive relay domain or a transpressive restraining bend, bound by two NNE-trending sinistral shear corridors. These formed at an angle to regional NW-SE directed shortening during the D2 and D3 regional deformation events (Ciancaleoni, 2018). On the deposit scale, gold mineralisation is controlled by a series of ENE-trending shear zones dipping at 10-40° towards the SSE. The mineralised shears crosscut the regional sub-vertical structural fabric which is not mineralised in the deposit area. The shear zones appear to display a reverse sense of shear and have been interpreted to have developed as stacked lenses, predominantly in the hanging wall of a basal thrust which propagates along the lithological contact between mafic volcanic and intrusive units in Lafiguè Centre, and between mafic intrusive and the granodiorite body in Lafiguè Nord (Figure 6-2).

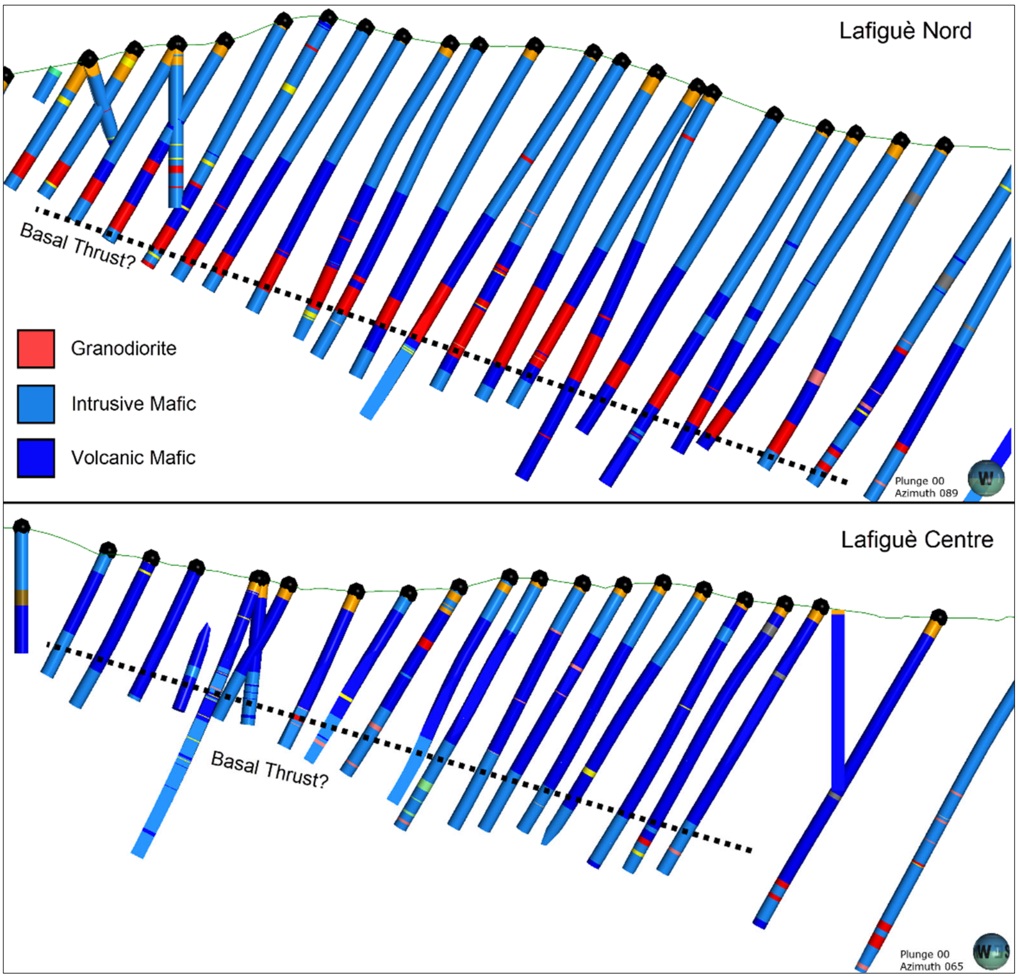


Figure 6-2: Cross-sections showing lithology logging and the interpreted position of a mineralisation-controlling basal thrust along the granodiorite-metagabbro contact in Lafiguè Nord and along the metabasalt-gabbro contact in Lafiguè Centre

Mineralisation is often hosted by quartz-carbonate-tourmaline-pyrite-pyrrhotite-gold veins as well as the associated biotite-tourmaline-sericite-chlorite-carbonate alteration zones (Figure 6-3A), where these veins typically exploit the gently dipping brittle-ductile reverse shear zones. Although the quartz-tourmaline lodes commonly host mineralisation at the primary lithological contacts across the Lafiguè project area, where quartz veins are barren or low grade, they can form planes of rheological contrast which have focussed auriferous fluids along vein contacts, mineralising the hanging wall or footwall rocks (Figure 6-3 B). Gold is also hosted within broader zones of altered, stacked shear zones in the hanging wall (and to a lesser degree, the footwall) of the main lithological contacts (Figure 6-3 C and D). In particular, the entire thickness of the granodiorite body in Lafiguè Nord is often mineralised, including disseminated pyrite, pyrrhotite and gold, along with a similar alteration assemblage to that associated with the quartz lodes. In the broader zones of stacked shears, there is a tendency towards higher grades at the footwall contacts which likely accommodated the greatest strain and associated fluid flow (Figure 6-4).

Figure 6-3: Photos of drillcore from Lafigué showing: (A) A typical high-grade quartz-tourmaline vein with associated alteration (including mineralised quartz selvages) in the hangingwall and footwall foliated metagabbro; (B) Finely disseminated mineralisation hosted within metabasalt footwall of an unmineralized quartz vein; (C) Gold mineralisation hosted as fine disseminations and within narrow quartz stringers (white circles) within the broader granodiorite package; and (D) Pyrite-pyrrhotite-gold associated with carbonate-sericite-biotite alteration assemblage within the sheared metagabbro unit in Lafigué Centre

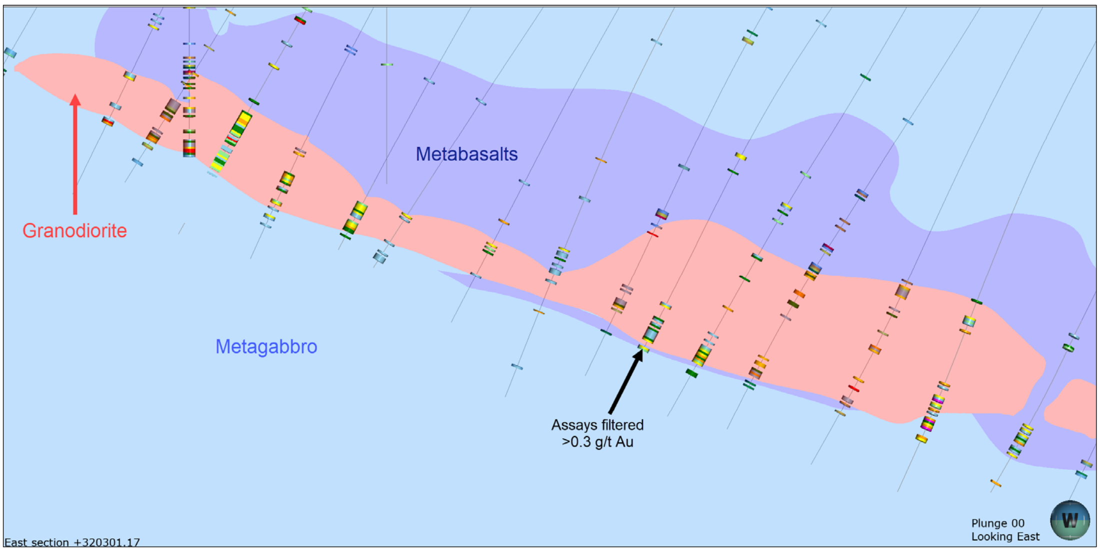


Figure 6-4: Cross-section (looking east) showing the spatial relationship between gold mineralisation (grades filtered >0.3 g/t Au) and the granodiorite intrusion located in Lafigué Nord

The shear zones show a typical C-S geometry, with the CS at an angle to the schistosity fabric due to shearing (Ciancaleoni, 2018). The veins are emplaced in the deformation corridors along both CS and S planes. The shear foliation and majority of veins show shallow-moderate dips towards the S or SSE. Mineralised quartz-carbonate-tourmaline veins typically exploit the stacked shear zones which cross-cut the weak, steep regional (Birimian) foliation, which dips at 65-75° S. The veins often demonstrate crack-seal and crackle breccia textures (Figure 6-3 A). Shear movement and the subsequent auriferous fluid ingress was focussed along lithological contacts providing the greatest zones of competency contrast, though in many areas, altered and mineralised shear zones extend into the hanging wall and footwall lithologies. These often-producing broad zones of disseminated mineralisation (Figure 6-5) as opposed to narrow, highly restricted mineralisation domains as observed elsewhere in the Birimian Au deposits of West Africa.

Figure 6-5: Typical broad mineralised intercept around a central quartz vein. Shearing was focused along the lithological contact between metagabbros and metavolcanics, with the zone of mineralisation directly correlating with the most intense shear fabric and associated alteration (Source: Ciancaleoni, 2018)

Mineralisation at Lafigué has been interpreted to have a strike length of approximately 2 km, trending ENE and dipping moderately to the SSE. Mineralisation has been intersected to depths of approximately 440 m below the surface in Lafigué North, which is approximately 700 to 900m of down-dip extension. The continuity of the mineralisation reduces towards Lafigué Centre and to the south and west. The deposit remains open at depth and in some areas, along strike.

7 DEPOSIT TYPES

The West African Lower Proterozoic greenstone belts are often referred to as Birimian Greenstone Belts and this includes a collection of Paleoproterozoic metasedimentary and metavolcanic units and associated intrusive complexes that are the dominant hosts of gold mineralisation in West Africa. The Birimian Greenstone Belts host multiple world class gold deposits situated within countries including Côte d’Ivoire, Burkina Faso, Ghana, Guinea, Mali, Niger and Senegal. These deposits can broadly be classified into the following types:

- Structurally-controlled, epigenetic lode or stockwork style mineralisation related to major shear zones with native gold (Poura, Burkina Faso; Kalana, Mali);
- Structurally-controlled, epigenetic lode or stockwork mineralisation related to major shear zones and characterised by the inclusion of gold in the crystal structure of the sulphides, often locked in arsenopyrite (Ashanti type: Obuasi, Ghana);
- Stratiform deposits hosted in tourmalinised turbidites (Gara Deposit Loulo, Mali);

- Disseminated sulphides hosted in volcanic or plutonic rocks (Syama, Mali; Yaouré, Côte d'Ivoire; granitoid-hosted, Ayanfuri, Ghana); and
- Paleo-placer deposits: Auriferous quartz-pebble conglomerates (Tarkwa, Ghana) and modern placers (eluvial, alluvial).

More specific to the Birimian, two major styles of gold mineralisation occur, which include:

- structurally controlled quartz vein style deposits; and
- chemical sediment hosted deposits

The Lafigué deposit resembles a typical shear zone-hosted deposit of the West African Paleoproterozoic greenstone terrane (Man-Leo Shield). The deposit is associated with the N-S-trending Oumé-Fetekro greenstone belt, and more specifically with a Birimian age complex of bimodal metavolcanics and meta-volcanoclastic rocks intruded by a series of felsic intrusions. Mineralisation is spatially and genetically related to shearing and fluid ingress along zones of competency contrast between different lithologies. There is a further spatial relationship between some mineralisation and felsic intrusive bodies. Gold is often free, occurring in quartz-carbonate-tourmaline veins or associated alteration haloes. Zones of shearing and alteration (mineralised or otherwise) can reach 10s of metres thick, pervading the hanging wall and footwall rocks away from recognised lithological contacts.

8 EXPLORATION

8.1 Summary

Following acquisition of the Fétékro permit area, Endeavour Mining Corporation commenced exploration activities in March 2017 to better understand the structural framework of the property and define and rank exploration targets. In 2017, an airborne vertical tilt-angle derivative ("VTEM") survey was flown across the permit area, to better define the regional structures. The survey area was flown in a northwest to southeast (N135°) direction with a traverse line spacing of 150 m, at a mean altitude of 84 m above the ground. A total of 1,858-line kilometres of geophysical data was acquired during the survey over an area of 257 km2. A structural interpretation of the VTEM survey data (Ciancaleoni, 2018) highlighted four tectonic domains (Figure 8-1), which included:

- Western tectonic domain marked by N020 sinistral shear zone and N040 regional foliation;
- Central tectonic domain, a transitional domain;
- Compressive relay domain marked by ENE trust; and
- Eastern tectonic domain, similar to western domain (sinistral N020 shear zone).

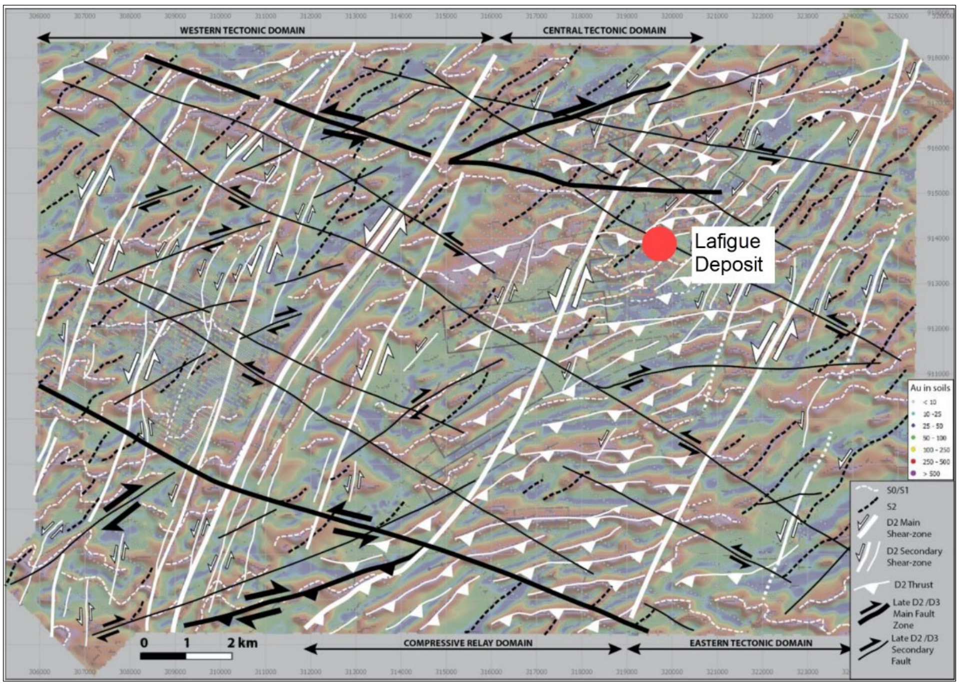


Figure 8-1: Interpreted structural framework across the Fétékro permit area based on a tilt-angle derivative (VTEM) image (SAGAX, 2017) overlain with in-soil gold values (Source: Ciancaleoni, 2018)

Geological mapping, regolith mapping and surveying of some historical artisanal works were carried out during this period in order to establish relationships between airborne geophysical survey anomalies and geological field observations. Additionally, a total of 73 grab samples were collected and assayed.

8.2 Nearby Targets

Based on the exploration completed, several targets in the wider Lafigué area were identified (Figure 8-2) and follow-up work was conducted.

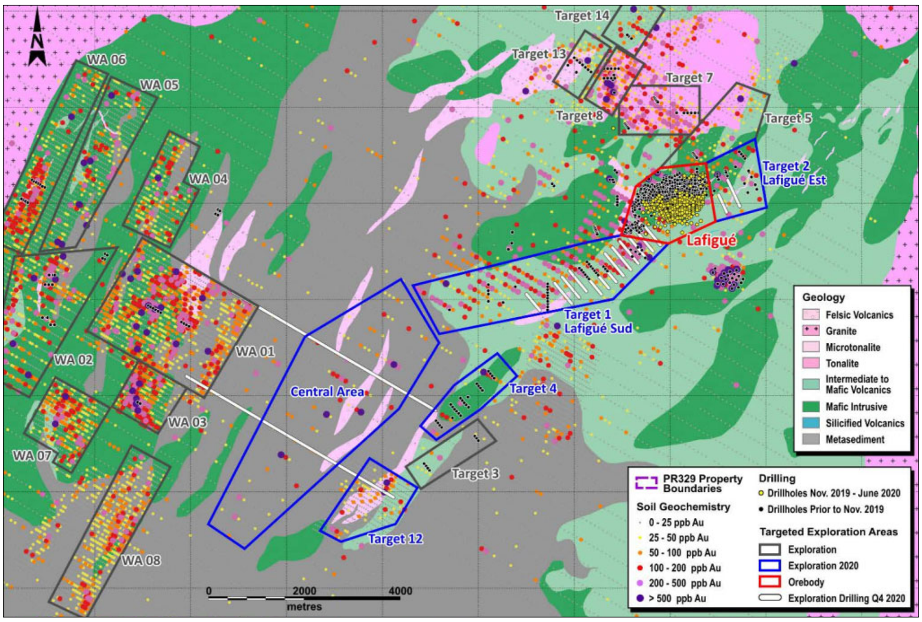


Figure 8-2: Plan map highlighting various exploration targets identified within the Fétékro permit area (Source: Endeavour Mining)

Several targets near to Lafigué and showing similar geology and in-soil gold anomalies were tested by wide spaced drilling, including Target 1 (Lafigué Sud), WA 01, Target 2 and Target 4. IP pole-dipole and gradient surveys were carried out on Lafigué North, Target 2 and Target 5 to better understand the mineralised structures and to delineate any extensions to the Lafigué Nord deposit. The IP anomaly on Target 2 was subsequently drilled, however no significant mineralisation was found.

Given the focus of exploration on Lafigué, relatively little exploration has been carried out on the targets not immediately adjacent to the main mineralisation. The identified targets include several gold in-soil anomalies to the west of Lafigué, denoted by the acronym “WA” (Figure 8-2). Limited exploration drilling of some of these anomalies, such as WA 01 and WA 06, identified mineralisation associated with NNE-trending sub-vertical shear zones with quartz veins and associated alteration haloes.

During 2019, LMCI conducted a regional soil geochemical survey on the central part of the permit as well as some areas in the west of the permit where existing anomalies (>50 ppb Au) had been identified. A total of 3,469 samples were taken, resulting in the delineation of five new targets in the Central Area. These are likely aligned along the regional NNE-trending structural fabric. RC reconnaissance drilling is planned for several targets within the central portion of the Fétékro permit area, including the Target 12 gold in-soil anomaly (Figure 8-2).

9 DRILLING

9.1 Endeavour Drilling (2017-Present)

Drilling at Lafigué since 2017 has comprised of five separate campaigns aiming to delineate the full down-dip and along-strike extent of mineralisation, as well as increase confidence in the geological and grade continuity through infill drilling. Additionally, some drillholes have been completed for various technical studies as part of the pre-feasibility study, including geotechnical studies and metallurgical testwork. A small number of sterilisation holes have been completed for mine planning purposes. Table 9-1 summarises all the drilling completed under Endeavour ownership from 2017 until the present MRE data cut-off date (15 May 2022). The drilling is also illustrated in Figure 9-1.

Table 9-1: Summary of drilling completed across PE 58 and PR 329 between 2017 and May 2022

Period	Type	Number	Metres	Drilling Contractor
2017	DDH	17	2,197	FORACO
	RC	179	12,464	FTE
2018	DDH	21	3,861	FORACO- GEODRILL
	RC	105	14,647	GEODRILL
	RC-DD	8	2,662	GEODRILL
2019	DDH	15	2,543	FORACO- GEODRILL
	RC	228	37,633	GEODRILL
	RC-DD	27	7,804	GEODRILL
2020	RC	169	35,941	GEODRILL
	RC-DD	130	41,556	FORACO- GEODRILL
2021	RC	416	62,572	GEODRILL
	DD	5	1,468	GEODRILL
	RC-DD	61	19,844	GEODRILL
2022	RC	222	25,000	GEODRILL
	RC-DD	7	1,310	GEODRILL

DD = Diamond core drilling; RC = Reverse Circulation drilling; RC-DD = Reverse circulation with a diamond core tail
Includes drilling across all permit areas, and for all purposes, including sterilisation, hydrology, geotechnical and geometallurgical testwork purposes

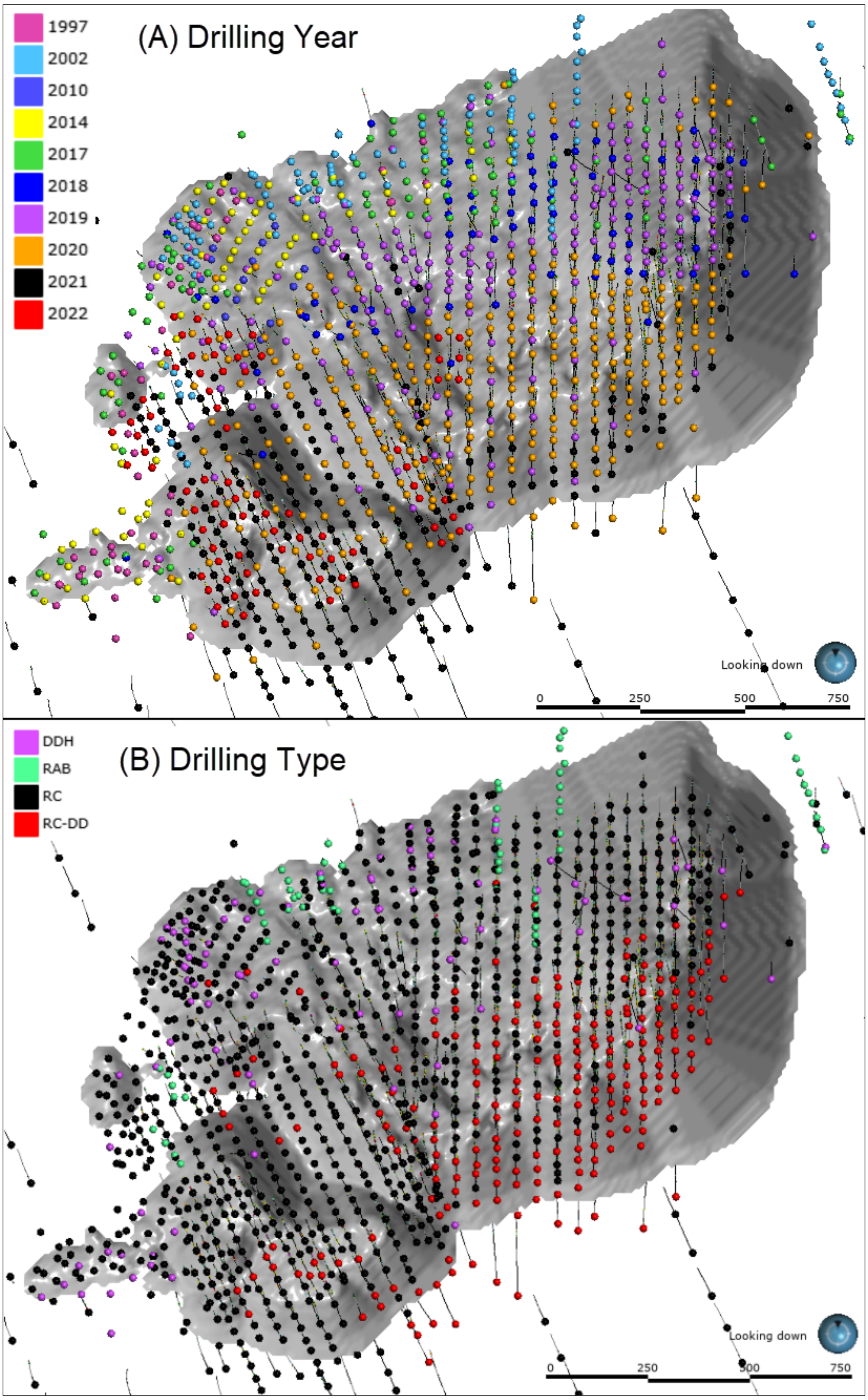


Figure 9-1: Plan views of the Lafigué prospect area and the 2022 optimised pit shell outline showing drillhole collars coloured by: (A) drilling campaign (year); and (B) drilling type

9.2 Drilling Methods

Drilling conducting during Endeavour's ownership of the Project has been carried out under the supervision of technically qualified personnel applying standard industry approaches. The drilling contractors used for each program are detailed in Table 9-1, with the current contractor being GEODRILL. Dr Lucy Roberts and Dr James Davey of SRK observed RC drilling practices during the site visit in May 2021, where drilling procedures appeared to be in line with industry best practice (Figure 9-2).

Figure 9-2: RC drill rig (hole ID: LFRC21-1405) and associated sampling setup observed by SRK personnel during their visit to Lafigué in May 2021

Drilling is carried out in two 12-hour shifts per drill rig, operating 6 days per week. A geologist supervises each drillhole, with geological technicians and other associated workers allocated to each drill rig for sampling purposes.

The paper logs for the majority of drillholes completed prior to 2017 have been located, reviewed, and digitised. SRK has not been able to confirm the drilling operating procedures for historical drilling campaigns (pre-2014).

9.3 Core Recovery

Core recovery has been recorded for all diamond core drilling at Lafigué since the 2010 drilling campaign. Core recovery was measured based on the length of core recovered relative to the length of each core run, with a global average of 98.3% recovery across the eight drilling campaigns since 2010 (Figure 9-3). Histograms and associated summary statistics for core recovery, broken down by drilling campaign are provided in Appendix B.

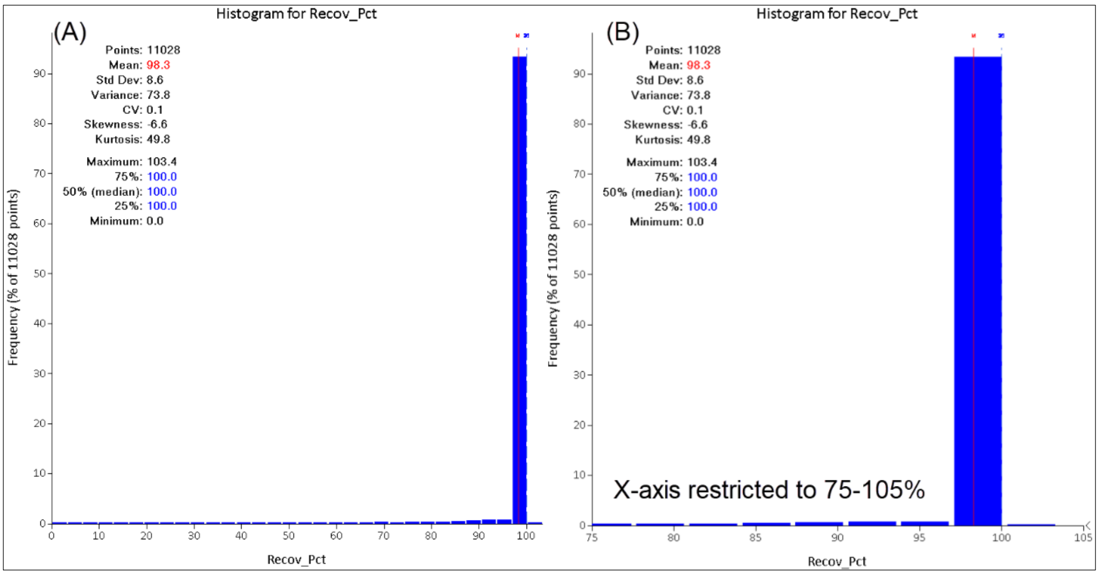


Figure 9-3: Histograms showing diamond core recovery (%) for the global drillhole database (2010 – 2022). Histogram (B) shows the same data with the X-axis (% recovery) restricted to 75-105%. Stated statistics exclude seven samples with recorded recoveries >105%.

9.4 Drillhole Surveying

9.4.1 Collar surveys

For all drillholes completed since 2017, collar surveys were conducted by Société Nationale de Topographie (SNT) using a differential GPS. Due to some accessibility issues associated with the COVID-19 pandemic, drillholes completed during 2020 were surveyed by Cabinet Kouamelan using a differential GPS. With the exception of some very minor corrections in the elevation of some drillhole collars, which SRK set to the topographic surface provided, no material issues were identified in the collar survey information.

9.4.2 Downhole surveys

Downhole surveys were completed in each drillhole using geographic north as a reference azimuth (magnetic declination: -3.4° in 2020). Table 9-2 summarises the downhole survey tools used by each contractor. Since 2019, with the drilling of deeper holes to test down-dip extensions, stabiliser rods have been used to better control downhole deviations.

Table 9-2: Summary of downhole survey tools used, split by drilling campaign / contractor

Year	Drilling Contractor	Downhole Survey Instrument
2017	FTE	Gyro
2017	FORACO	Reflex-EZ track
2017	GEODRILL	Reflex-EZ track
2018	FORACO	Reflex-EZ track
2018	GEODRILL	Reflex-EZ track + EZ-Gyro
2019	FORACO	Reflex-EZ track
2019	GEODRILL	EZ-Gyro + SPRINT Gyro
2020	FORACO	Reflex-EZ track
2021	GEODRILL	EZ-Gyro + SPRINT Gyro
2022	GEODRILL	EZ-Gyro + SPRINT Gyro

9.5 Logging and Photography

Diamond core drillholes are geotechnically logged and photographed at the drilling site, along with the marking up of an orientation line, where competent, oriented core has been recovered. Drillcore and RC chips are then transported to an LMCI sampling facility where detailed geological, structural and weathering logging takes place. Each drillcore log includes:

- Lithology:
  - Rock code(s)
  - Sulphide intensity
  - Carbonate intensity
- Alteration:
  - Alteration mineralogy and intensity
- Oxidation:
  - Oxide, Transition (“oxide-sulphide”), Sulphide
- Weathering:
  - Weathering code (LATR, MTLZ, SAPR, SAPRK, OVBD, NRCV, BDRK)
- Structure:
  - Structure code (qualitative observation)

9.6 Drillhole Orientation Relative to Mineralisation

The majority of DD and RC holes at Lafigué are grid-drilled, dipping at 50° or 60° towards 000° or 335°. Mineralisation typically dips at approximately 20° towards the S/SSE, resulting in drilling intersection angles of 90 to 110°. Overall, SRK considers drillhole orientations relative to mineralisation to be suitable to support the Mineral Resource estimate presented herein.

9.7 Drillhole Quantity and Spacing

The majority of DD and RC holes at Lafigué have been completed on a 20 to 40 m by 50 m grid, with some areas of closer drilling towards the up-dip portions of the deposit and wider spaced drillholes in down-dip areas (Figure 9-1).

The Lafigué drillhole database contains a total of 1,189 DD, DD-RC and RC exploration holes, the majority of which support the main area modelled in support of the present MRE. Although some areas, particularly in the western and down-dip portions of the deposit, would benefit from further drilling to increase confidence in geological and grade continuity, SRK considers the current drilling database sufficient to support the Mineral Resource estimate presented herein.

9.8 Drilling Summary

Overall, SRK is satisfied that the drilling procedures since 2017, including collar and downhole surveys, logging and sampling generally conform to industry best practice and provide a sound basis for the Mineral Resource estimate presented herein. Much less information is available regarding the drilling procedures associated with the historical drilling (pre-2017). SRK notes that the spatial accuracy and core recovery (pre-2010) from these holes presents a risk when compared to drillholes completed using industry standard operating procedures during the more recent drilling campaigns. This data is typically concentrated along the northern periphery of the deposit (up-dip portions). SRK considers that the risk is mitigated to some extent by the addition of numerous close-spaced holes completed between 2017 and 2019.

10 SAMPLE PREPARATION, ANALYSES AND SECURITY

10.1 Introduction

Much of the below information is summarised from the Lafigué Gold Project Pre-Feasibility Study (Lycopodium), with additional QAQC details pertinent to the 2020 to 2022 drilling campaigns. Sample preparation, analysis and security for the Lafigué Project are currently under the supervision of Endeavour geologists, with LMCI responsible between 2013 and 2016. The following processes and procedures relate to the drilling and sampling campaigns managed by Endeavour since 2017, with similar protocols being followed by LMCI between 2013 and 2016. Information pertaining to historical drilling and sampling prior to 2013 is limited to some QAQC results for the 2010 drilling campaign only, however much of the historical drilling (completed prior to 2010) has been twinned or followed up with close-spaced drilling during the 2014 and 2017 drilling campaigns, run by LMCI and Endeavour, respectively (see Figure 9-1).

10.2 Sampling Methods

10.2.1 RC Sampling

Reverse circulation ("RC") samples were collected in 1 m intervals in bulk bags directly from the cyclone discharge (Figure 10-1 A). Samples were riffle split into a labelled sample bag, producing a representative 2 to 4 kg sample split with a matching sample tag included in each bag. A duplicate 2 to 4 kg sample was retained for reference, alongside a small quantity of representative chips for geological logging purposes. The riffle splitters, sample tubs and other working surfaces were cleaned with compressed air between each sample. The sample rejects were bagged up and either remained at the drill pad or were transported to the sample management facility. The riffle splitting and sampling methodologies are summarised by the flow charts in Figure 10-2 and Figure 10-3.

Figure 10-1: Photographs showing: (A) sample collection from the RC rig cyclone; and (B) riffle splitters used to produce a 2 to 4 kg sample split

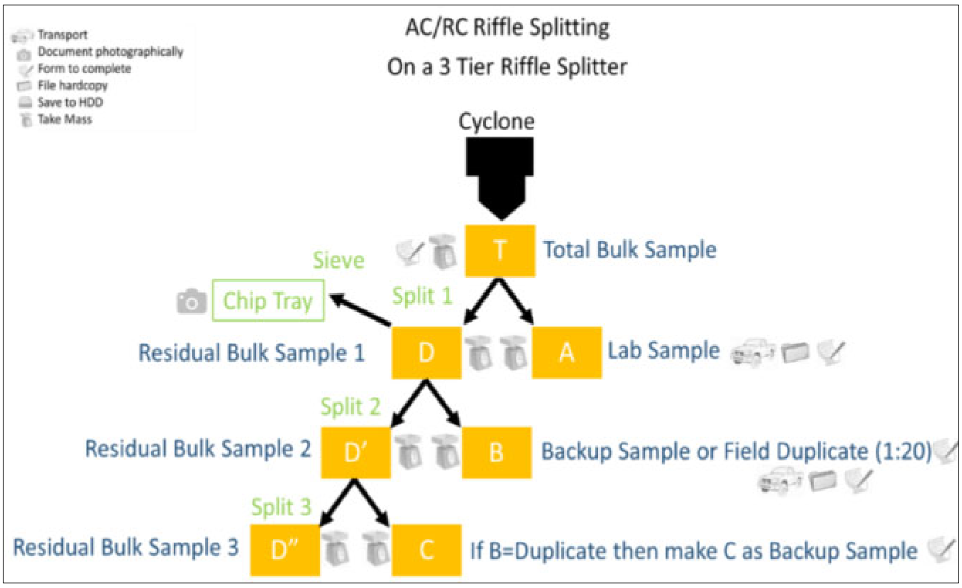


Figure 10-2: Sample splitting methodology using a riffle splitter

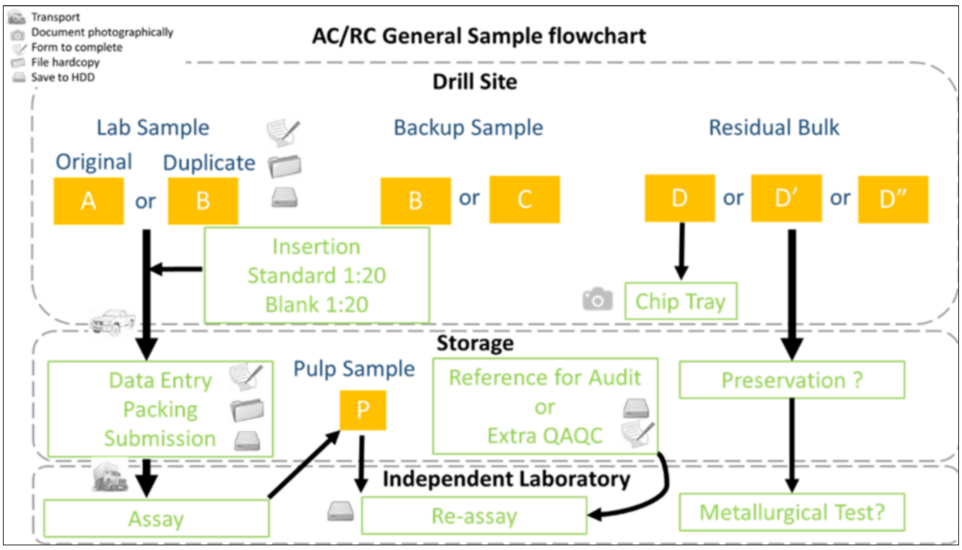


Figure 10-3: RC sampling flowchart

Polyweave bags containing approximately 30 samples were transported to the Bureau Veritas laboratory in Abidjan, Côte d'Ivoire. The Bureau Veritas laboratory is not currently ISO17025 accredited. The laboratory does, however, work under the accreditation of the global Bureau Veritas group of laboratories including Australia and Canada and are covered by the groups ISO9001, ISO14001, ISO18001 and IFIA certificates.

10.2.2 Diamond drilling sampling

Drillcore was placed in steel or timber core boxes, each marked with the borehole ID, and start and end depths for the corresponding core. Orientation lines were drawn on competent lengths of drillcore immediately, and then the core was geotechnically logged and photographed whilst still at the drill site. Core boxes were transported to the LMCI sampling facility where the core was geologically logged, and sampling intervals were marked. Drillcore was cut along its longitudinal axis, with half core samples selected from the right-hand side of each interval (looking down hole). Samples were tagged, bagged and transported to the Bureau Veritas laboratory in Abidjan. The remaining half of each core was retained for reference. Figure 10-4 shows the typical sampling procedures flowchart for DD drillholes.

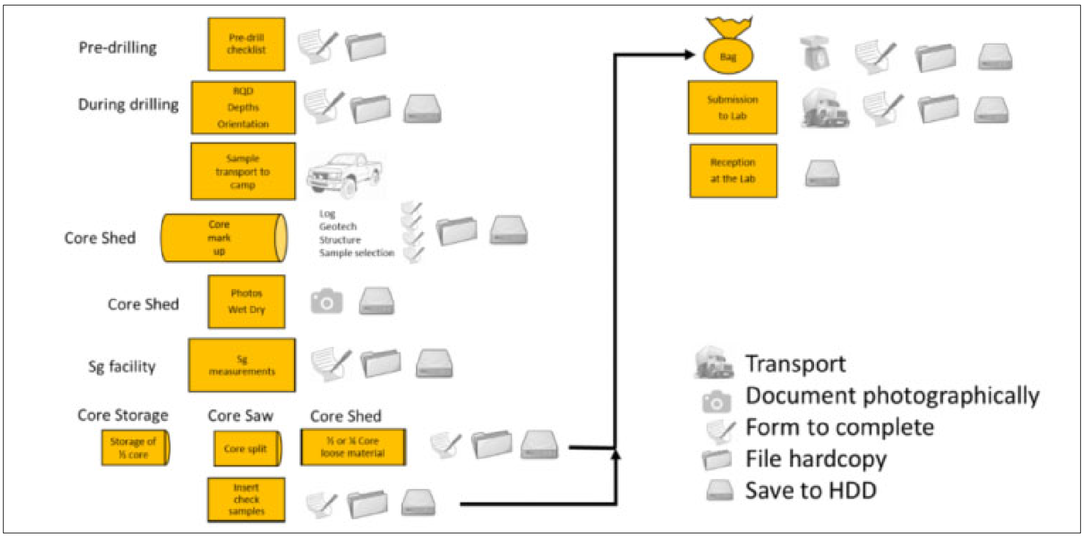


Figure 10-4: DD sampling flowchart

10.2.3 Sample submission

Both RC and DD sample submissions to the Bureau Veritas laboratory in Abidjan were accompanied by a submission form detailing the sample numbers. Bureau Veritas staff cross-referenced the samples received with the submission forms to ensure all samples were received, and then logged the samples in the laboratory information management system ("LIMS").

10.3 Sample Preparation

The sample preparation procedures, applicable to both core and RC samples, undertaken at the Bureau Veritas Laboratory in Abidjan included:

- oven drying at 105°C to 110°C;
- crushing using a jaw crusher such that 75% passes a 2 mm diameter mesh;
- sub-sampling with a riffle splitter;

- pulverisation of approximately 0.5 kg with an LM2 pulveriser such that 90% passes a 75 µm mesh;
- homogenisation of a 250 gram pulp split for transfer to the fire assay circuit.

10.4 Assay Analysis

All samples taken since 2017 were analysed by fire assay with an atomic absorption finish (BV code FA450) using a nominal 50 g charge. Samples returning a grade greater than 10 g/t Au were reanalysed by fire assay with a gravimetric finish (BV code FA550 or FAGRA01).

10.5 Quality Assurance and Quality Control

Quality assurance / quality control (QAQC) sampling programmes are typically designed to identify and assess contamination or bias in the analytical results and allow analytical precision and accuracy to be quantified, providing confidence in the underlying sample data used for the purposes of estimating Mineral Resources.

SRK has reviewed the QAQC sample analysis for the 2017 to 2022 drilling campaigns, including the data previously presented in the Lafigué Gold Project Pre-Feasibility Study, as well as completing an analysis of QAQC results provided for the 2010 and 2014 drilling campaigns prior to LMCI/Endeavour ownership. SRK was not provided with any QAQC sample results for earlier drilling (1997 and 2002 drilling campaigns) and as such, assay data from these drilling campaigns present a risk in terms of accuracy and precision of the associated assay grades. Given the relatively small proportion of drilling (<8% of total RC + DD drillholes used for the MRE), SRK considers it reasonable to include these data in the MRE presented herein.

A summary of QAQC sample insertion rates for drilling between 2010 and 2022 is provided in Table 10-1. The below sections summarise the results of the QAQC review for drilling conducted between 2010 and 2022.

Table 10-1: Summary of QAQC samples inserted during the 2010 to 2022 Lafigué drilling campaigns

Sample Type	Drilling Campaign								Total	%
	2010	2014	2017	2018	2019	2020	2021	2022*		
Regular samples	1,661	6,497	6,164	13,005	38,919	73,585	81,288	22,073	243,192	100%
Blank	12	93	92	61	123	209	424	215	1,229	0.51%
Blank (coarse)	12	428	191	535	1,683	3,212	3,356	812	10,229	4.21%
CRM Combined	36	435	360	744	2,250	2835	4,730	1,282	12,672	5.21%
G300-8			115	5					120	0.05%
G302-3								12	12	
G307-2	2	144							146	0.06%
G310-6				244					244	0.10%
G310-8			115						115	0.05%
G910-10								12	12	
G311-2	2	145							147	0.06%
G316-2			3	167					170	0.07%
G318-10								52	52	
G910-8	4	146	10	5					165	0.07%
G913-3							160	402	562	0.23%
G913-9			117	236	749	1 ,439	1,173		2,275	0.94%
G914-2				59	750	1,424	1,718	391	4,342	1.79%
G915-6				28	751	1,411	1,679	413	4,282	1.76%
G998-8	23								23	0.01%
Std-UNKN	5								5	0.00%
Field duplicates	-	-	394	796	2,511	4,312	4,794	1,284	14,091	5.79%
Pulp duplicates	47	509	-	-	-	-	-	-	556	0.23%
Total QAQC Samples	**107**	**1,465**	**1,037**	**2,136**	**6,567**	**10,568**	**13,304**	**3,593**	**38,777**	15.95%

*Drilling completed up until end April 2022

10.5.1 Certified reference materials (CRM)

CRM are samples that can be used to measure the accuracy of analytical procedures and are composed of material that has been thoroughly analysed and certified by several laboratories to accurately determine its grade within known error limits. The CRM used at the Lafigué Project were sourced from Geostats and covered a grade range of 0.63 to 5.85 ppm Au. Since 2019, three CRM (G913-9, G914-2 and G915-6) have primarily been used, covering a grade range of 0.67 to 4.91 ppm Au. In total, the CRM insertion rate between 2010 and 2022 drilling was 5.2%, with an overall failure rate of <1% (outside ±3 SD). Based on a review of CRM failures, the majority can likely be attributed to inadvertent CRM sample swaps. SRK recommends that procedures are updated to minimise the risk of future CRM sample swaps at the sample management facility. SRK notes that CRM G310-6 had some issues systematically under-reporting Au by approximately 0.05 ppm (8%) between 2017 and 2018, however in the context of none of the other CRM, including another CRM in the same grade range (G910-8) significantly and systematically under-reporting Au grade, and of the CRM in question having been retired in 2018, SRK do not consider this issue material to the accuracy of assay results which form the basis of the MRE. Additionally, gold was systematically under-reported by approximately 0.1 ppm (10%) for CRM G910-10 during the 2022 drilling programme up until 12 May 2022 (a total of 12 sample submissions), whereafter performance of the CRM abruptly improved to be approximately aligned with the certified value for this material (0.96 ppm). After correspondence with the Endeavour database manager, it is considered likely that the earlier under-performance of this CRM may be attributed to a systematic mislabelling of a batch of these CRM with an alternative, lower grade CRM (G913-1 – 0.82 ppm Au) which was used on a separate exploration programme. SRK does not consider this issue material to the accuracy of assay results which form the basis of the MRE, however it is recommended that sequences of inaccurate results for a given CRM are monitored more closely such that these may be investigated more promptly in future.

A summary of CRM sample performance, split by drilling campaign and drilling type, is provided in Table 10-2, with all CRM control plots presented in Appendix C.

Table 10-2: Summary of CRM performance, split by drilling campaign and type

Year	Drilling Type	CRM	Number of Submissions	Gold Grade (g/t)	Standard Deviation	Number of Failures	Failure Rate
2010	DD	G307-2	2	1.08	0.05	0	0.0%
		G311-2	2	4.93	0.18	0	0.0%
		G910-8	4	0.63	0.04	0	0.0%
		G998-8	3	5.85	0.39	0	0.0%
	RC	G998-8	20	5.85	0.39	1	5.0%
2014	DD	G307-2	38	1.08	0.05	0	0.0%
		G311-2	38	4.93	0.18	0	0.0%
		G910-8	38	0.63	0.04	0	0.0%
	RC	G307-2	106	1.08	0.05	1	0.9%
		G311-2	107	4.93	0.18	0	0.0%
		G910-8	108	0.63	0.04	0	0.0%
2017	DD	G300-8	12	1.07	0.06	0	0.0%
		G310-6	4	0.65	0.04	0	0.0%
		G316-2	3	1.04	0.04	0	0.0%
		G910-8	10	0.63	0.04	0	0.0%
		G913-9	12	4.91	0.17	0	0.0%
	RC	G300-8	103	1.07	0.06	1	1.0%

Year	Drilling Type	CRM	Number of Submissions	Gold Grade (g/t)	Standard Deviation	Number of Failures	Failure Rate
		G910-8	103	0.63	0.04	1	1.0%
		G913-9	105	4.91	0.17	0	0.0%
2018	DD	G310-6	66	0.65	0.04	1	1.5%
		G316-2	36	1.04	0.04	2	5.6%
		G913-9	59	4.91	0.17	1	1.7%
	RC	G300-8	5	1.07	0.06	0	0.0%
		G310-6	126	0.65	0.04	1	0.8%
		G316-2	84	1.04	0.04	2	2.4%
		G910-8	5	0.63	0.04	0	0.0%
		G913-9	133	4.91	0.17	1	0.8%
		G914-2	59	2.48	0.08	0	0.0%
		G915-6	28	0.67	0.04	0	0.0%
	RC-DD	G310-6	52	0.65	0.04	0	0.0%
		G316-2	47	1.04	0.04	2	4.3%
		G913-9	44	4.91	0.17	0	0.0%
2019	DD	G913-9	19	4.91	0.17	0	0.0%
		G914-2	13	2.48	0.08	0	0.0%
		G915-6	14	0.67	0.04	0	0.0%
	RC	G913-9	587	4.91	0.17	1	0.2%
		G914-2	583	2.48	0.08	2	0.3%
		G915-6	598	0.67	0.04	1	0.2%
	RC-DD	G913-9	143	4.91	0.17	0	0.0%
		G914-2	154	2.48	0.08	0	0.0%
		G915-6	139	0.67	0.04	0	0.0%
2020	RC	G913-9	697	4.91	0.17	1	0.1%
		G914-2	679	2.48	0.08	0	0.0%
		G915-6	669	0.67	0.04	0	0.0%
	RC-DD	G913-9	742	4.91	0.17	2	0.0%
		G914-2	745	2.48	0.08	0	0.0%
		G915-6	742	0.67	0.04	2	0.3%
2021	RC	G913-3	43	2.36	0.18	0	0.0%
		G913-9	503	4.91	0.17	0	0.0%
		G914-2	693	2.48	0.08	0	0.0%
		G915-6	651	0.67	0.04		0.0%
	RC-DD	G913-3	70	2.36	0.18	0	0.0%
		G913-9	147	4.91	0.17	0	0.0%
		G914-2	398	2.48	0.08	0	0.0%
		G915-6	405	0.67	0.04	0	0.0%
2022	RC	G913-3	375	2.36	0.18	0	0.0%
		G914-2	367	2.48	0.08	0	0.0%
		G915-6	389	0.67	0.04	1	0.3%
		G318-10	49	4.58	0.17	0	0.0%
		G302-3	12	2.33	0.12	0	0.0%
		G910-10*	12	0.97	0.04	2	16.7%
	RC-DD	G913-3	27	2.36	0.18	0	0.0%
		G914-2	24	2.48	0.08	0	0.0%
		G915-6	24	0.67	0.04	0	0.0%
		G318-10	3	4.58	0.17	0	0.0%

*G910-10 samples prior to 12 May 2022 considered likely to be sample swaps/mislabelled – see explanation in text.

10.5.2 Blank samples

The insertion of blanks is intended to identify if there has been contamination during the sample preparation process. Two types of blanks have been used at Lafigué:

- Coarse crush blank – granite sourced from a quarry near Abidjan
- Fine blank – a fine fluvial sand sourced from a river in Abidjan

Both blank materials are reported to have been tested at multiple laboratories within Côte d'Ivoire.

During the 2010 to 2022 drilling campaigns, coarse and fine blank samples were inserted at overall insertion rates of 0.5% and 4.2%, respectively, broadly in-line with Endeavour's policy of at least one blank sample insertion per 30 regular submissions. Endeavour considered a grade greater than 10x the limit of detection (i, e. >0.05 ppm) a failure. Overall, the performance of the blank samples was excellent between 2010 and 2022. Although approximately 10-15% of blanks samples returned Au grades greater than the detection limit (>0.005 ppm), none of these were greater than 0.05 ppm and therefore considered a failure. An example blank control plot is shown in Figure 10-5, with plots from each drilling campaign between 2010 and 2022, split by drilling type, presented in Appendix C.

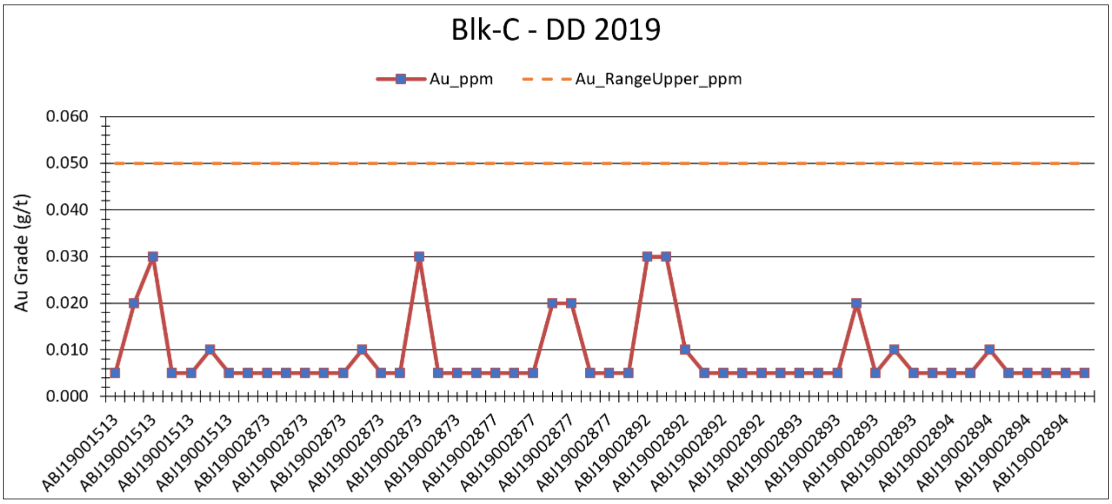


Figure 10-5: Example coarse blank control chart showing blank sample grades for the 2019 DD drilling campaign

10.5.3 Duplicate samples

The precision of sampling and analytical results can be measured by re-analysing a portion of the same sample using the same assay methodology. The variance between the original and duplicate result is a measure of the precision.

Precision is affected by mineralogical factors such as grain size and distribution and inconsistencies in the sample preparation and analysis processes. There are a number of different duplicate sample types which can be used to determine the precision for the sampling process, sample preparation and analyses. Field duplicates assess the variability of two samples taken across the same interval, indicating the overall repeatability of the assayed results. Field duplicates can also help detect sample number mix-ups and assess the natural local-scale grade variation or nugget effect. A relatively small number of pulp duplicates were inserted into the sample stream during the 2010 and 2014 drilling campaigns in order to assess laboratory precision.

In total, some 14,647 (14,091 field and 556 pulp/laboratory) duplicate samples were submitted for analysis during the 2010 to 2022 drilling campaigns, equating to insertion rates of approximately 5.8% and 0.2% for field and pulp duplicates, respectively.

In general, excluding a small number of anomalous or very high-grade results in the coarse material, the duplicate samples show a reasonable degree of correspondence with original samples. The coefficients of determination, i.e., R^2 values, are listed in Table 10-3, and are typically within specified failure limits. An example duplicate control plot is shown in Figure 10-6, with the remaining plots, split by drilling type and campaign, presented in Appendix C.

The 2014 DD laboratory duplicate results show a relatively poor degree of correspondence (R^2 = 0.41) and there is little documentation detailing potential sources of imprecision in the sampling. Given the relatively small number of 2014 DD drillholes supporting the MRE (24 drillholes, or <2% of the total supporting drillholes), SRK does not consider this to be a material issue. However, SRK does recommend that, where this material is still available, some duplicate sample material be re-analysed to better assess the precision of these assay results.

Table 10-3: R^2 values of duplicate sample pair populations, split by drilling campaign and type between 2010 and 2022

Year	Drilling Type	Dup Type	R^2	No. Excluded Anomalous Results
2010	DD	Pulp	0.57*	3
	RC	Pulp	0.99	-
2014	DD	Pulp	0.41	-
	RC	Pulp	0.99	3
2017	DD	Field	0.98	-
	RC	Field	0.96	16
2018	DD	Field	0.90	3
	RC	Field	0.94	-
	RC-DD	Field	0.99	-
2019	DD	Field	0.99	-
	RC	Field	0.96	3
	RC-DD	Field	0.98	-
2020	RC	Field	0.98	-
	RC-DD	Field	0.92	-
2021	RC	Field	0.99	-
	RC-DD	Field	0.93	-
2022	RC	Field	0.99	-
	RC-DD	Field	0.99	-

*Only 10 duplicate pairs analysed

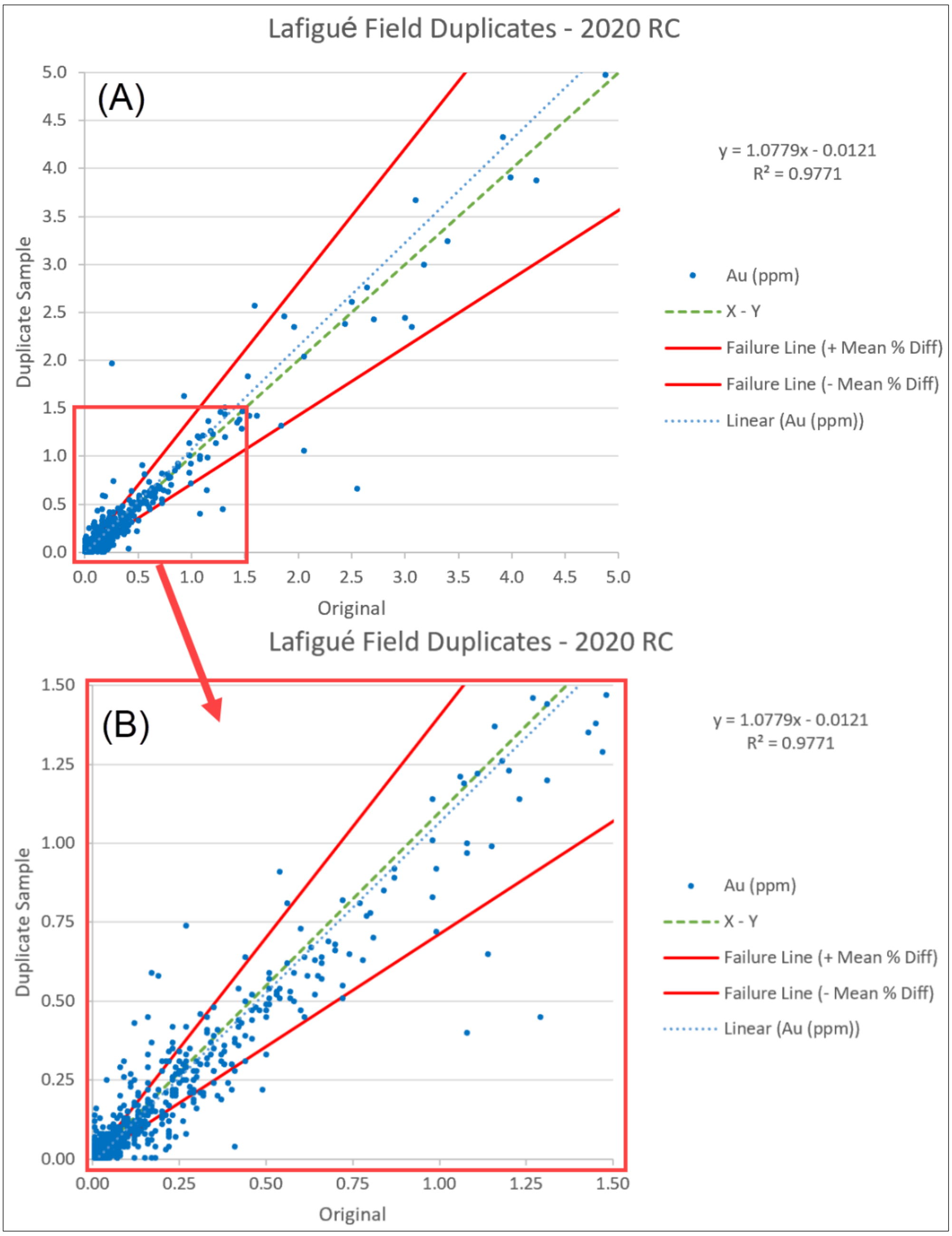


Figure 10-6: Example field duplicate control plot for duplicate sample pairs sampled during the 2020 RC drilling campaign at Lafigué. Plot (B) shows the same data with axes restricted to 1.50 g/t Au

SRK note that selected duplicate samples provide reasonable coverage in context of the average mineralisation domain grades, ranging from below detection limit into the tens-of-ppm in most drilling and sampling campaigns. Full duplicate charts are presented in Appendix C.

10.6 Density Analysis

The density database supplied to SRK includes a total of 2,667 measurements taken between 2014 and 2021. Density determinations were carried out using drillcore samples representing the full range of lithologies and weathering intensities present at the Project. Competent sections of core (160 g to 1,000 g in mass) were cut and dried in the sun for 2 days prior to measurements being taken.

Sample densities were measured on-site using the Archimedes principle of first weighing the sample dry, and then submerged in water within a wax/plastic coating. Moisture content was not measured and is assumed to be negligible following drying of the sample. The following equation was used to generate specific gravity, which at room temperature correlates to density:

$$Archimedes\ SG\ = \frac{\text{Weight of sample (g)}}{Weight\ in\ air\ (g) -\ weight\ in\ water\ (g)}$$

The average density values applied for tonnage estimation, split by lithology, are shown in Table 13-9.

10.7 Chain of Custody and Sample Security

All DD and RC samples for analysis were transported to Endeavour's secure (walled and lockable) sample management facility where geological logging and QAQC sample insertion took place. Sample batches were placed in to sealed and numbered polyweave or plastic bags for transport. Each sample shipment is verified by Endeavour personnel during loading, and a verification document is signed by Endeavour and laboratory staff prior to departure from the sample management facility. Upon receipt of the sample shipment at the laboratory, laboratory personnel verified the sample inventory before sample preparation begins. All aspects of the sample collection and dispatch were conducted by Endeavour personnel, or under the supervision of Endeavour personnel.

10.8 Summary

Overall, SRK considers the majority of sample preparation, analyses and security protocols to conform to industry best practice. SRK notes the absence of QAQC sample results for the 1997 and 2002 drilling campaigns and as such, assay data from these drilling campaigns present a risk in terms of accuracy and precision of the associated assay grades. The results of the 2010-2022 QAQC programmes are summarised as follows:

Accuracy:

The results for a range of 15 different certified reference materials submitted for analysis between 2010 and 2022 are generally acceptable, with some minor CRM sample swapping issues identified.

Precision:

A total of 14,091 field duplicates were inserted into the sample stream between 2017 and 2022, generally returning reasonable correlations between original and duplicate samples. SRK is not aware of any investigation into whether assay disparities are primarily attributable to the natural heterogeneity of the deposit, and therefore would be expected to reduce with increasingly homogenised coarse and pulp duplicate sampling, or whether these disparities represent a material issue related to sampling protocols. SRK recommend taking and submitting coarse crush and pulp duplicates in future drilling campaigns in order to assess precision through the entire sample preparation and analysis process.

Contamination:

A total of 11,458 coarse and fine blank samples were analysed between 2010 and 2022 using conventional fire assay analysis. No material issues with contamination were noted.

The QAQC analyses presented are generally of sufficient quality to support the subsequent geological modelling and grade and tonnage estimate.

11 DATA VERIFICATION

11.1 Historical Data Validation and Verification

Upon acquisition of the Project in 2016, Endeavour implemented a SQL-based database management system ("DBMS"), where all historical data generated from the Fetekro Project was audited during the importation process. Errors screened during this process included:

- Inconsistent collar coordinates;
- incorrect or missing down hole survey records;
- missing sample assay records; and
- missing or overlapping downhole interval records.

SRK did not observe any material issues with the historical drillhole data provided, however does note the absence of QAQC sampling during these periods (prior to 2010), as discussed in Section 10.5.

11.2 Database Checks and Independent Verification

Since 2013 all data acquired across the Fetekro Project area is managed using the built-in data integrity requirements of an industry standard SQL-based DBMS, where database checks include identifying:

- inconsistent collar coordinates;
- incorrect or missing DTH survey records;
- missing assay records;
- missing data or overlapping interval errors; and
- incorrect 3D plotting of drillhole traces.

Any errors highlighted during this process are actioned by the Endeavour database management team as appropriate.

Prior to the exportation of a final database from the DBMS, an audit is undertaken by the central database team within the DBMS. Additional checks are completed by the database management team using the software-based auditing tools provided in the Geosoft Target package.

While SRK has not independently verified the database management procedures carried out by Endeavour, a review of the exported database did not highlight any major issues. Adjustments made to the drillhole database for use in the MRE are summarised in Section 13.2. In addition to the above database verification procedures carried out by the Endeavour database management team, SRK has cross-checked a selection of assay results in the drillhole database with their corresponding original laboratory assay certificates and identified no significant issues.

11.3 Twinned Hole Comparison

No twinned drillholes have been completed at Lafigué, however a limited number of pairs of drillholes each spaced within 4 m of each other, does allow some short-scale comparisons to be made. A statistical comparison of samples within the estimation domains in three examples of these pairs of drillholes (Table 11-1) indicate a relatively poor relationship between close-spaced drillholes, particularly where grades in one of the holes is elevated, as seen in D0597A. This is likely predominantly due to the inherent nugget effect and short-scale variability of gold mineralisation, however bias in the contrasting sampling procedures between drilling campaigns, such as between 2002 versus 2014, and between RC and DD holes cannot be precluded on the basis of these limited data. SRK does note, however, that although Au grades are not always continuous over short distances, the available paired drillholes do broadly delineate the same package of mineralised rock and therefore support the interpreted reasonable continuity of the mineralised structure(s).

A visual comparison of Au grades within the three examples of close-spaced drillholes detailed in Table 11-1 is presented in Figure 11-1. SRK recommends twinned drillholes are completed at several, representative locations across the deposit, including a comparison of RC and DD types.

Table 11-1: Summary statistical comparison of close-spaced drillholes at Lafigué

Hole ID	Year	Separation (m)	Weighted Average Grade		Main Mineralised Interval Thickness	
			Au (g/t)	Delta	m	Delta
LF14-039	2014	3.5	2.5	-39%	5.7	58%
LFRC02-50	2002		1.5		9.0	
D0597B	1997	3.5	3.6	44%	31.2	12%
D0597A	1997		5.2		35.1	
LFDD19-669	2019	2.5	4.31	-24%	4.3	65%
LFRC02-56	2002		3.29		7.0	

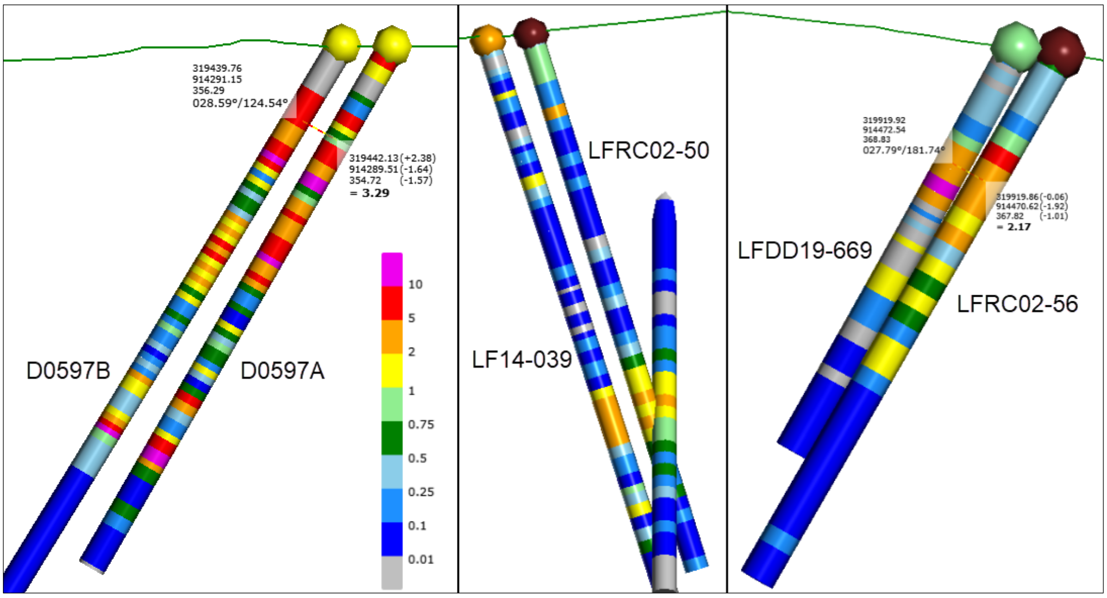


Figure 11-1: Cross-section views showing a visual comparison of Au grades in close-spaced drillholes

11.4 Collar and Survey Verification Surveys

A total of thirteen 2018 and 2019 drillhole collars were resurveyed by Kouamelan during 2020, with no material discrepancies found. Subsequently, all drillholes completed since 2020 have also had verification collar surveys to confirm their positions. Prior to undertaking the MRE, both collar and downhole surveys were checked visually in 3D in order to highlight any clear errors in the survey readings. Additionally, Qualified Person, Dr Lucy Roberts and Dr James Davey of SRK verified the position of five drillholes collar locations (from drilling campaigns ranging from 2014 to 2021) during their visit to site between 14 May and 16 May 2021.

12 MINERAL PROCESSING AND METALLURGICAL TESTING

The following section is largely summarised from the relevant section of the NI 43-101 Technical Report completed for the Lafigué Pre-Feasibility Study in 2021. The reader is referred to the NI 43-101 Technical Report completed for the Lafigué Feasibility Study (in preparation) for a more detailed summary.

12.1 Introduction

Two metallurgical and comminution testwork programmes have been undertaken at the Lafigué Gold Project, comprising a scouting testwork programme in 2018 and a more comprehensive testwork programme in 2019.

Testwork was principally undertaken by ALS Metallurgy ("ALS") in Perth, Western Australia, under the direction of Lycopodium Minerals Pty ("Lycopodium"). SAG mill comminution test analysis was completed by JK Tech Pty ("JKTech") of Queensland, Australia, and thickening testwork was carried out by GBL Process Pty ("GBL") in Perth, Western Australia.

12.2 Sample Selection

Testwork samples were selected to be representative of the mineable resource at Lafigué, inclusive of a range of ore lithologies, weathered states and head grades.

Comminution samples were selected to cover the typical range of lithologies and grades, primarily around the lithological contact zones where gold mineralisation is typically focussed. Samples provided good geographical coverage across the area incorporated into the 2019 Mineral Resource model, which broadly covers a similar footprint to the current model, including samples from Lafigué North, Centre and South zones. Individual host lithologies and adjacent country rock types (considered as likely dilution) were also sampled for comminution testing. No oxide samples were suitable for comminution testing as the material was considered too fine for breakage or work index testing.

In June 2018, a total of 163.5 kg of quartered diamond drill core was delivered to ALS, from which two comminution composites, and ten initial variability composites were produced. Three master composites were generated from variability composites.

In September 2019, a total of 779 kg of half and quarter diamond drillcore was delivered to ALS, from which 12 fresh ore comminution composites, 29 final variability composites (23 fresh and 6 oxide), and three master composites were generated.

12.3 Comminution Testwork

Comminution testwork was undertaken to assess comminution parameters and allow the design of a crushing and milling circuit appropriate for the plant throughput and feed type. Two comminution composites were tested in 2018 and a further 12 composites were tested in 2019. The testwork involved SAG Mill Comminution ("SMC") tests, Bond Abrasion Index determination and Bond Rod Mill Work Index determination. Comminution results are reasonably consistent and indicate that fresh ore is competent with a high breakage energy component, and the abrasion index is low to moderate.

Comminution testwork results were provided to Orway Mineral Consultants ("OMC") for comminution circuit selection and equipment sizing.

12.4 2019 Master Composites

Based on results from the 2018 scouting testwork programme, it was considered that the samples within each group of weathering state could be considered equally representative and combined to form master composites per weathering type. As such, three master composites were generated from variability composites, comprising fresh, high-grade fresh and oxide master composites. The fresh master composite was considered of primary importance given it represented the majority of the ore material (>85% according to the 2019 Mineral Resource estimate).

Multi-element head assays were determined for each of the master composite samples; triplicate gold assays were performed by fire assay analysis. The average gold grade of the fresh, high grade fresh and oxide master composites was 2.05 g/t, 32.8 g/t and 3.06 g/t, respectively. The high-grade fresh composite did not have the expected elevated sulphide and other metal grades typically associated with high-grade gold ores; therefore, this composite became an additional variability sample.

No significant concentrations of deleterious elements for gold leaching were identified, including low levels of base metals, antimony, tellurium, and organic carbon. Mercury and arsenic levels were considered low (Lycopodium, 2021).

12.5 Mineralogical Testwork

Sub-samples of the fresh and high-grade master composites were ground to 80% passing through a 75 µm mesh ("P_{80} 75 µm"), separated into a gravity concentrate and analysed using QEMSCAN (quantitative evaluation of minerals by scanning electron microscopy) and XRD (X-ray diffraction). Gold was identified as both coarse free/liberated grains and as encapsulated grains in sulphides, principally pyrite and pyrrhotite as well as minor silver tellurides. Gold grains in the high-grade composite sample were mainly associated with bismuthotelluride minerals. Optical microscopy indicated that gold grains typically range in size between 2 µm and 500 µm, with the largest identified gold grain approximately 1.5 mm in size (identified in the high-grade composite sample).

12.6 Optimisation Testwork Programme

12.6.1 Grind optimisation testing

The fresh and oxide master composite samples underwent grind size tests to establish optimal grind sizes for gold recovery. Cyanidation tests were conducted on the fresh master composite sample following gravity gold recovery tests at a range of grind sizes (P_{80} 125, 106, 90 and 75 µm).

The fresh composite returned high leach extractions (>97%) across the crush size range tested, with some increase in leach kinetics and gold extraction as grind size decreased. For all grind sizes, the bulk of gold dissolution occurred within four to eight hours. The samples were relatively insensitive to grind size with only a 1% difference in gold extraction over the size range tested. Silver head grades were very low (0.6 g/t Ag), with recoveries averaging 75% and not being materially impacted by grind size.

Testwork on the oxide composite was completed at grind sizes of P_{80} 75 and 106 µm only, with this minor contribution to the feed blend being unlikely to determine the selected grind size during operations.

A grind size of P_{80} 106 µm was selected for design and further testing following an economic evaluation of optimum grind size. This grind size provided an optimal balance of similar gold extraction, but lower operating costs compared to the finer grind sizes tested.

12.6.2 Leach optimisation testing

Leach optimisation testwork assessing cyanide concentration, slurry density and air/oxygen tests were conducted on the fresh and oxide master composite samples at the selected grind size of P_{80} 106 µm. Optimisation testwork was focussed on the fresh ore composite as it represents the majority of the mineralised rock to be processed at Lafigué.

Testwork indicated that high gold extractions (98-99%) were achieved with air-only sparging (no high purity oxygen required), across a range of cyanide concentrations (Lycopodium, 2021). A cyanide concentration of 0.025% w/v NaCN was selected as the optimal addition rate.

Slurry density tests showed that overall gold recoveries were similar at all tested densities, indicating the slurry density had little impact on leaching (lycopodium, 2021).

Conflicting slurry rheology results for the oxide ores were measured, indicating that blending within oxide ores or with fresh ores may be beneficial in managing the materials handling characteristics. In contrast, fresh ores consistently have low viscosities and good settling rates.

12.7 Variability Testwork Programme (2019)

12.7.1 Head assays

Confirmatory testwork using the optimised leach conditions established for the master composites was conducted on the 23 fresh and six oxide variability composites selected to represent various ore domains, weathered states, grade ranges and mineralisation styles. Most variability composites were used to formulate the master composites; however, some samples were retained for variability testwork only.

A small proportion of samples showed relatively consistent triplicate gold assays, indicating a component of disseminated, fine gold, however most of the samples showed significant variability in the triplicate gold assay, indicating the presence of coarse (nuggety) gold. Deleterious element concentrations were generally low, with very low base metal content.

12.7.2 Leach testwork

Using the selected conditions from the Optimisation Testwork Programme (Section 12.6), gold extraction from variability composites was reasonably consistent and broadly aligned with master composite results. Cyanide and lime consumption were similar to the master composite consumption rates with the exception of the oxide ores, where lime consumption was considerably higher, indicating the presence of clays in the material.

Average fresh ore residue grades were slightly higher than the bulk leach result and were significantly lower for the oxide samples. Some samples displayed slower leach kinetics with leaching continuing through to the end of the test (24 hours), although final tails grades were generally acceptable (Lycopodium, 2021).

The oxide material was generally considered as free milling, with relatively high gold recoveries.

Additional testwork to investigate slower leach kinematics in some variability composites identified that increased cyanide addition (maintaining a higher free cyanide excess concentration since consumption remained low) and an increased leach time (36 hours) were sufficient to enhance the kinetics and overall leach gold extraction.

13 MINERAL RESOURCE ESTIMATE

13.1 Introduction

SRK has collated the available exploration information from the Lafigué deposit and has prepared an MRE in accordance with the CIM Definition Standards. Table 13-1 summarises the available drilling data. The MRE and accompanying Statement is the responsibility of the Qualified Person, Dr Lucy Roberts (MAusIMM CP).

Table 13-1: Summary of exploration drilling data for the Lafigué deposit

Period	Type	Number	Total length (m)
1997	DD	14	1,447
	RC	37	1,549
2002	RC	32	1,281
	RAB*	94	1,803
	DD	11	461
2010	RC	11	1,109
	DD	4	396
2014	RC	54	4,638
	DD	23	1,864
2017	RC	179	12,464
	DD	17	2,197
2018	RC	105	14,647
	DD	21	3,861
	RC-DD	8	2,662
	TRCH*	1	19
2019	RC	228	37,633
	DD	15	2,543
	RC-DD	27	7,804
	TRCH*	1	17
2020	RC	164	35,207
	RC-DD	126	41,144
2021	RC	412	61,762
	DD	1	207
	RC-DD	61	19,844
2022	RC	222	25,000
	RC-DD	7	1,310

*Visually considered during modelling but not included in the Mineral Resource estimate
**Drillholes completed for geotechnical or hydrology purposes are excluded from the above totals
DD = Diamond core drilling; RC = Reverse Circulation drilling; RC-DD = Reverse circulation with a diamond core tail; TRCH = Trench
MRE database cut-off date: 15 May 2022

This section describes the methodology used to estimate the Mineral Resources and summarises the key assumptions considered by SRK. SRK considers that the Mineral Resource estimate reported herein is a sound representation of the grade and tonnage of the deposit at the current level of sampling.

Leapfrog Geo version 2021.2 was used to review and model the Mineral Resource estimation domains, prepare assay data for geostatistical analysis, construct the block model, estimate metal grades, and tabulate Mineral Resources.

SRK carried out the following steps to produce the MRE:

- database compilation and review;
- construction of wireframe geological models in Leapfrog Geo 2021.2 software;
- statistical analysis and definition of domains;
- geostatistical analysis (variography) within estimation domains;
- block modelling and grade interpolation using Leapfrog Edge software;
- model validation;
- Mineral Resource classification;
- consideration of reasonable prospects for eventual economic extraction (RPEEE); and
- reporting of Mineral Resources.

13.2 Data Adjustments

The database was directly exported from the Microsoft Access database managed by Company geologists and Endeavour database managers. The following drillhole data was included:

- Collars, including collar co-ordinates, drilling type, hole lengths;
- downhole surveys;
- sample assay intervals;
- lithology logging;
- density;
- mineralisation intervals;
- alteration logging;
- logged structures;
- weathering logging; and
- oxidation logging.

Minor adjustments to the database provided were discussed and rectified prior to continuing with the MRE as part of the data review process; changes included:

- Exclusion of drillholes completed for hydrology or geotechnical purposes, where these drillholes were not assayed; and
- where necessary, missing Au values were set to half of the limit of detection ("LOD"), 0.005 g/t.

SRK notes that samples from rotary air blast ("RAB") holes and exploration trenches were not used in the grade estimate but were considered during the generation of mineralisation wireframes.

13.3 Geology and Mineralisation Models

13.3.1 Lithological domains

In order to produce a simplified lithological model, SRK consolidated the logged lithology codes into a refined lithology field. Simplified lithological domains based on four refined lithology codes (intrusive felsic, extrusive felsic, intrusive mafic and extrusive mafic) were produced as intrusions in Leapfrog Geo, along with an overlying laterite domain (Figure 13-1). SRK notes that some discrepancies between the logging of intrusive and extrusive forms of each rock type has resulted in some localised inconsistencies in the lithological wireframes, and recommends these intervals are relogged and refined, for use in future iterations of the lithology modelling.

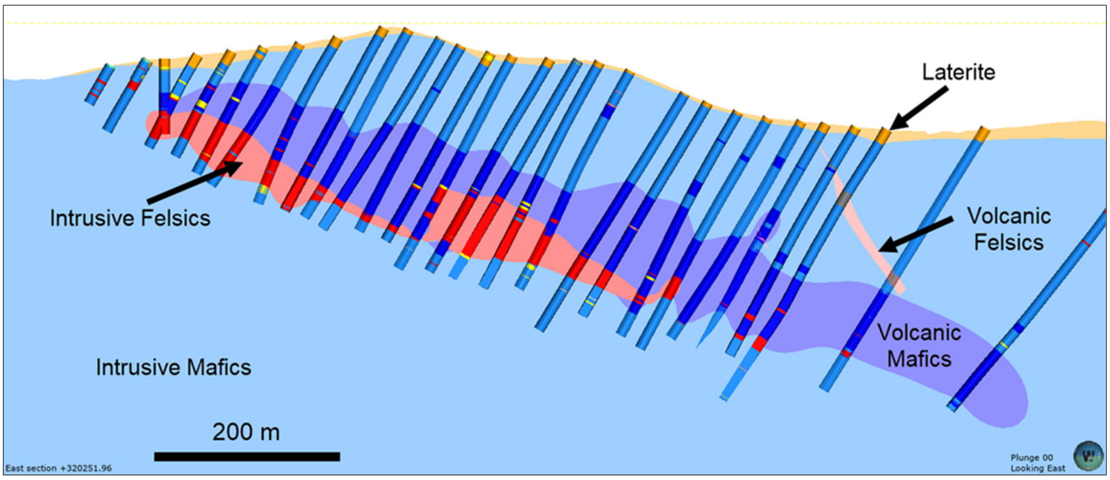


Figure 13-1: Cross-section (looking east) showing the simplified lithological model with the associated lithology logging

13.3.2 Weathering domains

Weathering surfaces were modelled on the basis of weathering logging, where the weathering profile reaches an average depth of approximately 15 to 25 m to fresh rock. Surfaces were produced for the base of the overburden/laterite, saprolite and saprock domains, with all material below the saprock footwall modelled as "fresh" material (Figure 13-2).

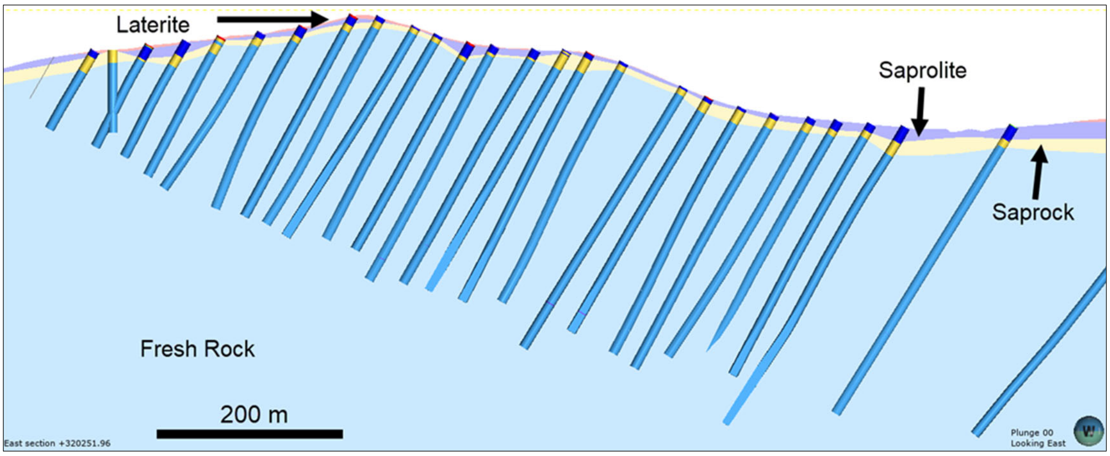


Figure 13-2: Cross-section (looking east) showing the four modelled weathering domains with the associated logging

13.3.3 Mineralisation domains

In the absence of a clear indication of an appropriate modelling cut-off from the Au grade distribution (Figure 13-3), SRK selected a modelling cut-off by assessing the extent and continuity of a series of indicator interpolant shells at different cut-off grades with respect to the assay grades of visually continuous mineralised structures.

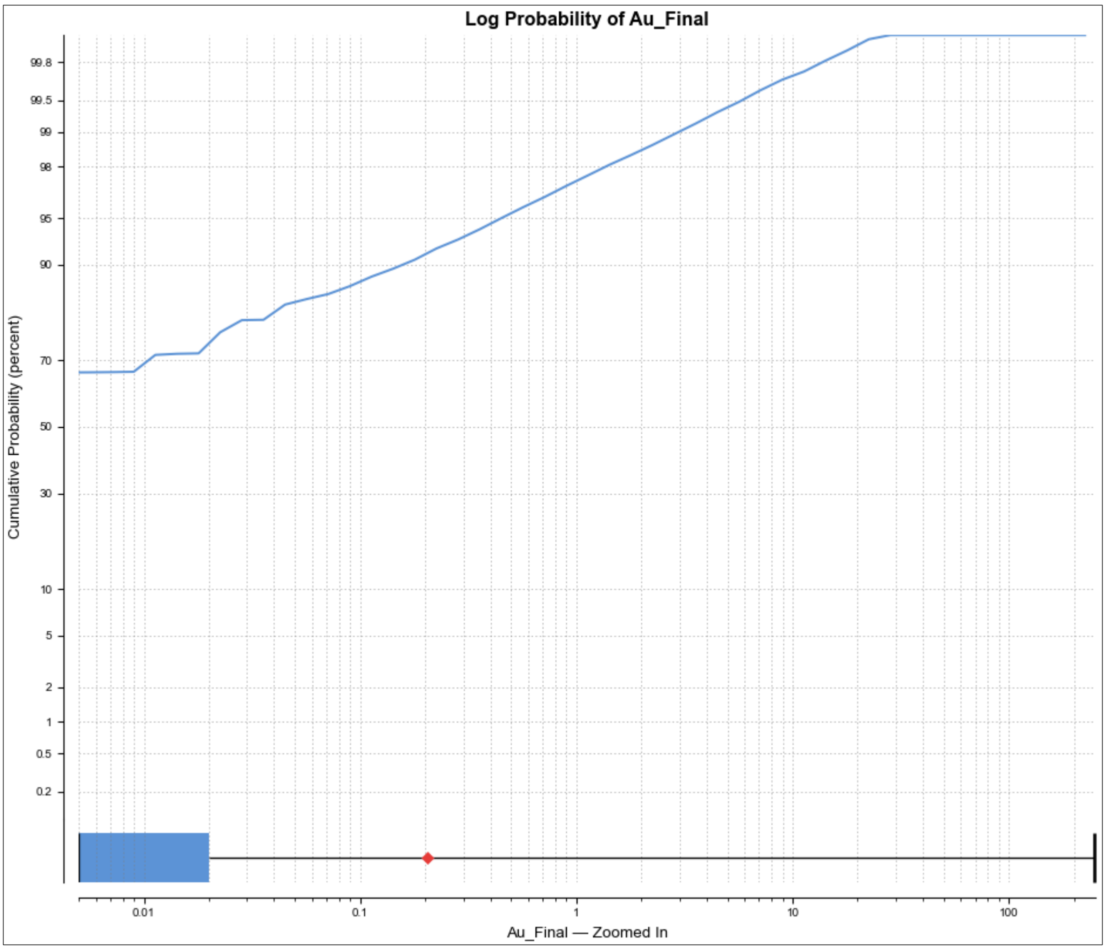


Figure 13-3: Log-probability plot of raw gold assays (filtered to >0.005 g/t Au)

SRK selected a nominal modelling cut-off grade of 0.30 g/t Au for the modelling of Au mineralisation, using an indicator interpolant with a probability (called 'ISO value' in Leapfrog software) of 0.4. Given the clear control of the lithological/rheological contacts on mineralisation, a series of surfaces were produced from the primary lithological contacts, such as the footwall of the intrusive felsic unit. These surfaces were used to produce a structural trend (Figure 13-4), where the trend and orientation of these surfaces influenced the trend and degree of continuity of the indicator interpolant volumes in each direction.

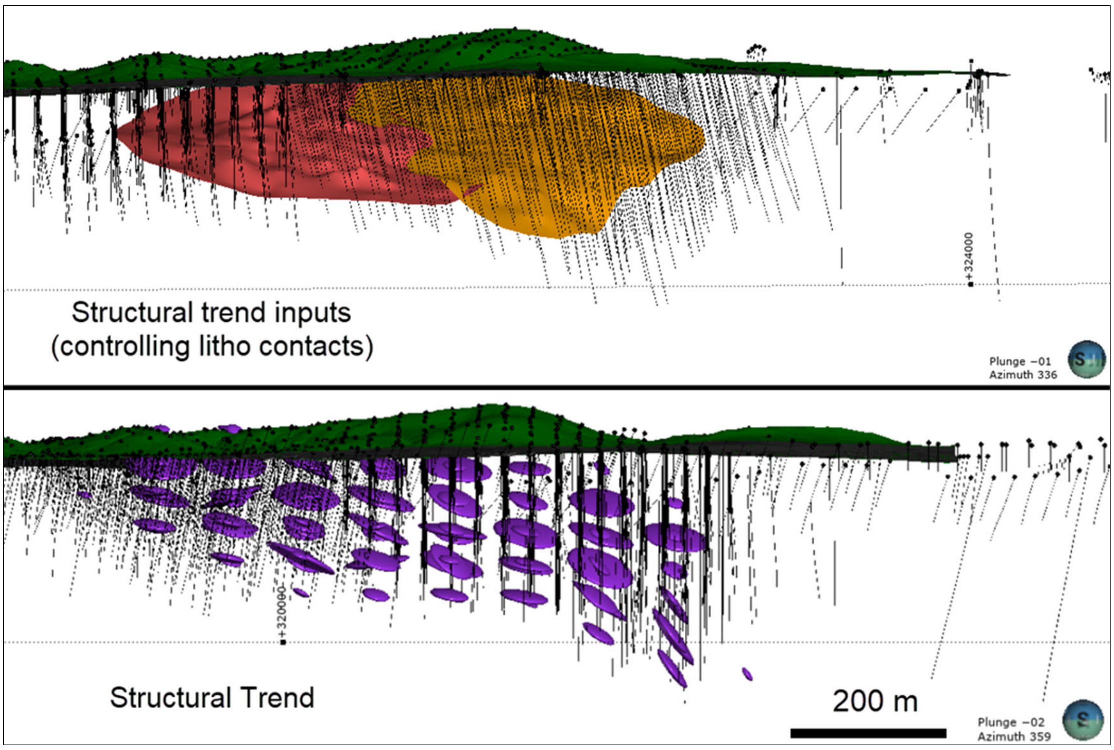


Figure 13-4: Isometric views of the approximate surfaces interpreted to be controlling mineralisation (IFEL footwall surface = orange; IMAF – VMAF contact = red) and the resultant structural trend, represented by purple disks (lower image)

A single indicator interpolant volume was produced, including multiple manual adjustments using indicator polylines. Where mineralised structures were relatively thin, additional wireframes were produced based on sample selections in order to more accurately reflect the geometry and continuity of these structures (Figure 13-5). Vein wireframes based on sample selections were mainly utilised in Lafigué Centre, where mineralisation width and continuity is typically reduced. The final mineralisation domains used for grade and tonnage estimations are shown in Figure 13-6. The domain naming nomenclature is as follows:

- MMZ = Main Mineralisation Zone
- WMZ1 = West Mineralisation Zone 1
- V1-V32 = Vein domains
- LAT = Laterite

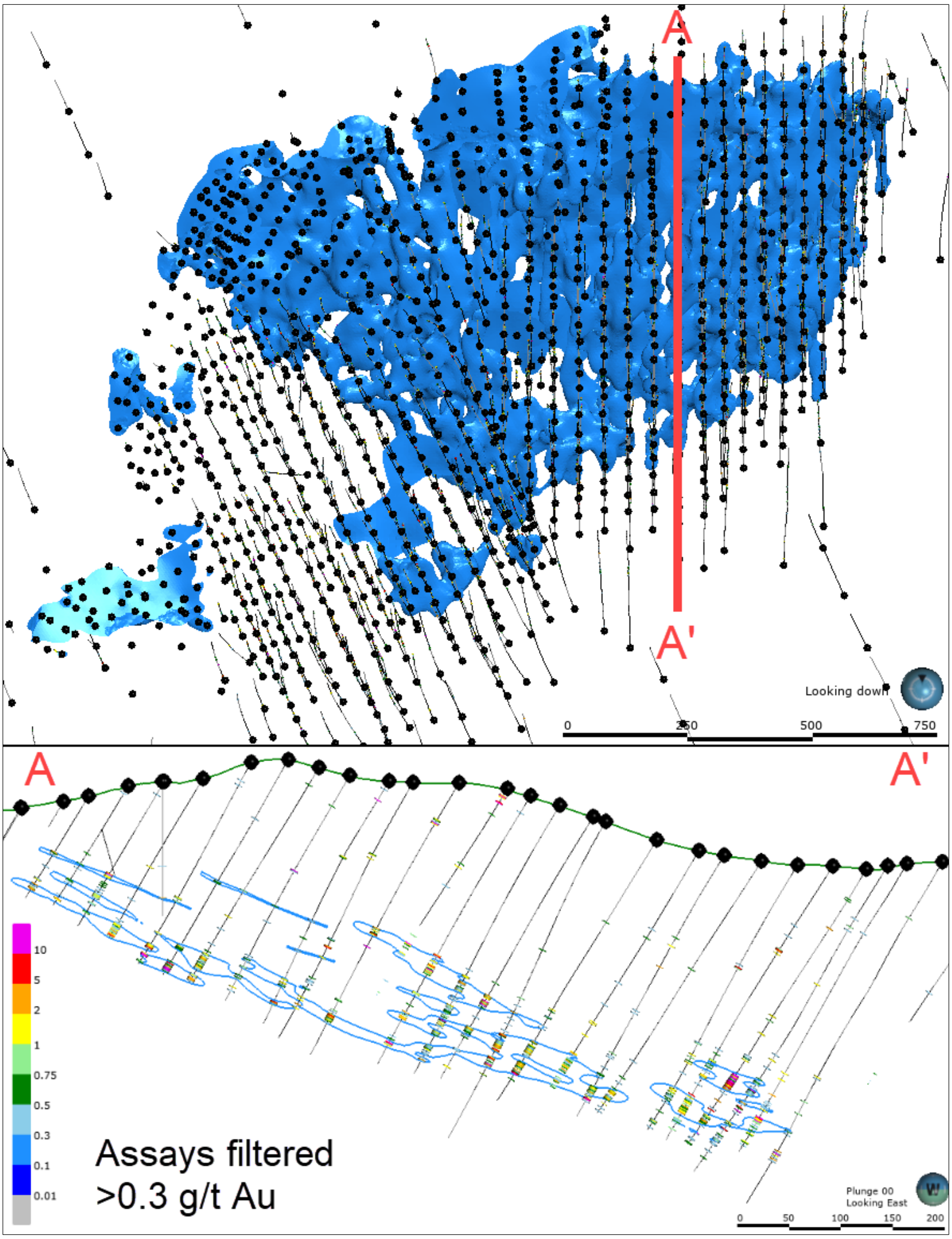


Figure 13-5: Upper image: plan view showing the extents of the mineralisation domains produced using an indicator interpolant. Lower image: A-A' cross-section (looking east) showing the mineralisation domains modelled using a 0.30 g/t Au threshold, including those domains modelled as vein wireframes based on interval selections

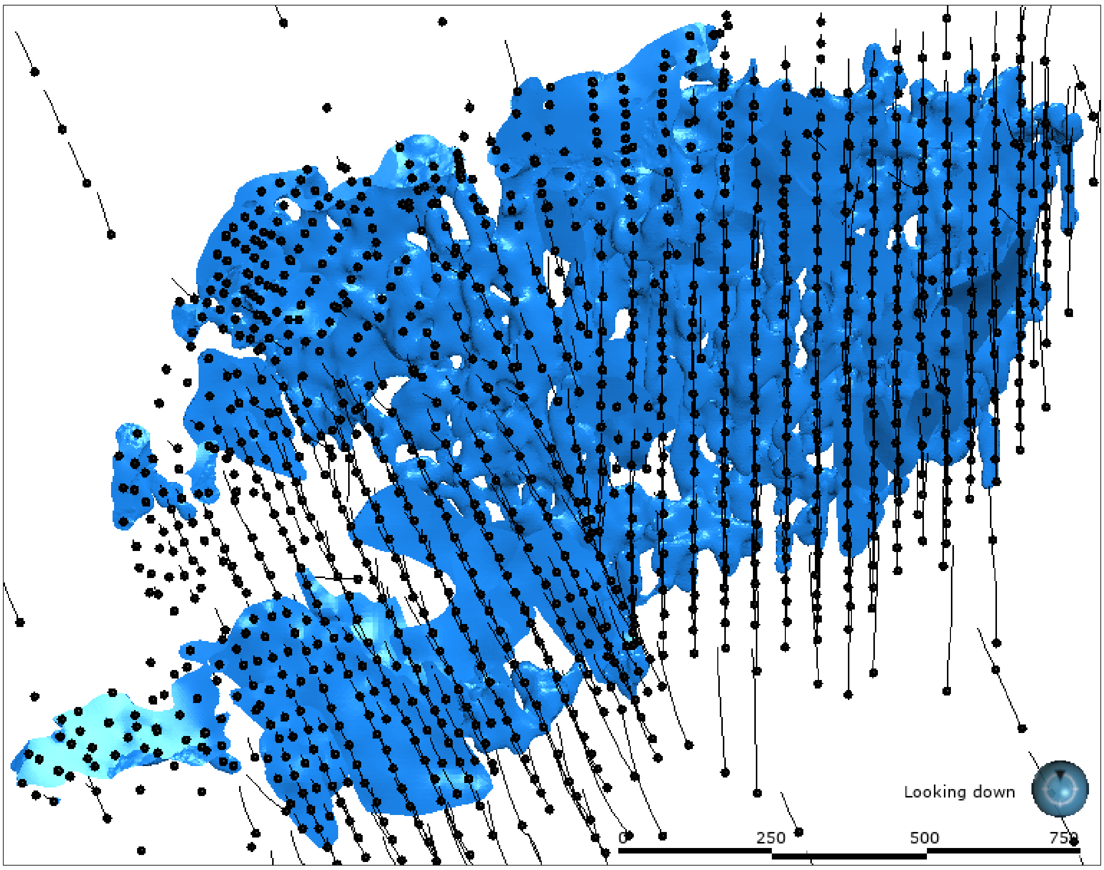


Figure 13-6: Plan view showing all mineralisation domains used for grade and tonnage estimates

13.4 Post-domaining Statistical Analysis

A classical statistical study was undertaken on the domained gold assay data to assess its suitability for grade estimation. The statistics are used to confirm that appropriate estimation domains have been modelled and the statistics remain as constant (as possible) throughout the domain to allow for stationarity (constant grade distribution) to be assumed.

The average Au grades within the modelled mineralisation domains demonstrates a distinction of grade populations between the felsic and mafic host lithologies, with mafic units hosting mineralisation with a higher average grade (Figure 13-7 and Table 13-2).

Table 13-2: Summary statistics for raw assay grades within modelled mineralisation domain, split by host lithology

Domain	No. Samples	Min	Max	Mean	Median	Standard Deviation	CoV
LAT	1,126	0.01	156.10	2.37	0.69	6.78	2.86
IFEL	6,011	0.01	249.60	1.64	0.57	6.10	3.71
VFEL	212	0.01	101.80	1.62	0.53	7.79	4.80
VMAF	5,651	0.01	163.60	2.77	0.58	9.75	3.51
IMAF	7,292	0.01	186.50	2.77	0.64	8.79	3.17
All Felsic	6,223	0.01	249.60	1.64	0.56	6.15	3.74
All Mafic	12,997	0.01	186.50	2.78	0.62	9.21	3.31

LAT = Laterite; IFEL = Intrusive Felsic; VFEL = Volcanic (extrusive) Felsic; VMAF = Volcanic (extrusive) Mafic; IMAF = Intrusive Mafic

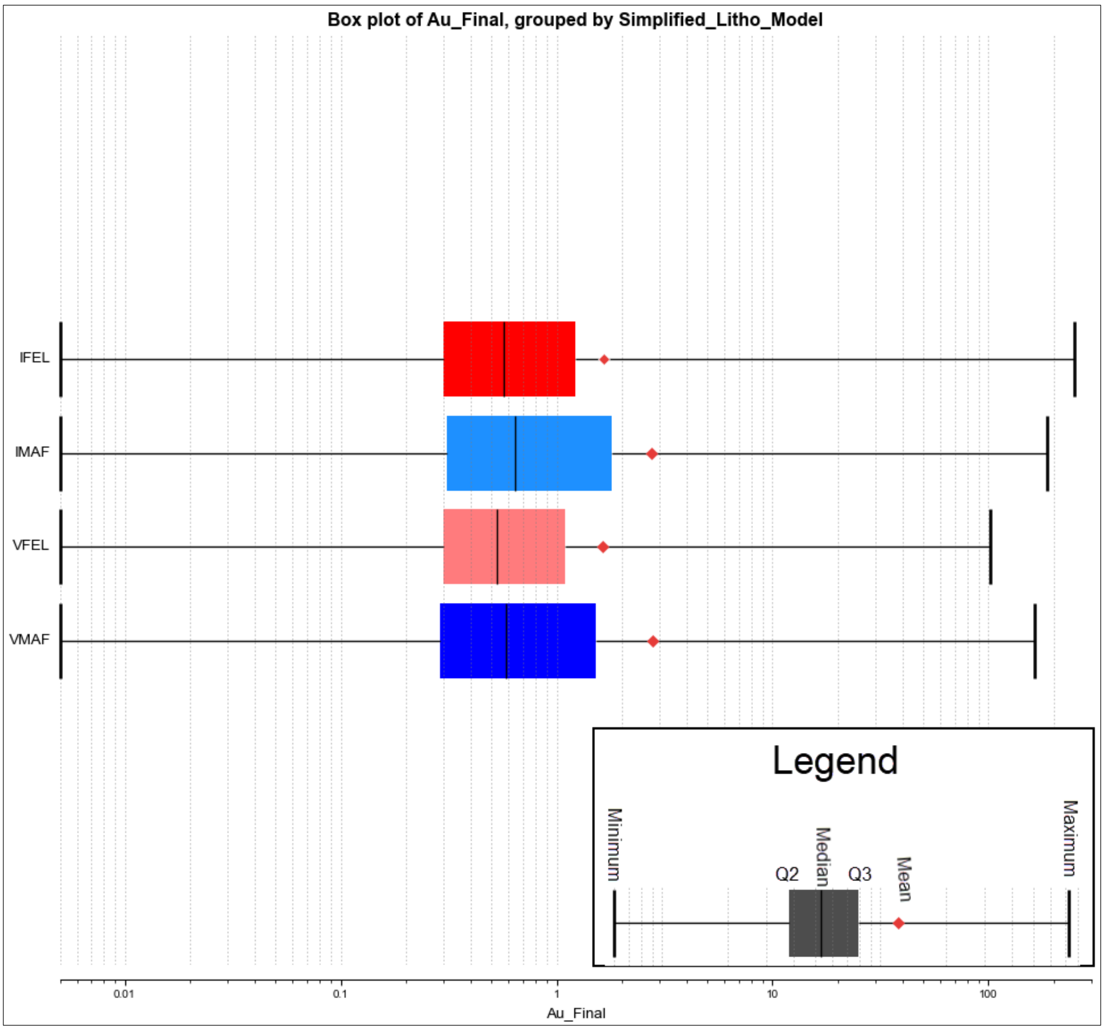


Figure 13-7: Box and whisker plot showing grade ranges (whiskers), upper and lower quartiles (box extents), mean grade (red point) and median grade (vertical line) of raw assays within mineralisation domains, split by host lithology

Given the localisation of mineralisation along the lithological contacts, such as along the IFEL footwall (Figure 13-8) and at intrusive/volcanic mafic contacts (Figure 13-9), rather than the concentration of distinct mineralised structures and grade populations within each of the lithologies, SRK did not split the mineralisation/estimation domains on the basis of host lithology for grade estimation purposes.

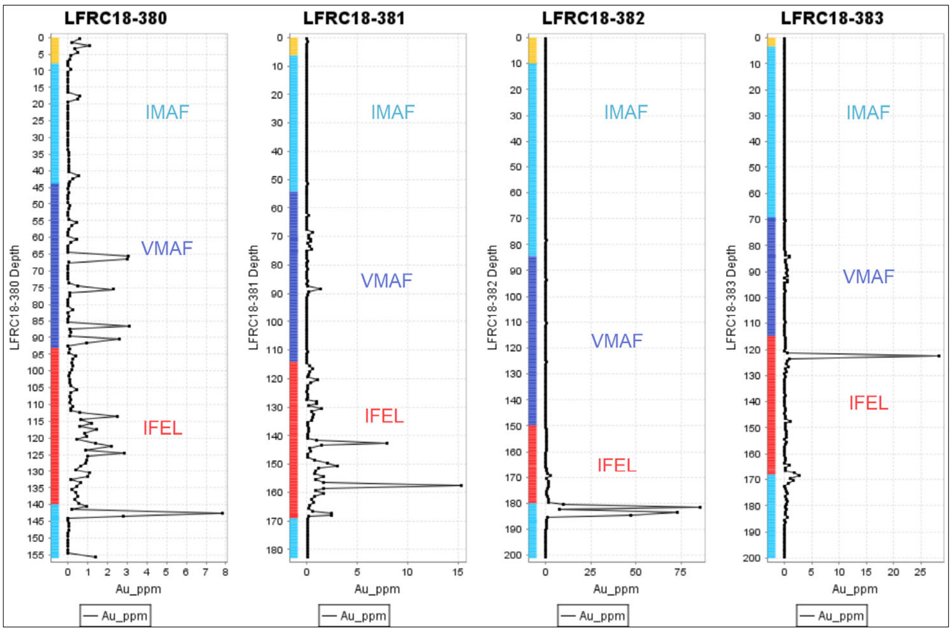


Figure 13-8: Downhole logs showing logged lithologies (coloured bars – left) and Au grades (black trace) in a series of drillholes in Lafigué Centre

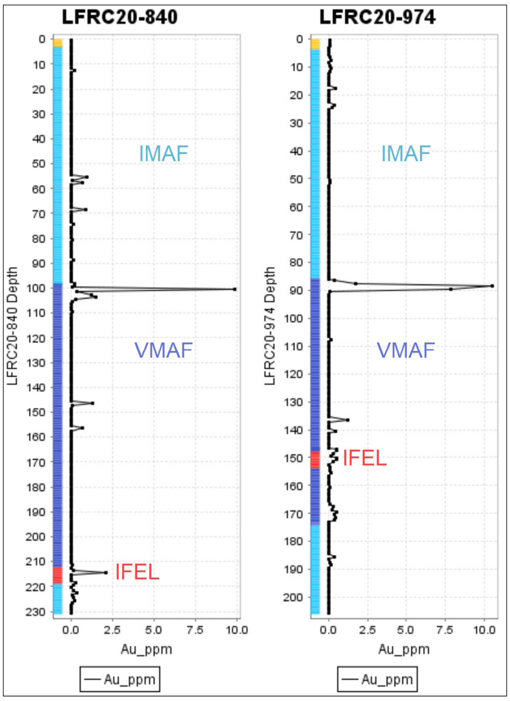


Figure 13-9: Downhole logs showing logged lithologies (coloured bars – left) and Au grades (black trace) in two drillholes in Lafigué Centre/South

Summary statistics for raw assay grades within each of the final estimation domains are presented in Table 13-3.

Table 13-3: Summary statistics for raw assays, split by estimation domain

Domain	No. Samples	Min (g/t Au)	Max (g/t Au)	Mean (g/t Au)	Median (g/t Au)	Standard Deviation	CoV
LAT	311	0.01	55	2.74	1.08	5.00	1.82
MMZ	17,961	0.01	250	2.28	0.59	7.89	3.45
V1	52	0.01	6	0.98	0.59	1.22	1.24
V2	30	0.01	102	4.76	0.45	19.33	4.06
V3	102	0.01	9	0.86	0.54	1.24	1.44
V7	44	0.01	9	1.06	0.39	1.90	1.80
V8	34	0.01	5	1.32	0.70	1.33	1.01
V9	23	0.01	8	1.47	0.48	2.15	1.47
V13	32	0.01	58	4.76	0.53	12.46	2.62
V16	79	0.01	52	2.05	0.42	7.11	3.48
V17	125	0.01	74	2.30	0.60	9.11	3.95
V20	111	0.01	60	2.62	0.65	8.20	3.13
V21	66	0.01	34	1.87	0.49	4.84	2.58
V22	348	0.01	156	4.63	0.97	14.37	3.10
V23	64	0.01	122	5.58	1.03	18.16	3.25
V25	120	0.01	119	3.59	0.83	12.39	3.45
V26	51	0.01	16	2.42	0.69	4.04	1.67
V27	18	0.01	34	6.87	1.31	10.94	1.59
V28	82	0.01	140	12.93	1.90	24.71	1.91
V29	86	0.01	46	3.16	0.88	6.77	2.14
V30	42	0.01	22	1.70	0.43	4.46	2.62
V31	168	0.01	34	1.72	0.69	3.15	1.83
V32	59	0.01	29	3.40	1.59	5.62	1.65
WMZ1	339	0.01	118	2.73	0.64	10.49	3.85

13.5 Compositing

Data compositing is undertaken to reduce the inherent variability that exists within the population and to generate samples appropriate to the scale of the mining operation envisaged. It is also necessary for the estimation process that all samples are assumed to be of equal weighting and should therefore be of equal length.

Based on the sample interval length distribution (Figure 13-10), where >95% of samples are ≤1 m in length, a composite length of 1.0 m was selected for grade estimation. Using a 1.0 m compositing interval, mean Au (g/t) grades range from to 0.99 g/t to 12.91 g/t across the 24 modelled domains. Composite statistics are summarised by estimation domain in Table 13-4.

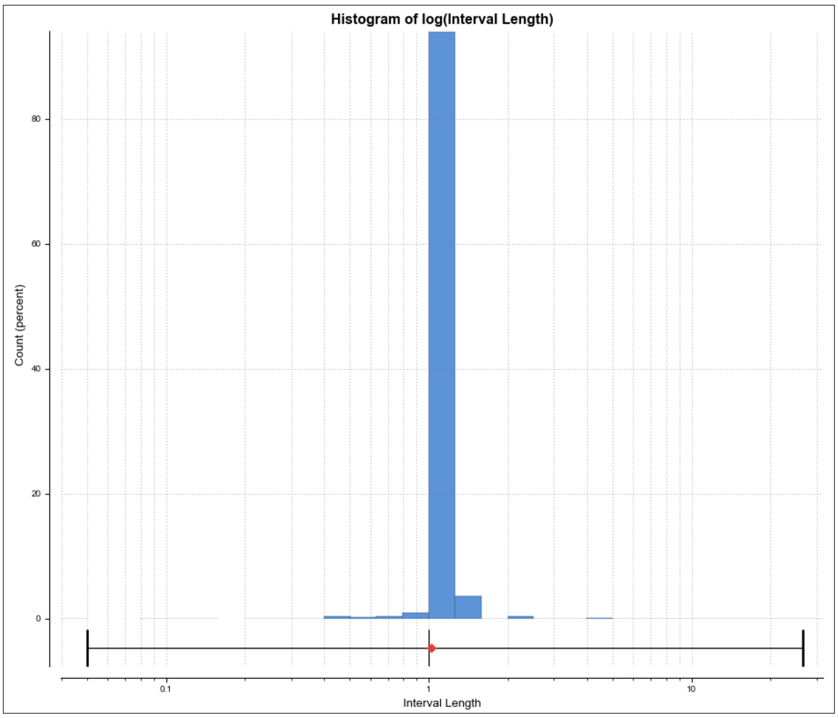


Figure 13-10: Log-histogram of sample interval lengths

Table 13-4: Summary composite statistics split by estimation domain

Domain	No. Samples	Min (g/t Au)	Max (g/t Au)	Mean (g/t Au)	Median (g/t Au)	Standard Deviation	CoV
LAT	255	0.01	55.4	2.72	1.19	4.98	1.83
MMZ	15,745	0.00	249.6	2.27	0.68	6.61	2.91
V1	32	0.01	5.2	0.99	0.57	1.10	1.11
V2	28	0.01	101.8	4.70	0.39	19.21	4.09
V3	64	0.00	8.0	0.84	0.50	1.13	1.35
V7	31	0.08	8.3	1.05	0.47	1.73	1.65
V8	21	0.04	4.5	1.31	0.81	1.22	0.94
V9	16	0.11	7.7	1.46	0.48	2.13	1.46
V13	24	0.05	57.1	4.84	0.60	12.47	2.57
V16	58	0.00	49.4	1.99	0.52	6.78	3.40
V17	119	0.00	74.2	2.31	0.60	9.12	3.95
V20	81	0.05	58.7	2.62	0.68	7.93	3.03
V21	63	0.01	33.8	1.88	0.50	4.84	2.58
V22	274	0.01	151.2	4.61	1.13	13.24	2.87
V23	47	0.01	119.4	5.52	1.21	17.75	3.22
V25	100	0.10	95.7	3.60	0.93	10.17	2.82
V26	37	0.01	16.1	2.54	0.76	3.78	1.49
V27	13	0.03	33.0	8.36	1.54	11.72	1.40
V28	72	0.10	91.2	12.64	3.58	19.90	1.57
V29	68	0.02	41.3	3.20	1.01	6.21	1.94
V30	33	0.12	21.1	1.76	0.43	4.23	2.40
V31	137	0.08	33.7	1.87	0.76	3.48	1.86
V32	46	0.12	20.2	3.45	1.68	4.76	1.38
WMZ1	298	0.00	113.6	2.68	0.78	9.60	3.58

13.6 Gold Grade Capping

The impact of isolated high-grade composites was assessed for each of the estimation domains. Caps or restricted searches can be used to reduce the impact of high grades throughout the entire domain. SRK investigated the presence of high-grade outliers by observing the grade distributions on log-histograms and log-probability plots for Au in each domain. SRK identified high-grade assays that could unduly affect the estimate based on population breaks indicated in both the log-histograms and log probability plots. Example log-probability plots for the largest two domains are shown in Figure 13-11, with plots for all domains presented in Appendix A.

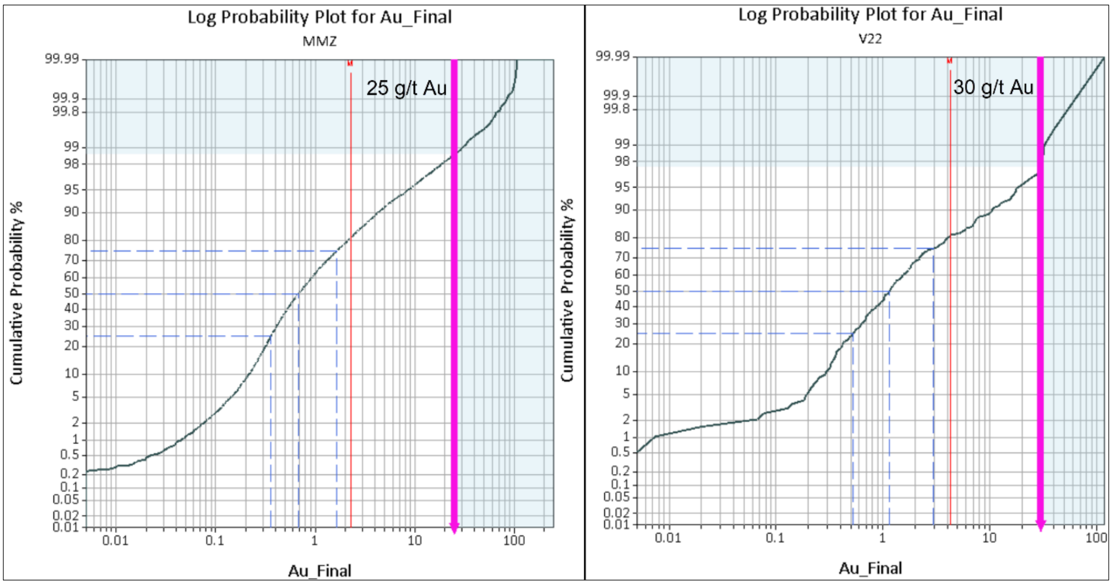


Figure 13-11: Log-probability plot showing selected capping grades (pink lines) for the Main_MinZone and V22 domains

Selected capping grades and the effects of these top cuts on the statistics of composites within each domain are shown in Table 13-5.

Table 13-5: Capping levels and summary statistics for capped composites, split by estimation domain

Domain	Cap (g/t Au)	Uncapped Mean (g/t Au)	Capped Mean (g/t Au)	% Change in Mean	Number of Samples Capped	% of Samples Capped	Standard Deviation	CoV
LAT	20	2.72	2.53	-7%	4	2%	3.61	1.43
MMZ	25	2.27	1.98	-13%	213	1%	3.96	2.00
WMZ1	20	2.68	1.90	-29%	8	3%	3.62	1.91
V1	-	0.99	0.99	0%	-	-	1.10	1.11
V2	15	4.70	1.60	-66%	1	4%	3.70	2.31
V3	-	0.84	0.84	0%	-	-	1.13	1.35
V7	-	1.05	1.05	0%	-	-	1.73	1.65
V8	-	1.31	1.31	0%	-	-	1.22	0.94
V9	-	1.46	1.46	0%	-	-	2.13	1.46
V13	20	4.86	3.00	-38%	2	8%	5.60	1.87
V16	10	2.02	1.21	-40%	2	3%	2.13	1.77
V17	10	2.31	1.23	-47%	3	3%	1.84	1.50
V20	10	2.62	1.56	-41%	3	4%	2.16	1.39
V21	10	1.88	1.36	-28%	3	5%	2.20	1.62
V22	22	4.61	3.29	-29%	16	6%	5.59	1.70
V23	16	5.52	3.06	-45%	4	9%	4.70	1.53
V25	15	3.57	2.70	-24%	2	2%	3.77	1.41
V26	10	2.37	2.18	-8%	2	5%	3.19	1.36
V27	15	6.85	5.90	-14%	3	23%	6.39	1.12
V28	27	12.91	8.59	-34%	13	18%	10.07	1.14
V29	12	3.16	2.54	-20%	6	9%	3.53	1.39
V30	10	1.63	1.39	-15%	2	6%	2.33	1.79
V31	15	1.82	1.69	-7%	1	1%	2.36	1.37
V32	20	3.42	3.25	-7%	1	2%	4.74	1.38

13.7 Boundary Analysis

In order to ascertain whether ‘hard’ or ‘soft’ boundaries between domains should be utilised during grade interpolation, a boundary analysis was undertaken. The process involves a statistical analysis of samples close to each domain (wireframe) boundary.

The Au grades decrease sharply across the boundary between each of the primary mineralisation domains, at spacings much less than the average drill spacing (Figure 13-12), which supports the differentiation of these zones during modelling and the implementation of hard boundary conditions during interpolation of Au grade into the block model.

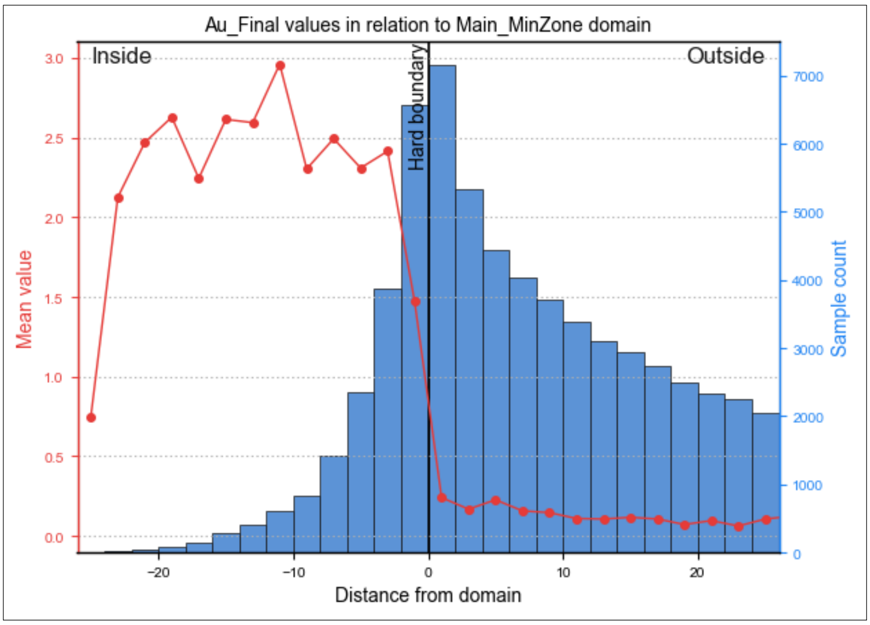


Figure 13-12: Domain boundary analysis for Au (g/t) in the Main Mineralisation Zone

Where a boundary analysis was undertaken for the contact between the laterite and underlying primary mineralisation domains, no statistically significant distinction in average grades was apparent across the contact (Figure 13-13), and so soft boundaries with a range of 3 m were used.

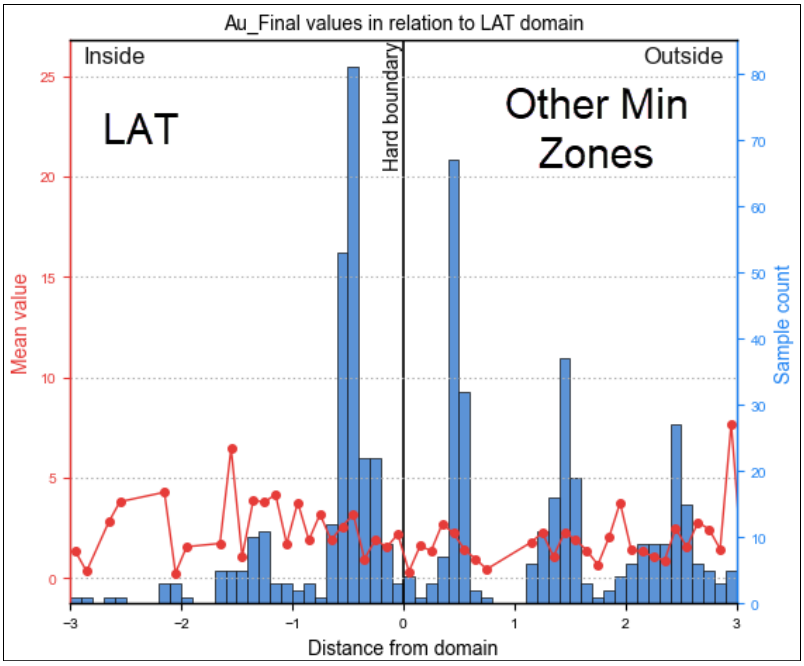


Figure 13-13: Domain boundary analysis for Au (g/t) between the laterite and all other (primary) mineralisation domains

13.8 Geostatistical Analysis

A geostatistical analysis (variography) of the composited Au assay grades was undertaken for each of the main estimation domains. The purpose of the study was to examine the 3D variability and spatial relationships between composite samples, and to derive appropriate variogram models to be used in block grade interpolation. Each domain was analysed separately, first using a variogram map to understand the principal directions of grade anisotropy and choose the major, semi-major and minor directions for analysis. After the directions were chosen, a down-hole variogram was generated to understand the grade variability at short-scales and define the nugget effect. Variograms for three directions were then modelled (using common sill values) to the variance of the data. An example of the directional variograms generated is shown in Figure 13-14.

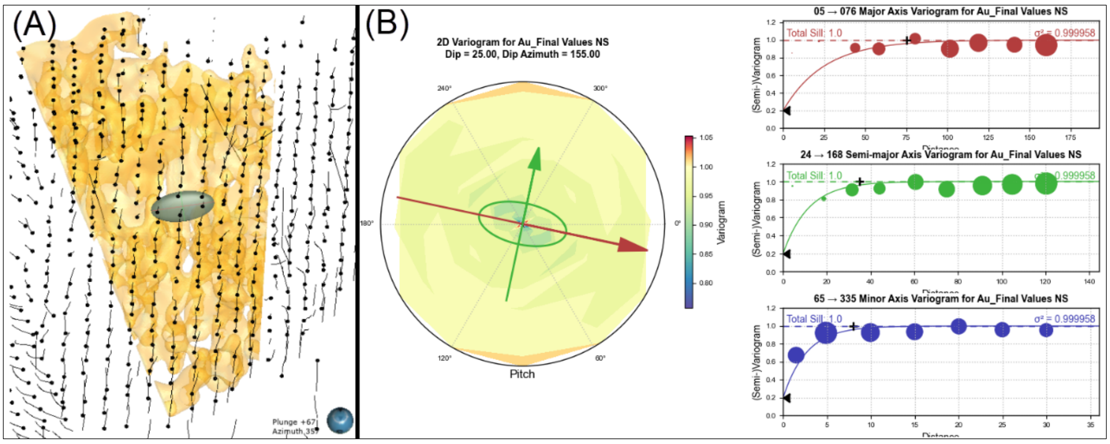


Figure 13-14: (A) Main Mineralisation Zone (Central) domain (orange solid), drillholes (black traces) and modelled variogram ranges displayed as an ellipsoid showing major, semi-major and minor axes directions; (B) Variogram map and normal scores transformed variograms for the major, semi-major and minor axes for Au (g/t) in the same domain

Given the high degree of litho-structural control on mineralisation in the central and eastern parts of Lafigué, the Main Mineralisation Zone was sub-domained on the basis of three dominant structural trends (Figure 13-15). Variography and subsequent estimation was completed for each of the structural sub-domains, with full variogram parameters summarised in Table 13-6. The geostatistical analysis has produced adequate variograms to allow for Ordinary Kriging ("OK") to be utilised for grade interpolation. In smaller, less well-informed domains where there were significantly fewer samples (typically <50 samples), adequate quality variograms could not be produced and an inverse-distance-weighted estimation approach was adopted.

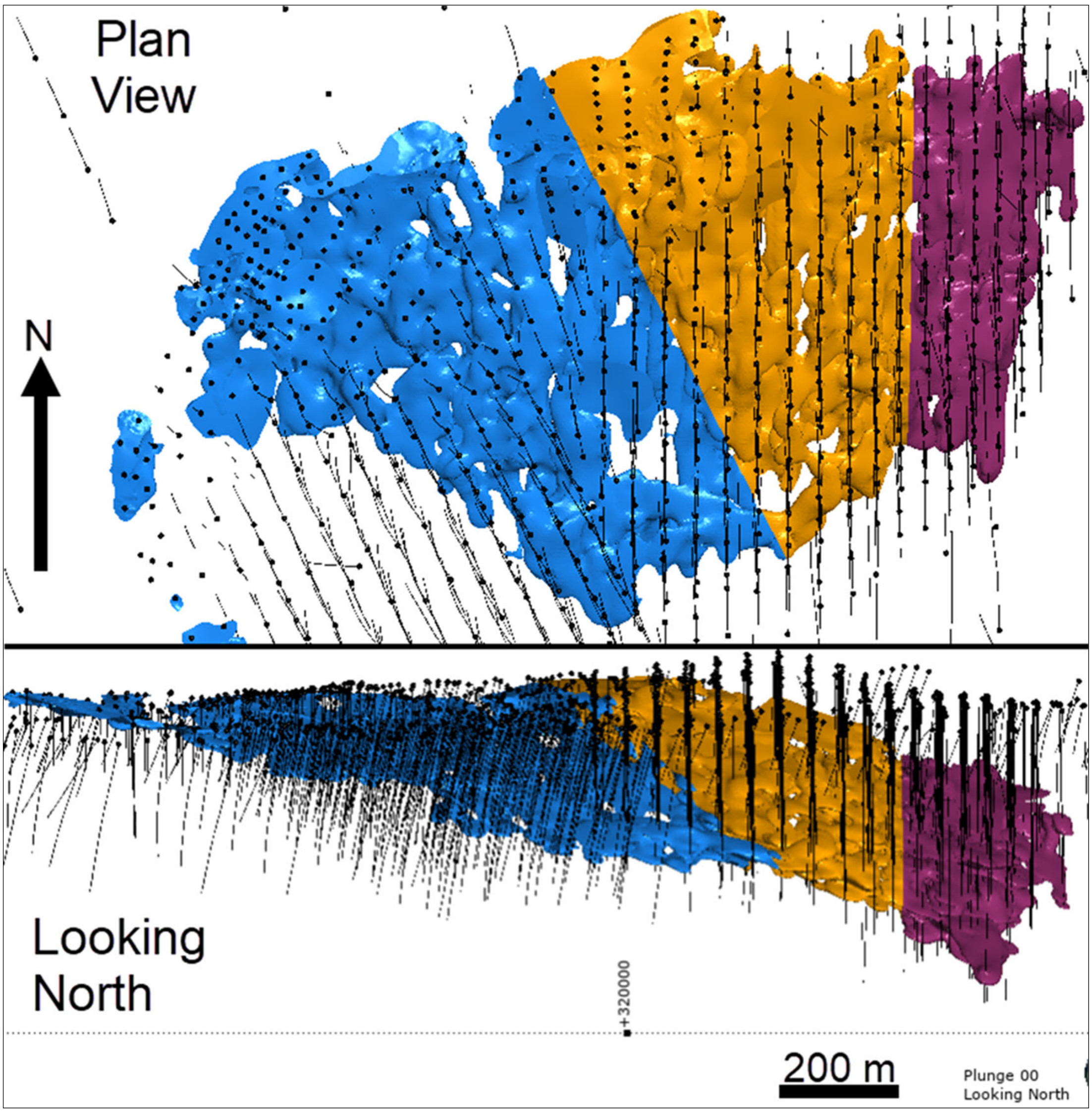


Figure 13-15: Plan view and isometric view (looking north) of the Main Mineralisation Zone structural sub-domains used for variography and grade estimation

Table 13-6: Variogram parameters used for estimation in the Main Mineralisation Zone, V17 and V22 domains

Variogram Name	Direction							Structure 1					Structure 2				
	Dip	Dip Azimuth	Pitch	Model space	Variance	NE	Norm. Nugget	Sill	Norm. sill	Major	Semi-major	Minor	Sill	Norm. sill	Major	Semi-major	Minor
MMZ Central	25	155	12	Data	45.4	20.7	0.46	25.51	0.56	75	35	8					
MMZ Central	25	155	12	Normal score	1.0	0.25		0.75		75	35	8					
MMZ East	30	140	17	Data	67.5	29.7	0.44	29.18	0.43	58	29	2	8.7	0.13	75	40	12
MMZ East	30	140	17	Normal score	1.0	0.25		0.33		58	29	2	0.40		75	40	12
MMZ West	20	150	0	Data	33.0	12.5	0.38	21.11	0.64	45	20	6					
MMZ West	20	150	0	Normal score	1.0	0.20		0.80		45	20	6					
WMZ1	20	143	39	Normal score	1.0	0.25		0.75		60	45	5					
WMZ1	20	143	39	Data	92.1	45.7	0.50	46.44	0.50	60	45	5					
V17	20	150	13	Normal score	1.0	0.40		0.65		55	55	6					
V17	20	150	13	Data	81.3	53.0	0.65	29.13	0.36	55	55	6					
V22	20	145	173	Normal score	1.0	0.30		0.70		55	45	3					
V22	20	145	173	Data	175.3	88.1	0.50	87.13	0.50	55	45	3					
V25	25	140	75	Normal score	1.0	0.30		0.75		70	50	3					
V25	25	140	75	Data	75.0	33.0	0.44	41.98	0.56	70	50	3					
V28	10	160	10	Normal score	1.0	0.30		0.70		50	50	6					
V28	10	160	10	Data	492.9	196.6	0.40	296.43	0.60	50	50	6					
V31	20.41	144	55	Normal score	1.0	0.40		0.65		38	35	2					
V31	20.41	144	55	Data	11.3	6.1	0.54	5.23	0.46	38	35	2					

NE = Nugget Effect

13.9 Block Modelling and Grade Estimation

13.9.1 Block model definition

The block model covered an area encompassing all modelled mineralised zones. The geometry and extents of the block model are summarised in Table 13-7. Parent block dimensions are 20 x 20 x 10 m and are sub-blocked to 2.5 x 2.5 x 1.25 m. No rotation was applied to the block model.

Table 13-7: Lafigué block model dimensions

Dimension	Origin	Block Size (m)	Number of Blocks	Minimum Sub-blocking (m)
X	318950	20	106	2.50
Y	913150	20	83	2.50
Z	-180	10	65	1.25

13.9.2 Grade interpolation

Search ellipsoid parameters were tailored to consider the number of drillholes to be used, based on the average drillhole spacing, with the search orientation aligned with the model variograms obtained for each domain. In the Main Mineralisation Zone, dynamic anisotropy was utilised due to account for the variable orientation of each structural sub-zone (Figure 13-16). The variable orientation was informed by surfaces representing the primary mineralisation-controlling structures/lithology contacts as described in Section 13.3.3. Individual domains were estimated separately using hard boundaries (with the exception of the Laterite domain and Main Mineralisation Zone sub-domains) in order to prevent drillhole data from one domain affecting block grades in a neighbouring domain.

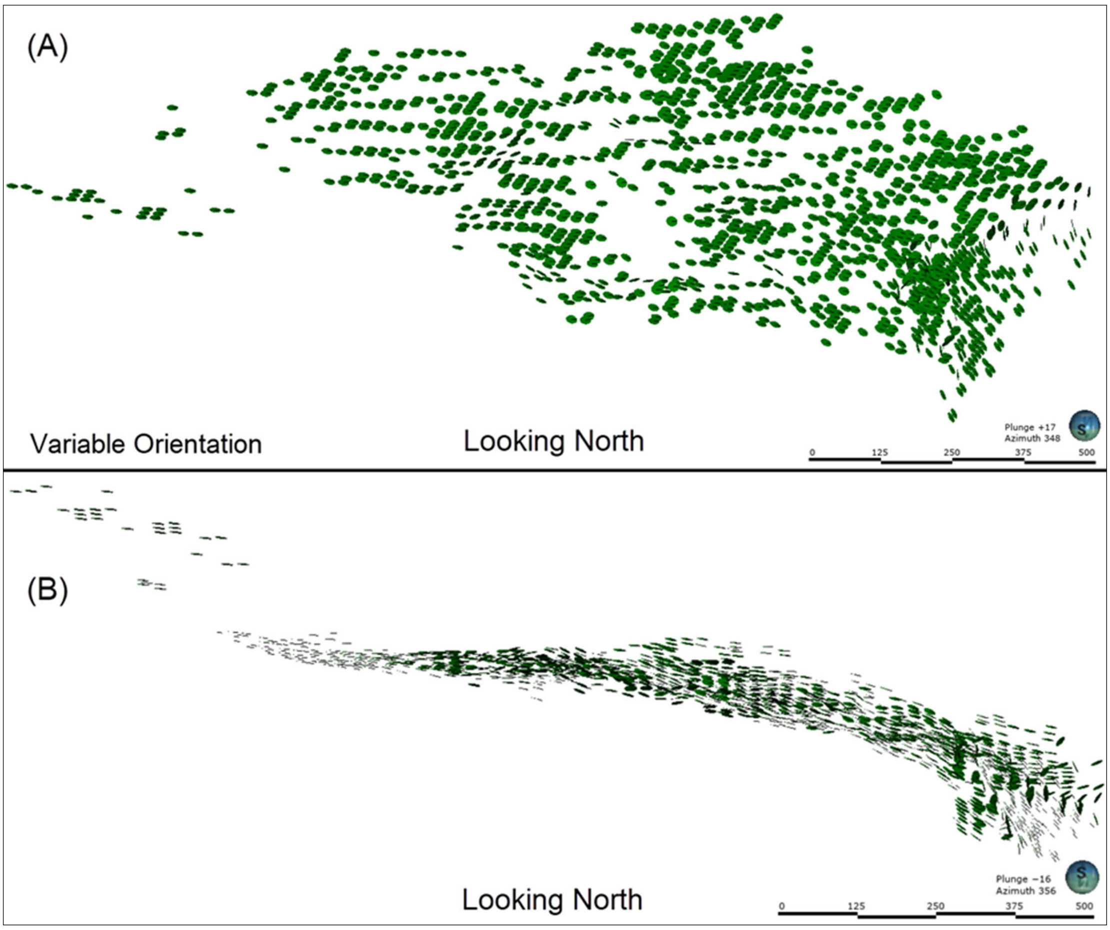


Figure 13-16: Isometric views (looking north) showing the variable orientation (dynamic anisotropy – displayed as green disks) for the Main Mineralisation Zone

13.9.3 Neighbourhood analysis

Kriging neighbourhood analysis ("KNA") was undertaken in order to optimise the block size, sample selection criteria and discretisation used during grade interpolation. The initial KNA process was based on comparisons of kriging efficiency ("KE") and slope of regression ("SoR"), when varying each of the above parameters independently. The kriging efficiency estimates the degree of correspondence between the estimated block histogram and that of the true block grades, where a KE of 100% would represent a perfect match between the two (Coombes, 2008). The slope of regression is a measure of conditional bias. That is, the tendency for higher grades to be under-estimated and lower grades to be over-estimated, where the slope of regression equation compares the estimated and theoretical true block grades (Coombes, 2008). A 1:1 relationship between theoretical true and estimated block grades would produce a slope of 1, meaning that the estimated high grades and estimated low grades correspond accurately to the respective theoretical true high and low grades. The flatter the slope (and therefore over-estimation of low grades and under-estimation of high grades), the lower the slope of regression. Figure 13-17 shows example plots for the Main Mineralisation Zone – Central domain, where KE and SoR are plotted as a function of selected block sizes and min/max samples selected. Overall, the KNA undertaken showed that the estimates were relatively insensitive to changing parent block size or min/max sample selection criteria (within reasonable ranges).

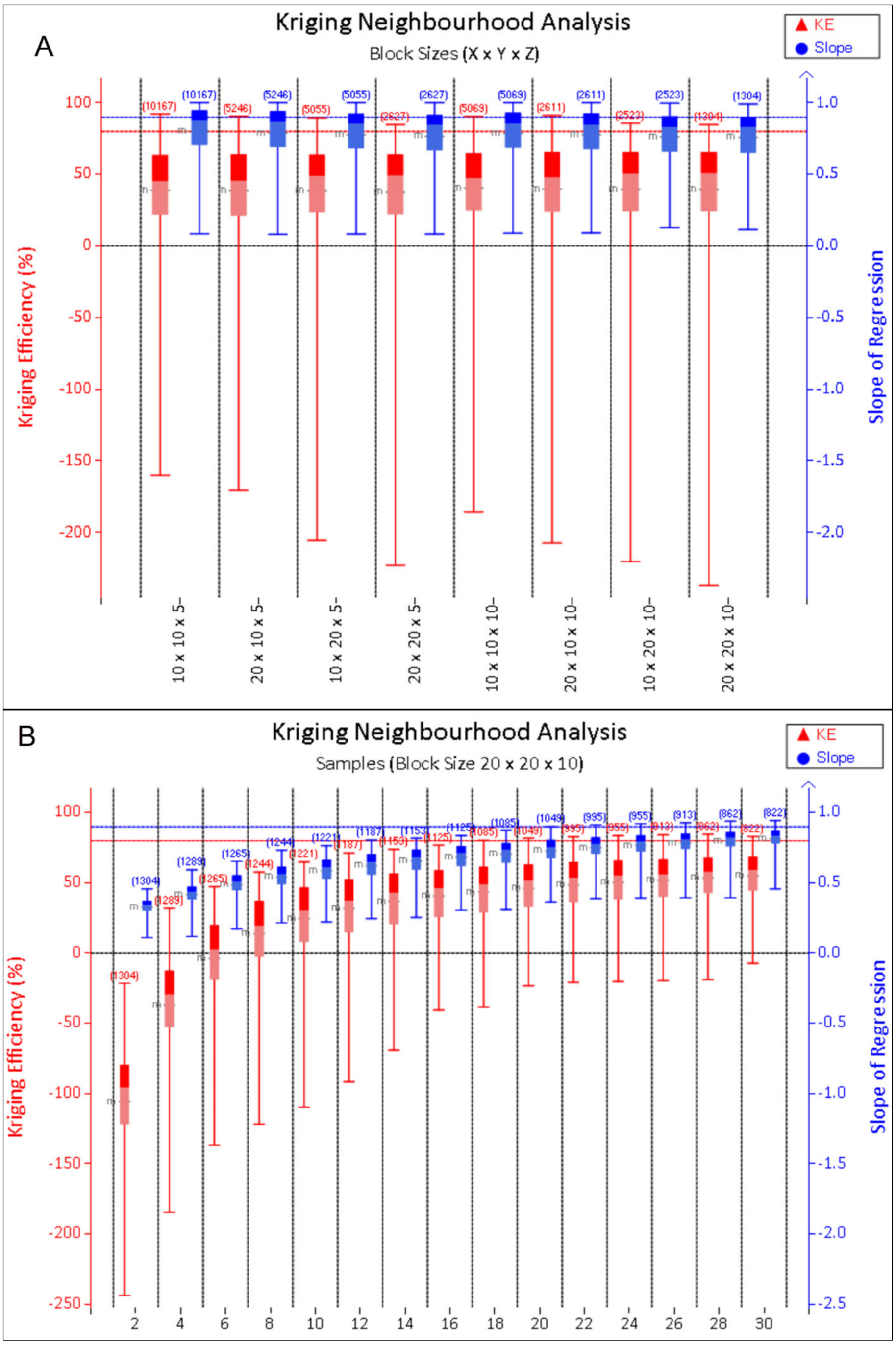


Figure 13-17: Series of box and whisker plots showing kriging efficiency and slope of regression as a function of changing, (A) parent block sizes, at increments of 5 m between 10 x 10 x 5 m and 20 x 20 x 10 m; and (B) number of samples used to inform block estimates between a minimum of 2 and a maximum of 30 samples

Additional sensitivity analyses were undertaken assessing the influence of changes to search ellipsoid dimensions by running a series of estimation runs and comparing a range of search ellipsoid dimensions appropriate to the drill spacing and variogram ranges. Defaults were selected for all other search parameters and remained constant for each sensitivity run, with only the search ellipsoid dimensions adjusted.

A discretisation level of 4 x 4 x 2 was set for all estimates. The final grade interpolation parameters used for each domain are detailed in Table 13-8.

Table 13-8: Summary of Lafigué estimation parameters

Domain	Estimation Method	Ellipsoid Directions			Ellipsoid Ranges			Number of Samples		Drillhole Limit	Sector Search		
		Dip	Dip Az.	Pitch	Maximum	Intermediate	Minimum	Minimum	Maximum	Max Samples per Hole	Method	Max Samples	Max Empty Sectors
MMZ_Central - Pass 1	OK	Variable Orientation			75	35	6	7	12	3	Quadrant	3	1
MMZ_Central - Pass 2	OK	Variable Orientation			100	45	10	7	12	3	Quadrant	3	1
MMZ_Central - Pass 3	OK	Variable Orientation			200	100	25	4	12	3	None	-	-
MMZ_East - Pass 1	OK	32	125	135	60	45	8	6	12	3	Quadrant	2	1
MMZ_East - Pass 2	OK	32	125	135	80	60	10	6	12	3	Quadrant	2	1
MMZ_East - Pass 3	OK	32	125	135	200	100	30	4	12	3	None	-	-
MMZ_West - Pass 1	OK	Variable Orientation			60	35	8	7	12	3	Quadrant	3	1
MMZ_West - Pass 2	OK	Variable Orientation			75	45	12	7	12	3	Quadrant	3	1
MMZ_West - Pass 3	OK	Variable Orientation			200	100	40	4	12	3	None	-	-
LAT - Pass 1	IDW^2	0	50	90	60	40	4	7	12	3	None		
LAT - Pass 2	IDW^2	0	50	90	75	50	6	5	12	3	None		
LAT - Pass 3	IDW^2	0	50	90	150	150	20	3	5	-	None		
V1 - Pass 1	IDW^2	25	165	0	60	60	60	8	20	7	None		
V1 - Pass 2	IDW^2	25	165	0	150	150	100	6	20	5	None		
V2 - Pass 1	IDW^2	25	175	0	60	60	60	7	20	6	None		
V2 - Pass 2	IDW^2	25	175	0	150	150	150	9	20	8	None		
V3 - Pass 1	IDW^2	55	135	150	60	60	60	9	20	8	None		
V3 - Pass 2	IDW^2	55	135	150	130	130	130	8	20	7	None		
V7 - Pass 1	IDW^2	30	190	5	50	50	20	7	14	6	None		
V7 - Pass 2	IDW^2	30	190	5	150	150	50	4	7	3	None		
V8 - Pass 1	IDW^2	25	175	10	45	45	20	6	10	5	None		
V8 - Pass 2	IDW^2	25	175	10	100	100	50	4	7	3	None		
V9 - Pass 1	IDW^2	25	175	20	60	60	60	11	20	8	None		
V9 - Pass 2	IDW^2	25	175	20	150	150	150	4	7	3	None		
V13 - Pass 1	IDW^2	25	180	10	60	60	60	9	14	8	None		
V13 - Pass 2	IDW^2	25	180	10	75	75	50	4	7	3	None		
V16 - Pass 1	IDW^2	20	145	0	40	40	25	9	20	8	None		
V16 - Pass 2	IDW^2	20	145	0	100	100	50	7	12	6	None		
V17 - Pass 1	OK	20	150	13	55	50	6	8	14	-	None		
V17 - Pass 2	OK	15	140	20	80	70	20	8	14	-	Quadrant	5	1
V17 - Pass 3	OK	15	140	20	200	150	30	3	12	-	Quadrant	5	1

Domain	Estimation Method	Ellipsoid Directions			Ellipsoid Ranges			Number of Samples		Drillhole Limit	Sector Search		
		Dip	Dip Az.	Pitch	Maximum	Intermediate	Minimum	Minimum	Maximum	Max Samples per Hole	Method	Max Samples	Max Empty Sectors
V20 - Pass 1	IDW^2	25	100	15	40	40	15	7	12	6	None		
V20 - Pass 2	IDW^2	25	100	15	75	75	50	4	7	3	None		
V21 - Pass 1	IDW^2	25	100	15	40	40	20	9	20	8	None		
V21 - Pass 2	IDW^2	25	100	15	100	100	60	7	14	6	None		
V22 - Pass 1	OK	20	145	170	55	45	6	7	12	-	None		
V22 - Pass 2	OK	20	145	170	80	70	10	7	12	-	Quadrant	4	1
V22 - Pass 3	OK	20	145	170	150	150	25	3	14	-	Quadrant	4	1
V23 - Pass 1	IDW^2	23	140	25	75	75	75	8	20	7	None		
V23 - Pass 2	IDW^2	23	140	25	125	125	125	8	20	7	None		
V25 - Pass 1	OK	25	140	75	45	40	10	6	10	3	None		
V25 - Pass 2	OK	25	140	75	80	70	20	6	10	3	Quadrant	4	1
V25 - Pass 3	OK	25	140	75	150	150	25	3	12	-	Quadrant	4	1
V26 - Pass 1	IDW^2	25	180	10	60	60	60	9	14	8	None		
V26 - Pass 2	IDW^2	25	180	10	120	120	120	9	14	8	None		
V27 - Pass 1	IDW^2	25	180	10	55	55	20	7	12	6	None		
V27 - Pass 2	IDW^2	25	180	10	100	100	50	4	7	3	None		
V28 - Pass 1	OK	10	160	10	60	60	10	7	12	-	None		
V28 - Pass 2	OK	10	160	10	75	75	12	7	12	-	Quadrant	4	2
V28 - Pass 3	OK	10	160	10	150	150	25	3	12	2	Quadrant	4	2
V29 - Pass 1	IDW^2	15	80	165	40	40	20	9	14	8	None		
V29 - Pass 2	IDW^2	15	80	165	100	100	50	7	12	6	None		
V30 - Pass 1	IDW^2	15	140	140	40	40	20	7	12	6	None		
V30 - Pass 2	IDW^2	15	140	140	100	100	50	4	7	3	None		
V31 - Pass 1	OK	20	145	55	40	35	8	7	12	-	None		
V31 - Pass 2	OK	20	145	55	75	70	12	7	12	-	Quadrant	4	1
V31 - Pass 3	OK	20	145	55	100	100	25	3	4	-	Quadrant	4	1
V32 - Pass 1	IDW^2	25	145	20	40	40	20	7	12	6	None		
V32 - Pass 2	IDW^2	25	145	20	100	100	50	4	7	3	None		
WMZ1 - Pass 1	OK	20	140	40	50	40	8	9	12	4	None		
WMZ1 - Pass 2	OK	20	140	40	100	80	12	9	15	4	Quadrant	4	1
WMZ1 - Pass 3	OK	20	140	40	200	150	25	3	10	-	Quadrant	4	1

13.10 Tonnage Estimation

The density database provided includes a total of 2,214 samples with logged lithology and weathering attributes and located within the resource model area. The samples are distributed across the Project area, with a slight spatial bias towards the centre and east of the deposit (Figure 13-18). Each lithology is represented in the density database; however, the laterite material is only represented by a single density measurement from a sample outside of the extents of mineralisation modelled by SRK. Although this represents a risk in terms of the representivity and accuracy of the density value applied to lateritic material, SRK considers the risk to be minimised by the limited remaining tonnage within this domain (see Section 13.12). SRK has coded the block model with average density values, split by lithology / material type. These values are listed in Table 13-9. SRK notes that a number of density samples (e.g., from hydrological drillholes) do not have accompanying lithology and weathering logging. With reduced certainty of what material type these samples represent, SRK excluded these samples from the statistics presented in Table 13-9.

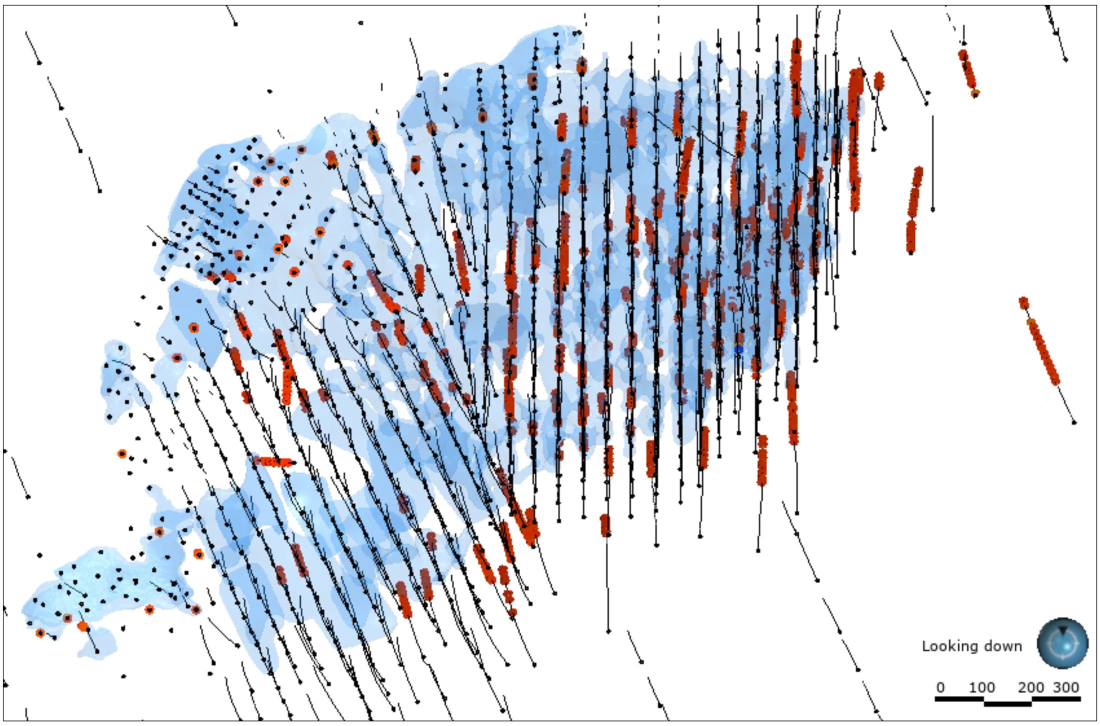


Figure 13-18: Plan view showing spatial distribution of drillholes with down-hole density measurements (red disks) relative to the modelled mineralisation domains (blue)

Table 13-9: Average density values, split by lithology

Lithology	Material Type	Number of Measurements	Mean Value (g/cm^3)	Notes
Laterite	Oxide	1	2.00	Single measurement outside of modelled mineralisation extents
Saprolite	Oxide	9	1.66	Excludes one anomalous measurement (2.6)
Saprock	Transition	17	2.51	Excludes one anomalous measurement (3.1)
Fresh - Mafic	Fresh	1,205	2.86	
Fresh - Felsic	Fresh	628	2.72	

13.11 Model Validation

SRK validated the block model through the following checks:

- local validation using visual inspections on sections and plans, viewing composites versus block estimates;
- global validation by comparison of de-clustered composite statistics versus block estimates; and
- local validation by comparison of average assay grades with average block estimates along different directions, through the generation of swath plots.

SRK considers that the block model reflects the current understanding of the distribution of mineralisation and is an acceptable basis for a Mineral Resource statement.

13.11.1 Visual validation

Visual validation provides a comparison of the interpolated block model on a local scale. A thorough visual inspection has been undertaken in 3D, demonstrating a good degree of correspondence between the block estimates and nearby composites (Figure 13-19 and Figure 13-20).

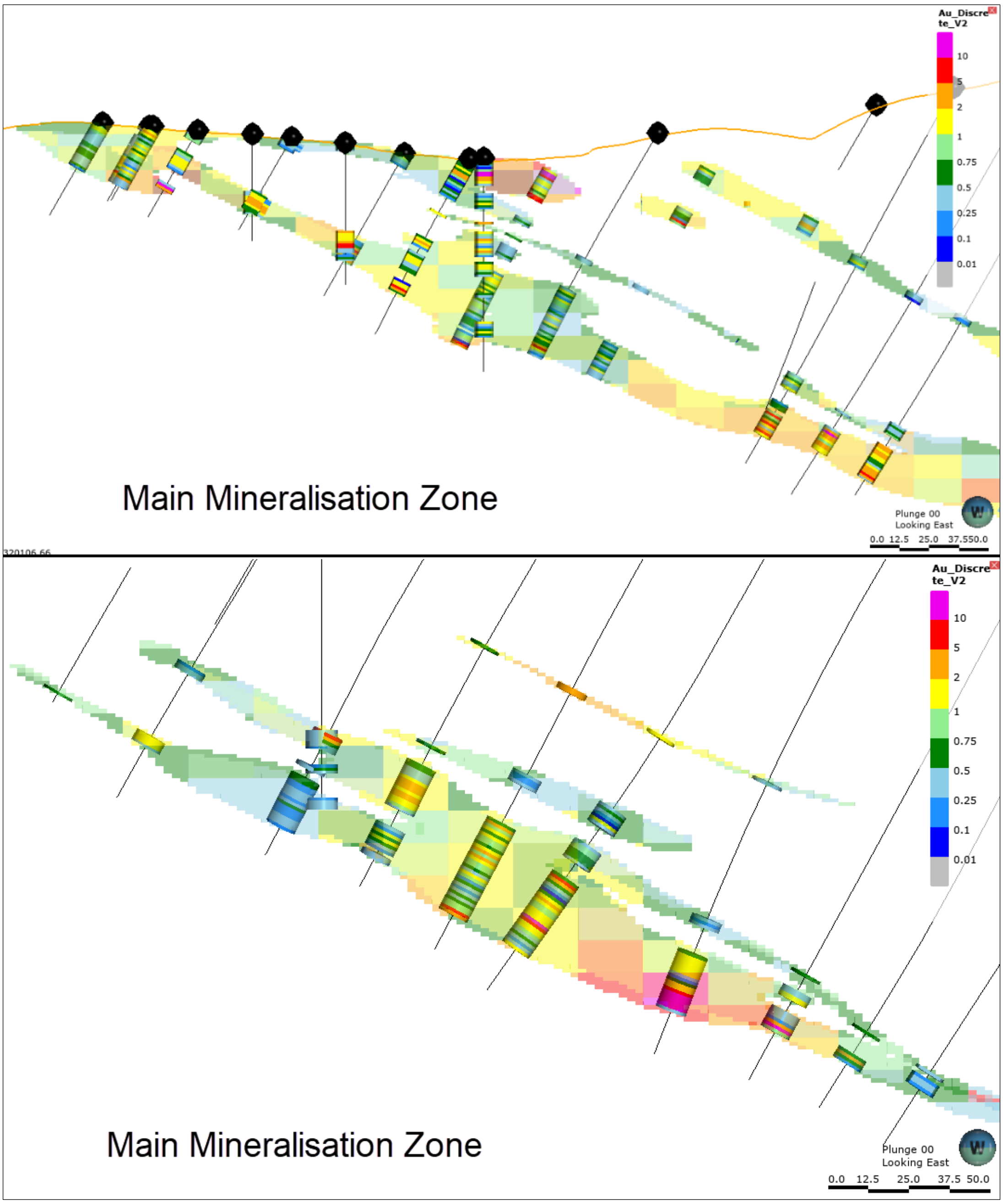


Figure 13-19: Cross-sections (looking east) through the Main Mineralisation Zone showing estimated block grades versus input composite grades, each coloured by Au grade (g/t)

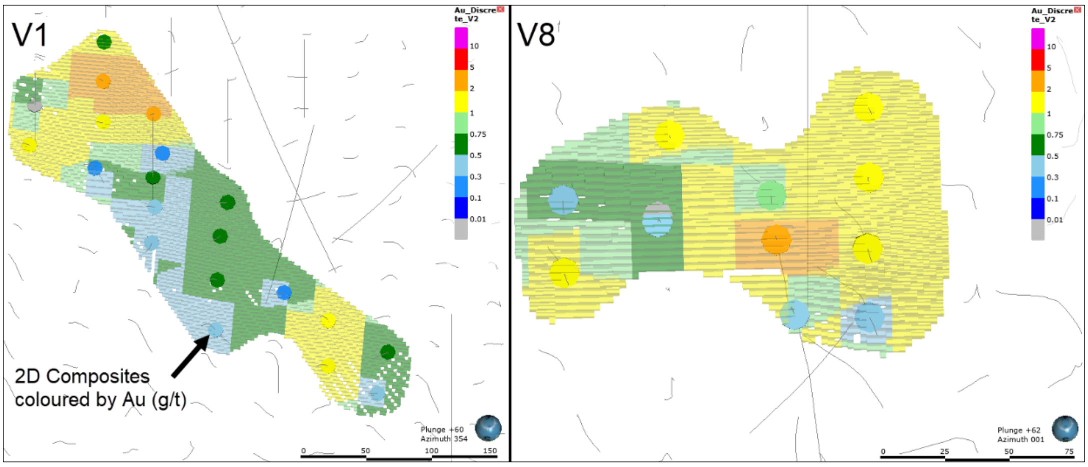


Figure 13-20: Isometric views showing estimated block grades versus input composite grades, each coloured by Au grade (g/t) for domains V1 and V8

13.11.2 Swath plots

As part of the validation process swath plots were generated in the X (easting), Y (northing), and Z (vertical) coordinate directions. Average grades for input samples and estimated blocks are calculated along a series of vertical and horizontal slices (swaths) and plotted on graphs. In effect, a moving average is calculated for blocks and samples along three coordinate axes; this enables the fit of the block model to the underlying data to be assessed. The number of samples per swath are plotted as bars.

Examples of swath plots for Au within the Main Mineralisation Zone and the V17 domains are shown in Figure 13-21 and Figure 13-22. Each of the mineralisation domains shows a good degree of correspondence between block model grades and composite grades in three dimensions, with the block model displaying a more smoothed profile, as anticipated with the Ordinary Kriging interpolation method used. Where the Inverse Distance (squared) interpolation method was used for the estimates in smaller, relatively poorly supported vein domains, the block estimates are typically slightly less smoothed.

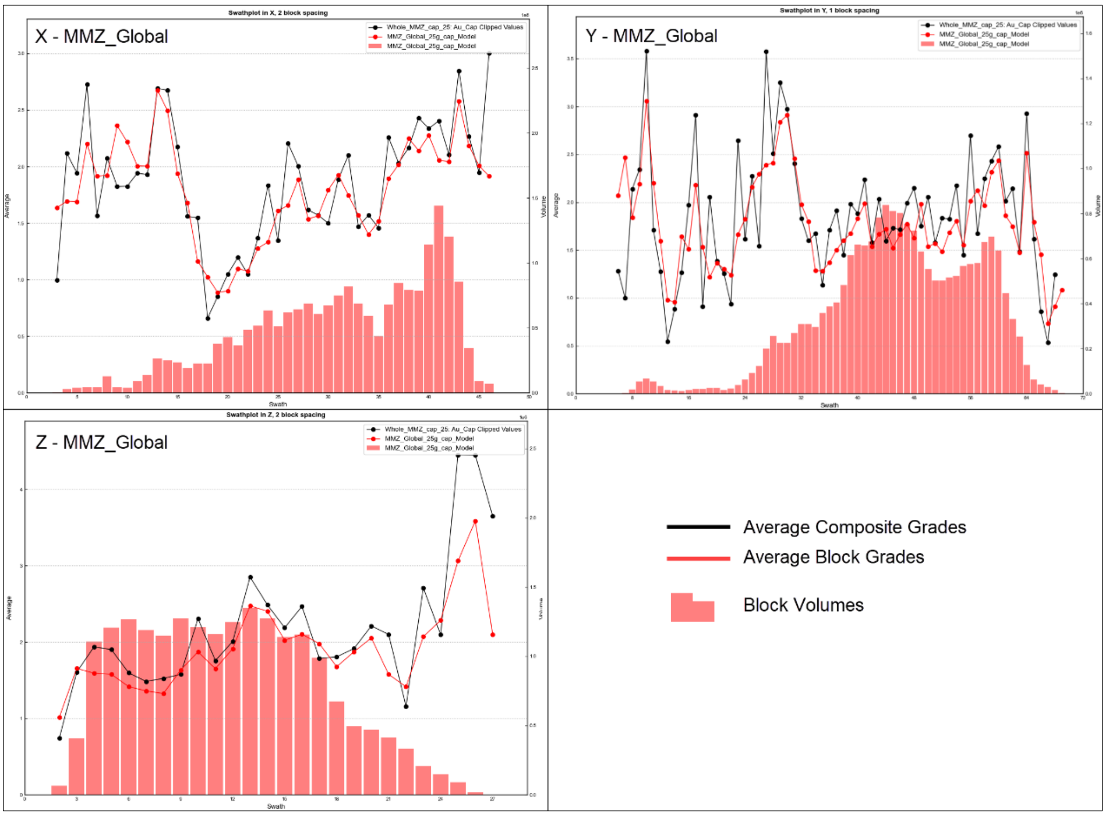


Figure 13-21: X, Y and Z swath plots showing estimated Au grades versus input composite grades for the Main Mineralisation Zone

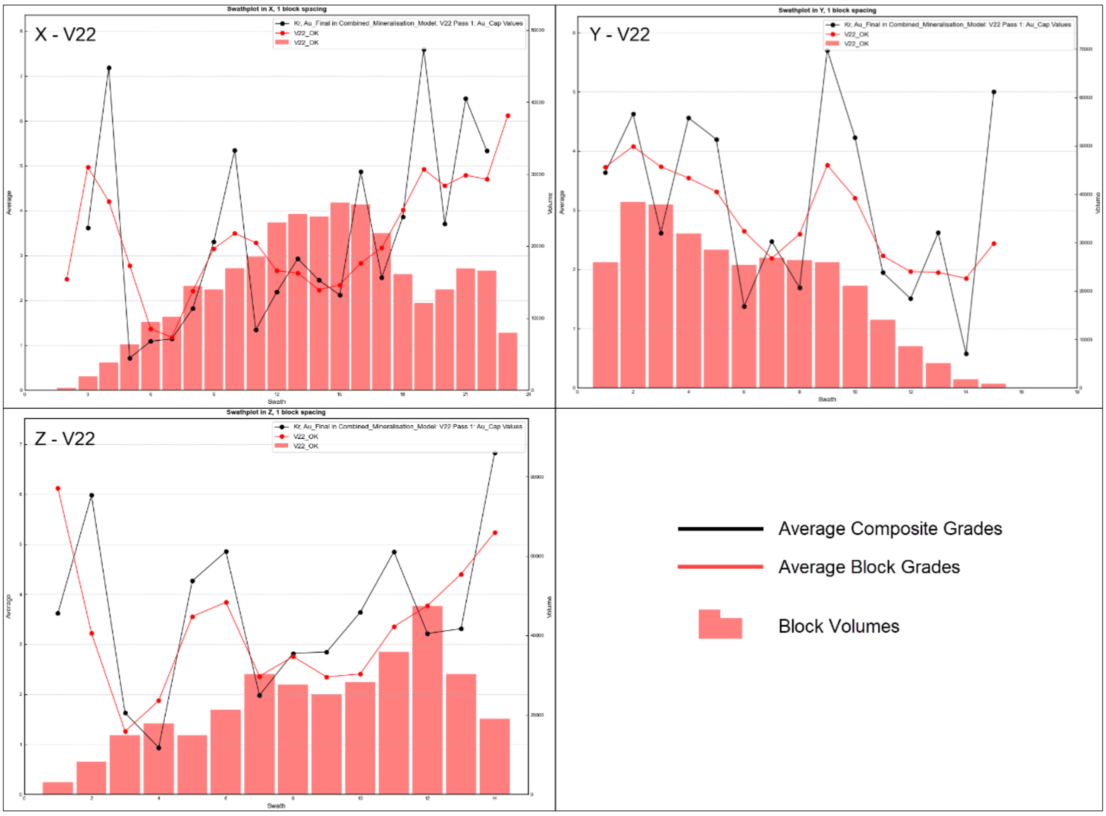


Figure 13-22: X, Y and Z swath plots showing estimated Au grades versus input composite grades for the Vein 22 domain

13.11.3 Statistical validation

To globally validate the estimates, the mean of the capped composite grades were compared to the mean of the estimated block grades, on a domain by domain basis (Table 13-10). Estimated Au grades generally correspond well with average input composite values, being within ±4% for each of the Main Mineralisation Zone structural sub-domains, and within 1% globally. There are larger discrepancies between estimated block mean and input composite means grades for some of the much smaller vein domains supported by relatively few composites (typically <50 composites). The mean composite grades of the smallest vein domains are sometimes significantly skewed by a small number of capped high-grade composites, however SRK is satisfied that through visual checks and reviews of swath plots, the estimated block grades of these domains are broadly representative of the input sample grades. The largest discrepancy between the mean composite and block grades is for domain V32, where two high grade intercepts influence a relatively large volume of blocks with high estimated grades.

Table 13-10: Mean composite grades compared to mean estimated block grades

Domain	Number of Composites	Capped Comp. Mean (g/t Au)	Decl. Capped Comp. Mean (g/t Au)	Block Mean (g/t Au)	% Diff.	Decl. Window Size (x,y,z in m)
MMZ	15,745	1.98	1.81	1.81	0%	20x20x10
V1	32	0.99	0.96	0.94	-2%	20x20x10
V2	28	1.60	1.76	1.79	2%	20x20x15
V3	64	0.84	0.86	0.84	-2%	20x20x10
V7	31	1.05	0.98	1.00	2%	20x20x10
V8	21	1.31	1.25	1.31	5%	10x10x5
V9	16	1.46	1.27	1.22	-4%	20x20x15
V13	24	3.00	2.27	2.32	2%	20x20x10
V16	57	1.21	1.26	1.16	-8%	20x20x10
V17	119	1.23	1.11	1.12	1%	25x25x15
V20	81	1.56	1.60	1.52	-5%	20x20x10
V21	63	1.36	1.20	1.16	-3%	25x20x10
V22	274	3.30	3.12	3.21	3%	20x20x10
V23	47	3.06	3.14	3.17	1%	25x20x10
V25	99	2.70	2.44	2.28	-7%	15x15x5
V26	37	2.18	2.29	2.23	-3%	20x20x5
V27	13	5.90	5.74	5.60	-2%	20x20x10
V28	72	8.59	7.20	7.49	4%	20x20x10
V29	68	2.54	2.39	2.44	2%	20x20x10
V30	33	1.39	1.37	1.22	-11%	20x20x5
V31	139	1.69	1.59	1.66	4%	15x15x5
V32	46	3.25	3.29	2.81	-15%	20x20x10
LAT	255	2.53	2.41	2.30	-5%	20x20x15
WMZ1	298	1.92	1.98	1.93	-3%	20x20x10

13.11.4 Global change of support analysis

In order to assess the degree of smoothing introduced into the estimated block model, SRK conducted a global change of support ("CoS") analysis using the Discrete Gaussian ("DG") method, whereby the estimated grade distribution is compared to a theoretical grade-tonnage curve for a range of parent block sizes.

The DG approach to Global CoS analyses is a relatively robust model which de-skews a grade distribution, providing a reasonable indication of a theoretical unsmoothed grade tonnage curve. This grade tonnage curve can then be compared to the OK estimate at the parent block size, which provides an indication of the level of smoothing within the OK model. The DG model is a useful methodology to indicate whether the kriging process has over or under smoothed the composite data. Large variations in the OK model from the DG model indicate that the kriging process would require review.

Figure 13-23 shows global CoS grade-tonnage curves for parent block sizes of 10 x 10 x 5 m and 20 x 20 x 10 m, alongside the corresponding OK model curves for the Main Mineralisation Zone. The grade tonnage curves indicate how the OK estimates have greater smoothing than the theoretical DG grade tonnage curves, as indicated by the gradient of the curves. The steeper the grade curve, the less smoothing is present in the model. This is to be expected with a smoothed OK model, as compared to the un-smoothed composite data. Figure 13-23 indicates that the OK estimates are relatively insensitive to parent block size around the reporting cut-off grade, though the global grade profile of the 20 x 20 x 10 m OK model shows a gradient/profile closer to the theoretical grade-tonnage profile and is therefore considered more appropriately smoothed than the OK model with 10 x 10 x 5 m parent blocks.

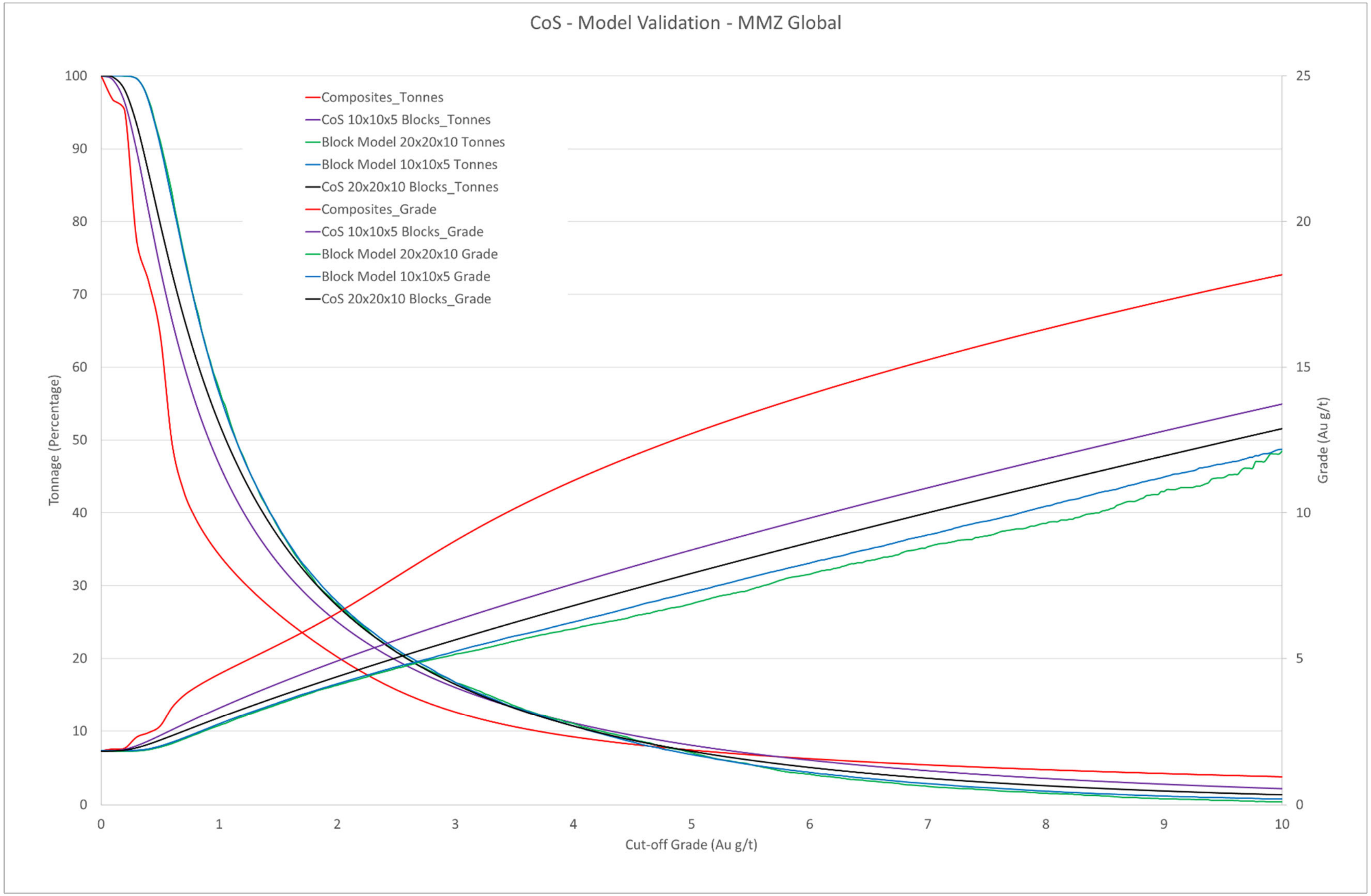


Figure 13-23: Global CoS grade-tonnage curves for the Main Mineralisation Zone compared with the corresponding OK block model grade-tonnage curves

Overall, SRK considers that the Au block estimates at the selected parent block size (20 x 20 x 10 m) reflect the current understanding of the distribution of mineralisation and is an acceptable basis for reporting the Mineral Resource statement.

13.12 Mining Depletion

To date, Lafigué has not been mined on a commercial scale but has been subject to substantial artisanal mining (Figure 13-24A). During SRK's site visit in May 2021 several thousand artisanal miners were active on the site, including some uncontrolled blasting activities. Concerns around safety inhibited the completion of a detailed survey of the artisanal workings at the time, however an aerial drone survey was completed on 17 August 2021 to assess the surface expression of the activities. The survey was of sufficient resolution to resolve the main open pit working areas which were typically on the scale of 10s metres at surface, and less than 10 m deep (Figure 13-25). Both SRK and the Client also observed deeper, and more laterally extensive trenches and access to underground workings, which were not resolvable from the drone survey (Figure 13-24B).

Figure 13-24: Photographs showing some of the artisanal workings observed during the SRK site to Lafigué on 15 May 2021

Since September 2021, Endeavour, alongside the Dabakala Gendarmes, have been undertaking an eviction exercise whereby the artisanal miners are being removed from site. Endeavour have stated in correspondence with SRK (Appendix E) that within the Resource area, which is to be fenced at Lafigué, approximately 30% of artisanal miners were removed by the end of September 2021, with this proportion increasing to 60% by mid-December 2021 and 98% by late January 2022. SRK understands that these figures are estimates. In addition, the site was bulldozed in Q1 2022 by Endeavour, with no survey of the underground workings having been conducted. SRK understands that, at the time of writing, the fence was still not completed, and the process of moving the artisanal miners from site was continuing.

In order to account for the artisanal mining depletion in the reporting of the current Mineral Resource Statement, and in the absence of a more recent survey since August 2021 and before the site was bulldozed in 2022, SRK has used the available height data obtained from the August 2021 drone survey to deplete the tonnes and grade from the main artisanal open pit excavations across the Lafigué deposit. In these areas, the density and grade fields in the block model have both been set to zero. In addition, in order to account for the depletion of the so-far poorly quantified volume of material mined from smaller trenches and underground workings, SRK has set all block grades to zero to a depth of 5 m below the pre-mining Lidar topographic surface within a defined set of boundaries considered to reflect the approximate lateral extent of artisanal mining activities (Figure 13-25 and Figure 13-26). Where the drone survey height data indicates that individual pits reach a depth greater than 5 m below the pre-mining topography, these volumes of the model have been depleted to the maximum depth extents surveyed. SRK considers this approach represents a reasonable approximation of the understanding of the average depth of workings across the deposit area. SRK stresses that some localised areas of mining are known to have reached significantly greater depths, including areas of up to 20 m below the pre-mining surface. SRK also highlights that the approach outlined above is based upon the last reliable survey of the site in August 2021, and that artisanal mining is known to have continued beyond this date. However, it has not been possible to improve on the estimate of the artisanal mining which occurred after the August 2021 drone survey due to the levelling of the site. SRK considers that this represents a risk to the Mineral Resource Statement presented herein and, in particular, the early stages of the mine plan.

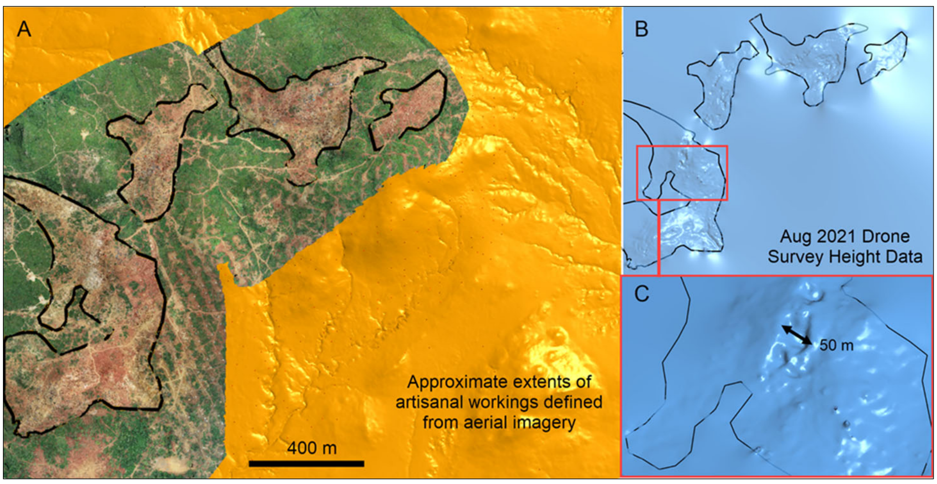


Figure 13-25: Plan images showing aerial imagery draped on the pre mining topography (A) with the interpreted approximate extents of artisanal mining activity. Image B and C show the drone survey height data (DTM) used to deplete the main open pit excavations

In order to assess the sensitivity of the stated Mineral Resources to the potential for more extensive or deeper artisanal depletion, SRK has completed an analysis to quantify the material currently present in the block model to depths of 10 m, 15 m and 20 m below the pre-mining/pre-levelled topography (Figure 13-26 and Table 13-11). In each case SRK has set the block model grades to zero but retained the block density so as to assume all Au metal has been depleted without significant mining of waste rock. In all scenarios both the density (and as such, the tonnage) and grade have been set to zero for all of the open pit volumes surveyed in August 2021. The table is reported on a global grade-tonnage inventory basis (Table 13-11, for illustrative purposes only). Table 13-11 shows that depletion to 5 m depth across the deposit results in a reduction of the oxide material inventory by 64%, with the majority of oxide material depleted at a depth of 10 m below the pre-mining topography. Transitional material is minimally impacted by depletion to a depth of 5 m, and only significantly impacted by depletion to a depth of 15 m or greater. SRK notes that there is limited impact on the material currently classified as Inferred Mineral Resources as the majority of this material is located in the down-dip areas of the deposit.

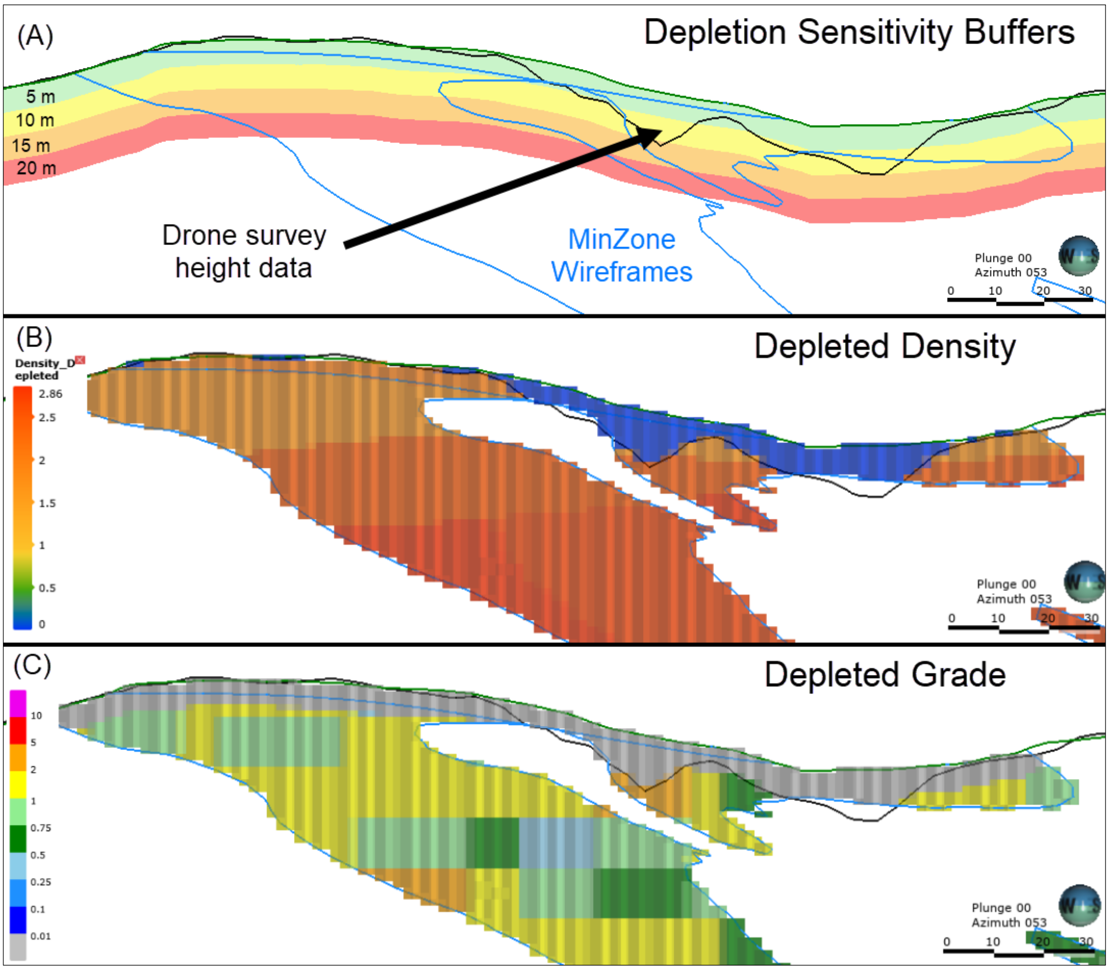


Figure 13-26: Cross-sections showing, (A) the drone depletion survey surface (black line) and pre-mining topographic buffer volumes used for the depletion sensitivity analysis (5, 10, 15 and 20 m); (B) block model depleted density accounting for the volume of material mined from the drone surveyed pits; and (C) block model depleted Au grade accounting for both the surveyed pits and also to a uniform depth of 5 m below pre-mining topography outside of the surveyed pits.

Table 13-11: Summary of the block model inventory inclusive of a sensitivity to artisanal depletion – Indicated and Inferred material (for illustrative purposes only)

	Material Type	Tonnes (Mt)	Average grade (g/t Au)	Contained Metal (koz)	% Difference Metal
Undepleted	Oxide	1.8	1.9	111	-
	Transition	1.9	1.7	103	-
	Fresh	45.2	2.1	2,989	-
	Total	**48.8**	**2.0**	**3,203**	-
After application of 5 m depletion	Oxide	0.8	1.5	40	-64%
	Transition	1.8	1.7	99	-4%
	Fresh	45.2	2.1	2,989	0%
	Total	**47.8**	**2.0**	**3,128**	**-2%**
After application of 10 m depletion	Oxide	0.4	1.3	16	-86%
	Transition	1.6	1.7	86	-16%
	Fresh	45.2	2.1	2,989	0%
	Total	**47.2**	**2.0**	**3,091**	**-3%**
After application of 15 m depletion	Oxide	0.2	1.2	7	-93%
	Transition	1.2	1.6	62	-39%
	Fresh	45.1	2.1	2,985	0%
	Total	**46.5**	**2.0**	**3,055**	**-5%**
After application of 20 m depletion	Oxide	0.1	1.1	5	-96%
	Transition	0.7	1.6	37	-64%
	Fresh	45.1	2.1	2,979	0%
	Total	**45.9**	**2.0**	**3,021**	**-6%**

*In all depletion scenarios, block grade and density values were set to zero above the provided drone survey height data. Below this level, only grade was set to zero to the specified depths.

In the absence of a detailed survey of all of the artisanal workings across the Project area, SRK highlights that the artisanal workings at Lafigué present a potentially significant risk to the early stages of the mine plan.

13.13 Mineral Resource Classification

The Mineral Resource estimate for Lafigué has been classified in accordance with the CIM Definition Standards and includes Indicated and Inferred Mineral Resources. In addition to the quality and quantity of exploration data supporting the estimates, SRK has considered the confidence in the geological continuity of the mineralised structures and the confidence in the tonnage and grade estimates, specifically:

- Grade data for the drilling campaigns has generally been collected and analysed using industry best practice. Where documentation of operating procedures is not available for historical drilling, this drilling has in most places been supported by close spaced 2017-2019 drilling. Adequate quality control measures are in place to monitor laboratory performance, drillhole collars have been surveyed using a differential GPS, and downhole surveys have been collected appropriately.
- The QAQC analyses presented are generally of sufficient quality to support the subsequent geological modelling and grade and tonnage estimate.

- The current geological model is based on a combination of litho-structural and assay data derived from DD and RC exploration drilling. Where stacked E-W-trending shear structures and lithology contacts act as the primary mineralisation-controlling features, confidence in the modelled grade continuity is relatively high for the main mineralised structures (MMZ) but somewhat reduced for some of the smaller, less continuous veins domains, particularly in Lafiguè Centre.
- The quality of the grade estimations has been reviewed on a global and local basis using various validation techniques and is considered a reasonable representation of the input sample grades.

SRK considers that the quality and spatial distribution of the data used, the geological continuity of the mineralisation and the quality of the estimated block model for Lafigué is sufficient for the reporting of Indicated and Inferred Mineral Resources, in accordance with the CIM Definition Standards. Isometric and cross-sectional views of the classified block model are shown in Figure 13-27. A summary of the specific criteria used to classify the block model is provided below:

Indicated Mineral Resources:

Where exploration drillholes used for the grade estimation are typically spaced at 20-40 m along sections, and 40-50 m between sections, providing a reasonable level of confidence in geological and grade continuity, SRK has classified this material as Indicated Mineral Resources. These areas of the model show a reasonable degree of grade continuity and typically coincide with modelled lithological contacts or lie within the intrusive felsic unit. These areas are also typically estimated by search passes 1 or 2 (see Section 13.9.2).

Inferred Mineral Resources:

Where exploration drillholes used for the grade estimation are typically spaced at 50 to 75 m in areas along-strike and down-dip from areas classified as Indicated Mineral Resources, SRK has classified this material as Inferred Mineral Resources. Additionally, areas drilled at closer spacings but where mineralisation controls are less well understood, and continuity is typically reduced, such as observed in some parts of Lafigué Centre, are also classified as Inferred Mineral Resources. These areas of the block model were primarily estimated in search passes 2 or 3.

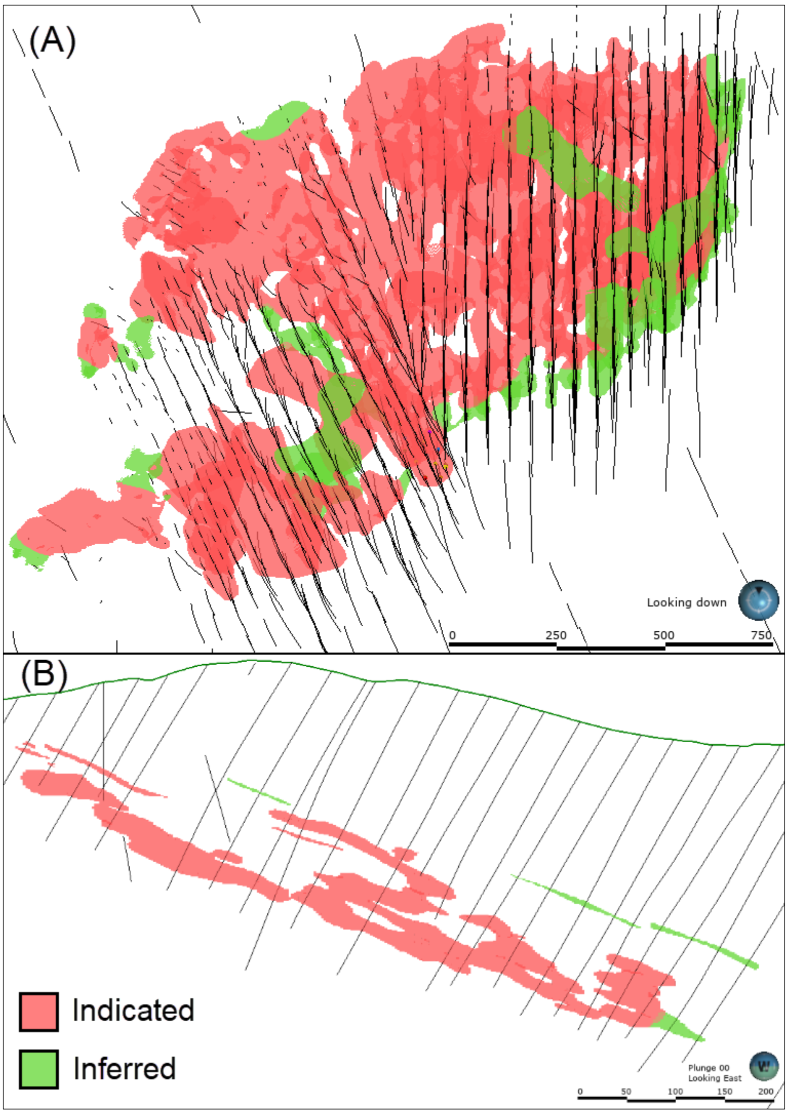


Figure 13-27 Isometric (A) and cross-section (B) views of the classified Lafigué block model with exploration drillholes used for the grade estimation shown as black traces

13.14 Assessment of Reasonable Prospects for Eventual Economic Extraction ("RPEEE")

13.14.1 Economic and technical input parameters

In order to determine which portion of the block model has reasonable prospects for eventual economic extraction by open-pit mining methods, SRK has applied basic economic considerations based on previous technical studies in order to generate an optimised pit shell within which the Mineral Resource is to be reported. The pit optimisation study has been carried out on the Mineral Resource based on a relatively optimistic gold price and technical parameters established as part of previous/current studies. The pit optimisation identifies material within the model with potential for open pit mining above a suitable gold cut-off grade. SRK has reviewed the economic and technical parameters used in the pit optimisation exercise and considers them appropriate for the purpose of indicating the proportion of the block model that demonstrates reasonable prospects for eventual economic extraction ("RPEEE").

The parameters used for the pit optimisation exercise are summarised in Table 13-12.

Table 13-12: Summary of key assumptions and technical / economic parameters for the Lafigué conceptual pit optimisation and cut-off grade calculation

Parameters	Units	Value	Source/Basis
Production			
Production Rate	(Mtpa)	4.00	
Geotechnical			
Max Surface Elevation	(Z Elevation)	420	
Laterite & Saprolite (Oxide) & Transition	(Z Elevation)	Above weathering wireframe OSA	
	(Deg)	33.0	END-011-GD DRAFT Report.pdf
Fresh	(Z Elevation)	Below weathering wireframe IOSA	
	(Deg)	33.0 – 51.0	END-011-GD DRAFT Report.pdf
Mining Factors			
Dilution	(%)	9%	Regularisation - block size 5x5x2.5
Recovery	(%)	98%	Regularisation- block size 5x5x2.5
Processing			
Recovery – Au (ox)	(%)	94.87	IF(au>=34,99.7,IF(au<=0.2,(68*au+82.2),(0.769*LN(au)+97)))/100)-0.5%
Recovery – Au (tr)	(%)	94.92	
Recovery – Au (fr)	(%)	95.08	Lycopodium 2021
Operating Costs			
Wavg Waste Mining Cost	(US$/$t_{rock}$)	2.65	2021 PFS Financial Model
Incremental Mining Cost	(USD/m bench)	0.0022	2021 PFS Financial Model
Reference Level	(Z Elevation)	350.00	2021 PFS Financial Model
Wavg Ore Mining Cost	(US$/$t_{ore}$)	2.12	2021 PFS Financial Model
Rehandle Cost	(US$/$t_{ore}$)	0.37	2021 PFS Financial Model
OFF ROM Rehandle	(US$/$t_{ore}$)	0.79	2021 PFS Financial Model
Rehabilitation Cost	(USD/t_{rock})	0.06	2021 PFS Financial Model
CIL - Oxide	(US$/$t_{ore}$)	7.47	Lycopodium 2021
CIL - Transition	(US$/$t_{ore}$)	7.47	Lycopodium 2021
CIL - Fresh	(US$/$t_{ore}$)	9.13	Lycopodium 2021
G&A	(US$/$t_{ore}$)	5.60	2021 PFS Financial Model
Sustaining Capital (SIB)	(USD/t_{ore})	1.87	2021 PFS Financial Model
Selling Cost Au (@USD1,500/oz)	(US$/oz)	71.8	EDV 2021 Assumptions
Metal Price			
Gold	(US$/oz) (US$/g)	1,500 49.83	EDV 2021 Assumptions
Discount Rate	(%)	5%	EDV 2021 Assumptions
Cut-Off Grade			
Marginal Operating Costs (ox)	(USD/t_{ore})	16.71	
Marginal Operating Costs (tr)	(USD/t_{ore})	16.86	
Marginal Operating Costs (fr)	(USD/t_{ore})	19.11	
Marginal Cut-Off Grade (ox)	(g/t Au)	0.4	
Marginal Cut-Off Grade (tr)	(g/t Au)	0.4	
Marginal Cut-Off Grade (fr)	(g/t Au)	0.4	
IS Marginal Cut-Off Grade (ox)	(g/t Au)	0.4	
IS Marginal Cut-Off Grade (tr)	(g/t Au)	0.4	
IS Marginal Cut-Off Grade (fr)	(g/t Au)	0.5	

t_{ore} = ore tonnes; t_{rock} = total rock tonnes; IS = In Situ; Wavg = Weighted average

13.14.2 Cut-off grade

Based on the above pit optimisation study and associated technical and economic input parameters, the in-situ marginal cut-off grades determined for reporting the Mineral Resource are given below:

- 0.4 g/t Au for oxide;
- 0.5 g/t Au for transition; and
- 0.5 g/t Au for fresh.

13.15 Mineral Resource Statement

The 2021 Mineral Resource statement for the Lafigué gold deposit is shown in Table 13-13.

Table 13-13: Mineral Resource statement for the Lafigué Gold Project, effective of 15 May 2022*

Classification	Material Type	Tonnes Mt	Au Grade g/t	Contained Metal koz
Indicated	Oxide	0.7	1.55	36
	Transition	1.7	1.71	94
	Fresh	43.8	2.06	2,896
	Total	**46.3**	**2.03**	**3,027**
Inferred	Oxide	0.1	1.22	4
	Transition	0.1	2.05	4
	Fresh	1.4	2.11	94
	Total	**1.5**	**2.05**	**102**

**In reporting the Mineral Resource Statement, SRK notes the following:*

- *The reported Mineral Resources are depleted to a drone survey provided to SRK by Endeavour. The survey was conducted on 17 August 2021 and only accounts for artisanal open pit development at surface. SRK understands that there were further artisanal mining workings underground, but these could not be captured by the drone survey. To account for this, outside of (and below, where necessary) the artisanal open pit workings, to a depth of 5m below the pre-mining topography, the grades have been reduced to zero. In the absence of any underground survey, and to reflect the uncertainty for these areas, SRK has not depleted the tonnages.*
- *Since September 2021, Endeavour have been undertaking an eviction exercise whereby the artisanal miners are being removed from site. Endeavour have stated in correspondence with SRK that as of late January 2022 98% of the artisanal miners were removed. In the absence of an updated survey and groundworks completed at site, SRK highlights the risk associated with more extensive depletion due to ongoing artisanal mining activity in the intervening period and or more extensive workings in the prior period, than is accounted for in this Mineral Resource Statement. A sensitivity analysis is provided in the accompanying report to inform the reader of the associated risks.*
- *The reported Mineral Resources have an effective date of 15 May 2022. The Competent Person for the declaration of Mineral Resources is Dr Lucy Roberts, MAusIMM(CP), of SRK Consulting (UK) Ltd. The Mineral Resource estimate was authored by Dr James Davey, also of SRK;*
- *Technical and economic assumptions were agreed between SRK and LMCI/Endeavour for mining factors (mining and selling costs, mining recovery and dilution, pit slope angles) and processing factors (gold recovery, processing costs), which were used to run a pit optimisation exercise. These factors were developed as part of the ongoing Feasibility Study for the Lafigué project, as stated below:*
  - *Mining cost: 2.12 (US$/$t_{ore}$)*
  - *Waste mining cost: 2.65 (US$/$t_{rock}$)*
  - *Processing cost: Oxide/Transition: 7.47 (US$/$t_{ore}$); Fresh: 9.13 (US$/t_{ore})*
  - *Selling cost: 71.8 (US$/oz Au)*
  - *Mining recovery: 98%*
  - *Mining dilution: 9%*
  - *Processing recovery: Oxide = 94.87%; Transition = 94.92%; Fresh = 95.08%*
  - *Average slope angles: 33-51°, dependent on geotechnical domain*
  - *G&A cost: 5.60 (US$/$t_{ore}$)*
  - *Discount rate: 5%*
- *SRK considers there to be reasonable prospects for eventual economic extraction by constraining the Mineral Resources within an optimised open pit shell using a gold price of USD 1,500/oz.*
- *Mineral Resources are reported within the optimised pit shell using cut-off of grades of 0.4 g/t Au (oxide); 0.5 g/t Au (transition) and 0.5 g/t Au (fresh), which are the marginal cut-off grades for CIL processing determined during the pit optimisation.*
- *Mineral Resources are reported as in-situ and undiluted, with no mining recovery applied in the Statement. All tonnages are reported on a dry basis.*
- *Mineral Resources are not Ore Reserves and do not have demonstrated economic viability, nor have any mining modifying factors been applied;*
- *Tonnages are reported in metric units, grades in grams per tonne (g/t), and the contained metal in kilo troy ounces. Tonnages, grades, and contained metal totals are rounded appropriately. 1 troy ounce is assumed to be the equivalent of 31.1034 g*
- *Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content. Where these occur, SRK does not consider these to be material.*

13.16 Grade-tonnage Curves

The results of grade-tonnage sensitivity analysis completed for Indicated Mineral Resources (given the very limited total contribution of Inferred Mineral Resources) at Lafigué are shown in Figure 13-28 to Figure 13-30, split by material type. This is to show the continuity of the grade estimates at various cut-off increments and the sensitivity of the Mineral Resource to changes in Au (g/t) cut-off. The tonnages and grades in these charts, however, should not be interpreted as Mineral Resource statements.

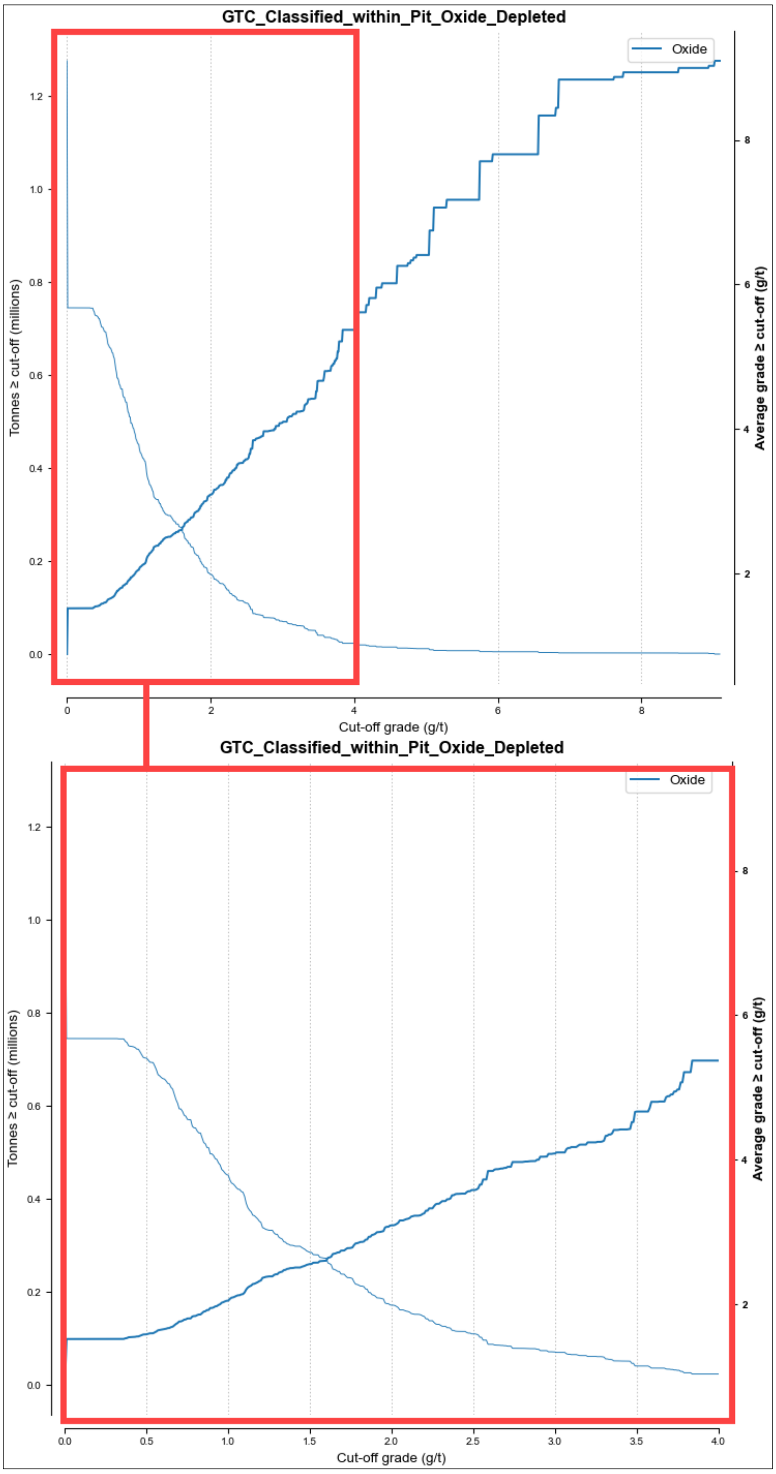


Figure 13-28: Grade-tonnage curve for Indicated oxide material within the US$1,500 optimised pit shell after depletion

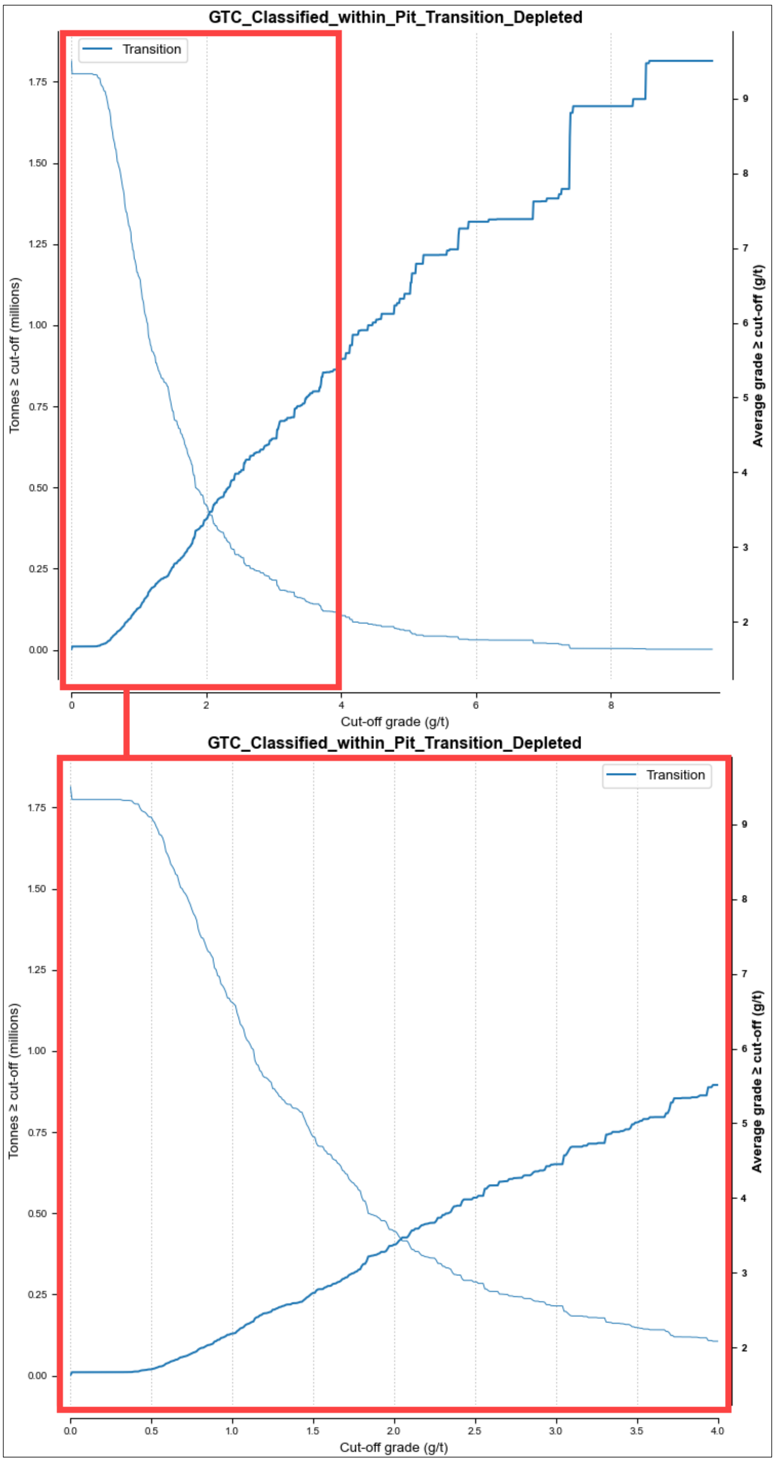


Figure 13-29: Grade-tonnage curve for Indicated transition material within the US$1,500 optimised pit shell after depletion

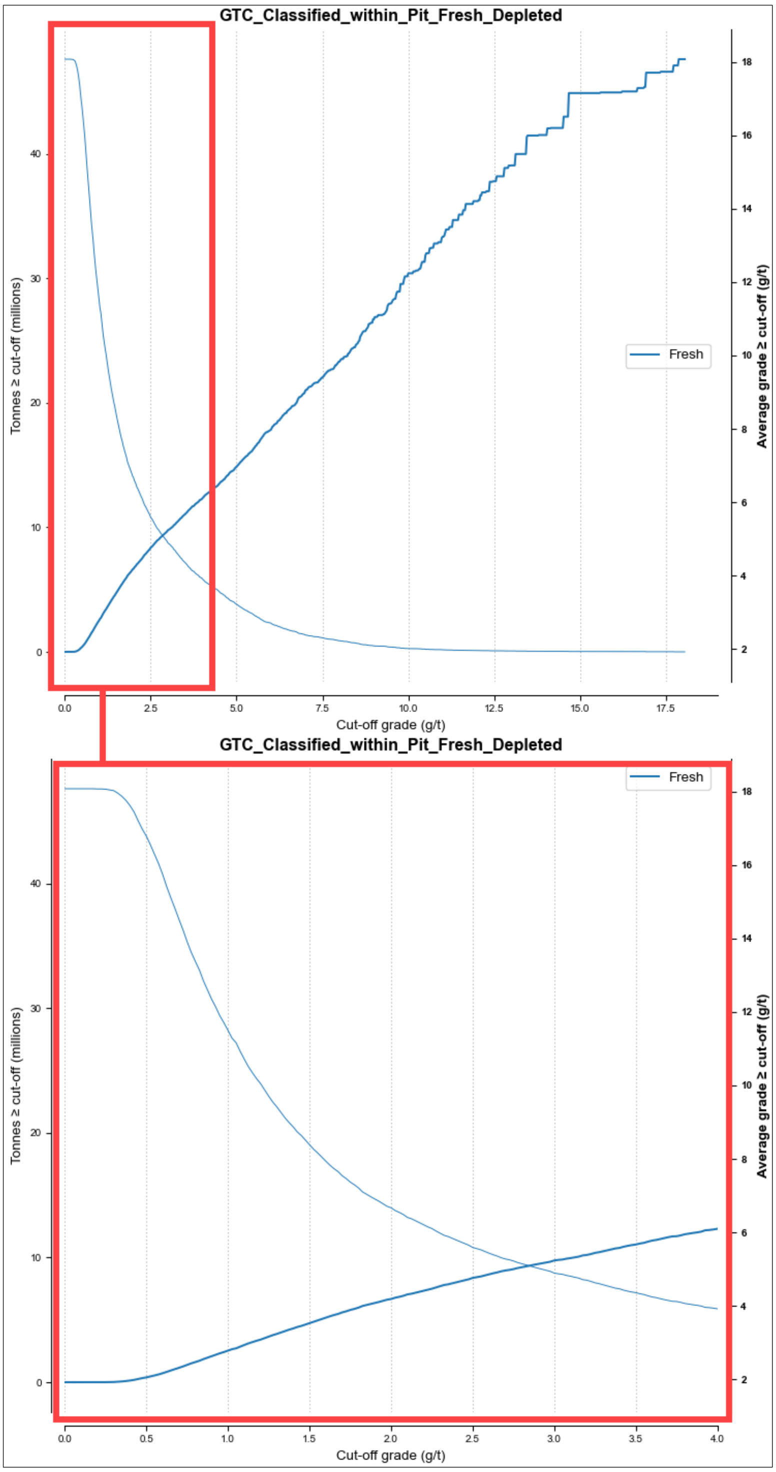


Figure 13-30: Grade-tonnage curve for Indicated fresh material within the US$1,500 optimised pit shell after depletion

13.17 Comparison with Previous Estimates

The previous Mineral Resource Statement for Lafigué was effective as of 21 September 2021, based on a previous version of the Mineral Resource model produced by SRK. A comparison of this Mineral Resource Statement with the 2022 SRK Resource Statement is provided in Table 13-4

Table 13-14: Comparison of the 2021 and 2022 Resource Statements (based on the respective SRK models)

Model	Reporting Pit	Cut-off Grade (g/t Au)	Classification	Tonnes (Mt)	Grade Au (g/t)	Contained Metal Au (koz)
SRK (2021)	2021 (Au price: USD1500/ oz)	Oxide: 0.4 Transition: 0.5 Fresh: 0.5 (IS MCOG)	Measured	-	-	-
			Indicated	44.8	2.0	2,917
			Inferred	3.6	2.4	270
			Total	**48.4**	**2.0**	**3,186**
SRK (2022)	2022 (Au price: USD1500/ oz)	Oxide: 0.4 Transition: 0.5 Fresh: 0.5 (IS MCOG)	Measured	-	-	-
			Indicated	46.3	2.03	3,027
			Inferred	1.5	2.05	102
			Total	**47.8**	**2.04**	**3,128**

**Rounding may result in apparent summation differences between tonnes, grade and contained metal content*

Since the SRK 2021 model was produced, approximately 245 additional infill drillholes have been completed during late 2021 and early 2022 in the project area, primarily in Lafigué Centre and around the periphery of the deposit. In the Lafigué Centre area the modelled mineralised structures were refined in the updated (2022) model to account for shorter-scale variability that was not apparent based on the previously available, wider-spaced drillholes, and some additional mineralised structures were also modelled. On a local basis some areas of the model reduced in volume whereas others increased (Figure 13-31), with intercepted grades generally remaining aligned with the global grade distribution already established from previous drilling and sampling at the deposit. Both the 2021 and 2022 SRK models define relatively broad packages of mineralisation above the natural cut-off of the available sample population. As a result, both models incorporated a greater proportion of samples within the 0.3 to 0.5 g/t Au grade range than pre-2021 models produced for the Lafigué Project.

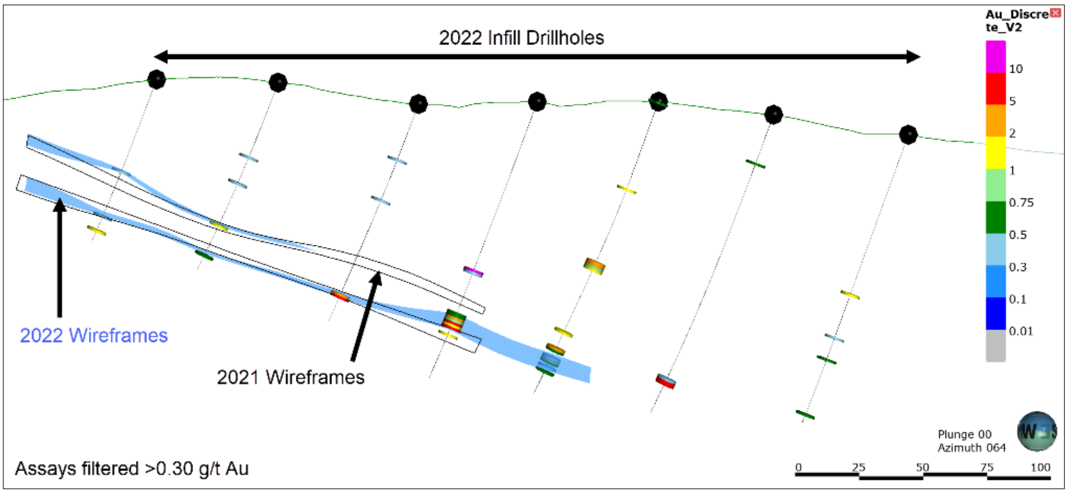


Figure 13-31: Cross-section (looking east) in Lafigué Centre comparing the extents of the 2021 wireframes (black outlines) and the 2022 wireframes (blue solids)

The majority of the technical and economic parameters used during the pit optimisation process completed for the purposes of satisfying the RPEEE test have not been modified or updated since the reporting of the 2021 Mineral Resource Statement for the Project.

14 INTERPRETATION AND CONCLUSIONS

SRK considers the geology at Lafigué to be reasonably well understood, with extensive exploration drilling, along with a dedicated structural studies of the deposit interpreting the main mineralisation-controlling structures and their dominant trends having been completed.

SRK is satisfied that the drilling and sampling procedures employed by LMCI/Endeavour, including collar and downhole surveys, logging, photography, core cutting, and sampling typically conform to industry best practice. Where these details are not fully documented for historical drilling (i.e., pre-2010), this drilling comprises a relatively small component of the overall database supporting the MRE and is often supported by relatively close spaced younger drilling. Overall, the QC sample analyses indicate that the data collected during the 2010 to 2022 drilling programmes are not influenced by any significant bias, contamination, or analytical issues.

SRK considers that the geological model developed for the 2022 MRE is a reasonable representation of the in situ mineralisation based on the available supporting data. The Mineral Resource classification categories attributed to the mineralised packages estimated in the 2022 model reflect SRK's confidence in the quality of the supporting data, as well as the level of understanding of both geological and grade continuity of the deposit, where these are reduced in Lafigué Centre and the farthest down-dip areas.

When comparing the SRK 2022 geological model versus the Endeavour 2020 model, the Mineral Resource Estimate has proven to be sensitive to the modelling approach, in particular, the degree to which lower grade mineralisation above the specified modelling grade threshold (0.3 g/t) is incorporated into the model, and the degree to which the resultant volumes are interpreted to represent the style and geometry of the in-situ mineralisation.

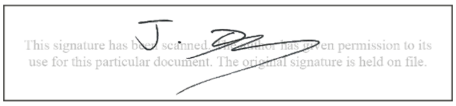
Overall, SRK is comfortable that the estimates of Au are a reasonable representation of both the input composites at the current level of sampling, and the style of mineralisation, as it is currently understood.

15 RECOMMENDATIONS

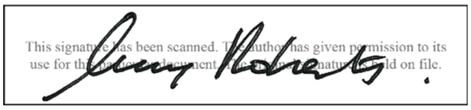
SRK recommend the following to improve confidence in the geological model and quality of the Mineral Resource estimates in future:

- The addition of some close-spaced exploration drilling or RC grade control drilling would be useful to investigate grade continuity in more detail. This is particularly the case in areas of the model where the continuity of the mineralisation wireframes is significantly supported by relatively low-grade composites, and where the continuity of mineralised structures reduces significantly in the Lafigué Centre area.
- Although Lafigué has not been mined on a commercial scale, SRK note that there has been significant artisanal mining activity at the site. SRK has depleted the declared Mineral Resources using elevation data obtained by a drone survey conducted by Endeavour the Client on 17 August 2021, and to a depth considered broadly appropriate for the average depth of artisanal workings at the time of the SRK site visit. However, in the absence of a detailed survey of these workings, this presents a potentially significant risk to the early stages of the mine plan. Where possible, a detailed survey and/or mapping of any workings not destroyed by the bulldozing of the site is recommended to be completed on an ongoing basis as overburden stripping and mining commences during the earliest periods of the mine life.

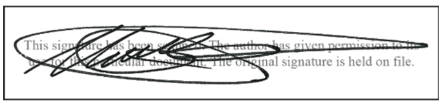
For and on behalf of SRK Consulting (UK) Limited



James Davey,
Consultant (Resource Geology),
Project Manager
SRK Consulting (UK) Limited



Lucy Roberts,
Principal Consultant (Resource Geology),
SRK Consulting (UK) Limited



Tim Lucks,
Managing Director and Principal Consultant (Geology & Project Management)
Project Director
SRK Consulting (UK) Limited

Date issued: 30 September 2022

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Glossary

Glossary Item	text inserted as example for definition of the term included in the Glossary as appropriate.

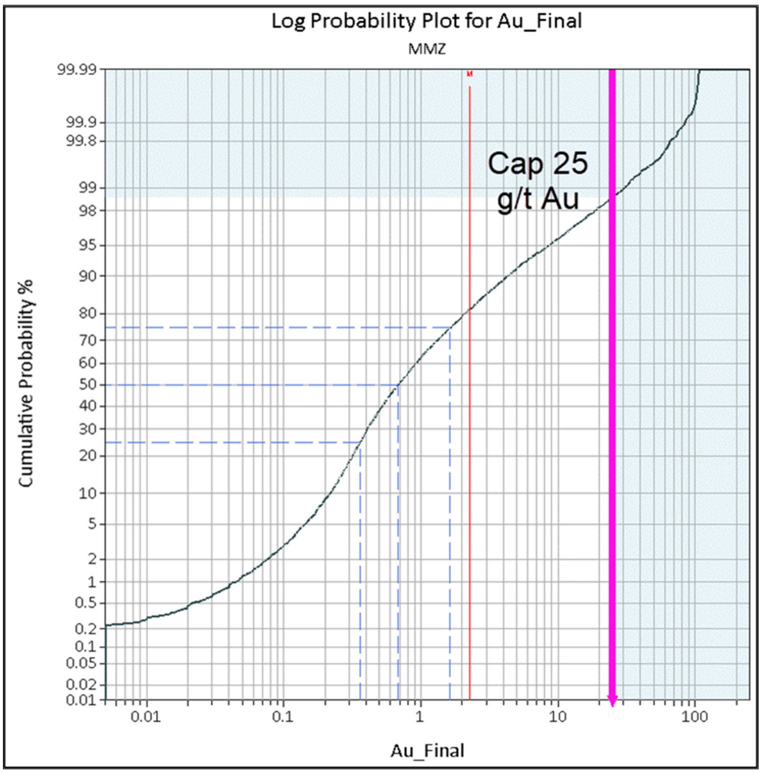
Abbreviations

IMMM	text inserted as example for definition of the term included in the Glossary as appropriate.

Units

Mt	Million metric tonnes

APPENDIX
A CAPPING LOG-PROBABILITY PLOTS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99.8
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80
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60
50
40
30
20
10
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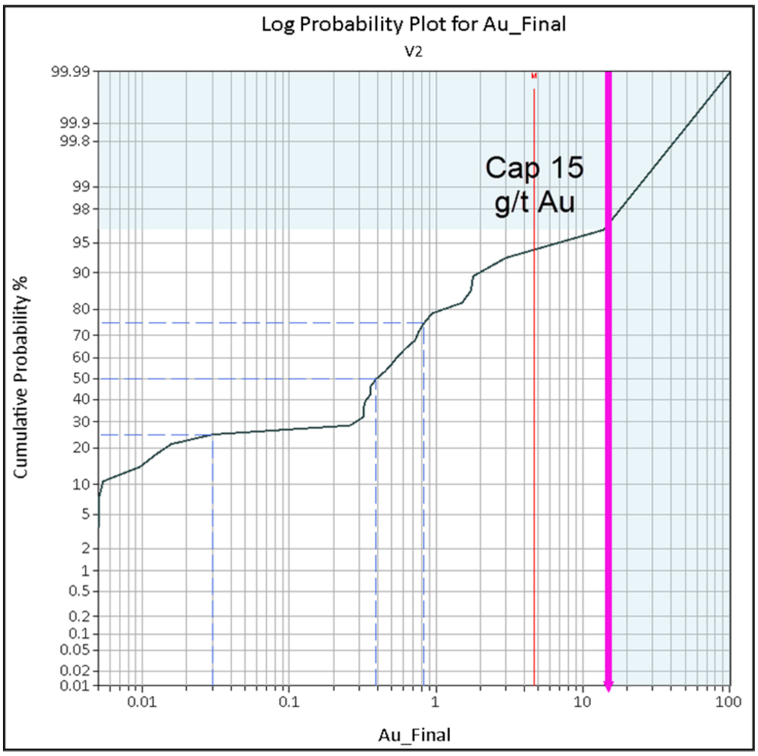
Au_Final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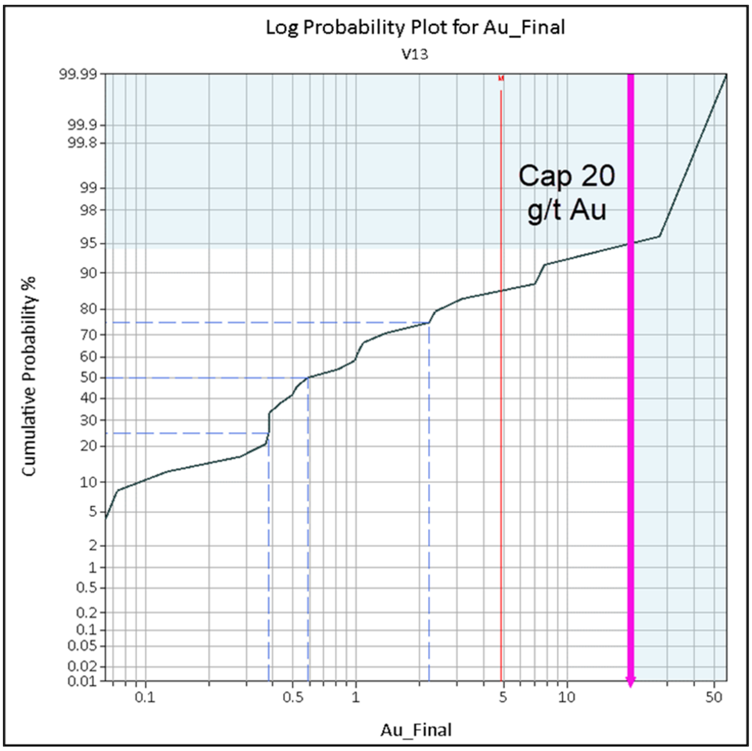
Au_Final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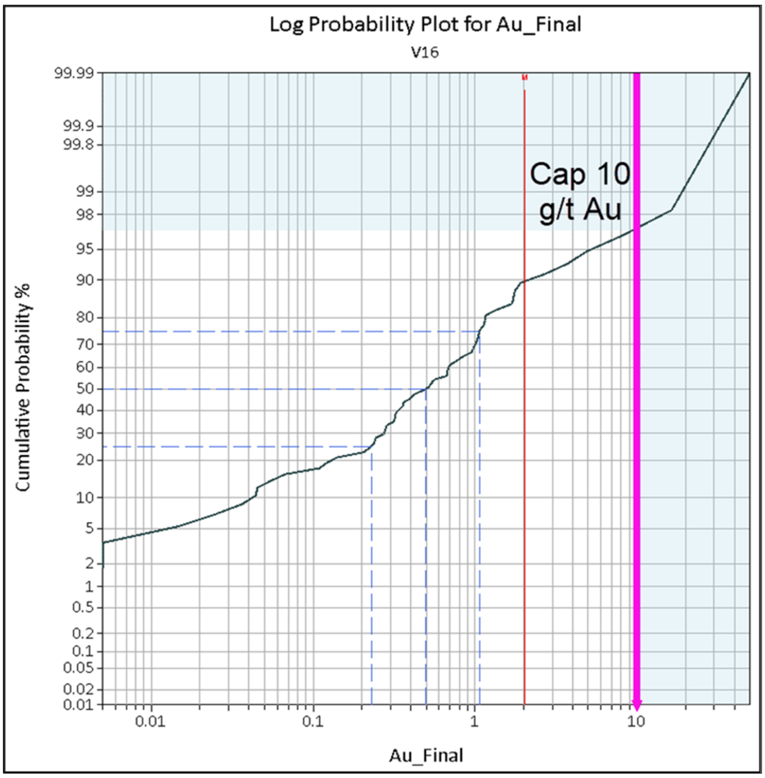
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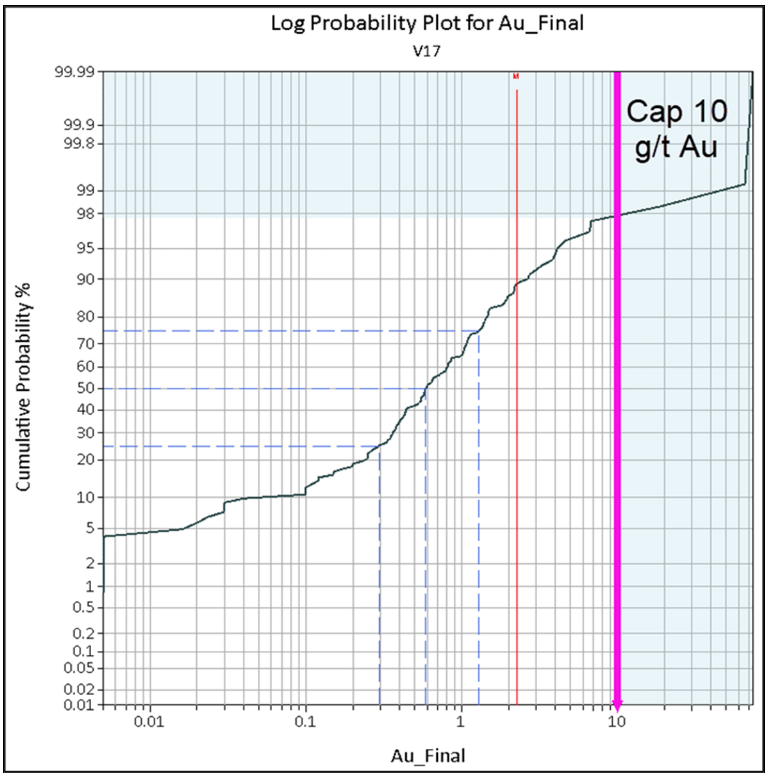
1
10
Au_Final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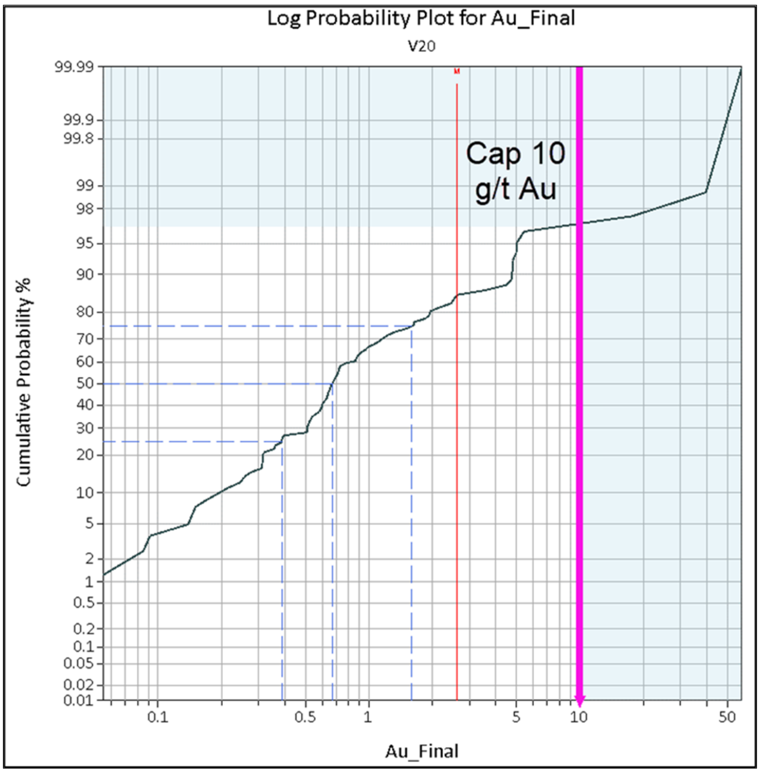
Cumulative Probability %
Au_Final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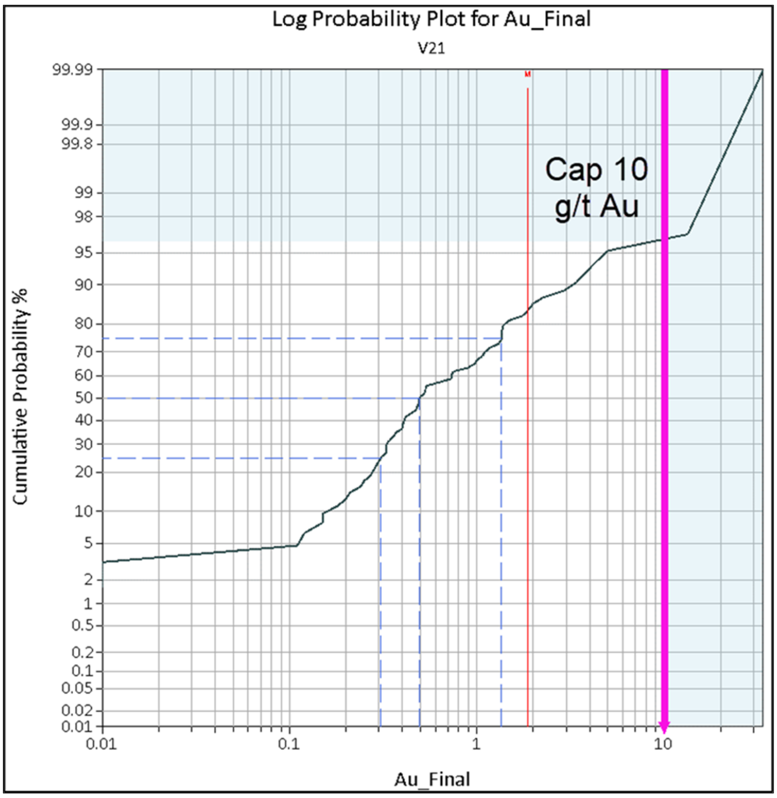
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Cap 10
g/t Au
Cumulative Probability %
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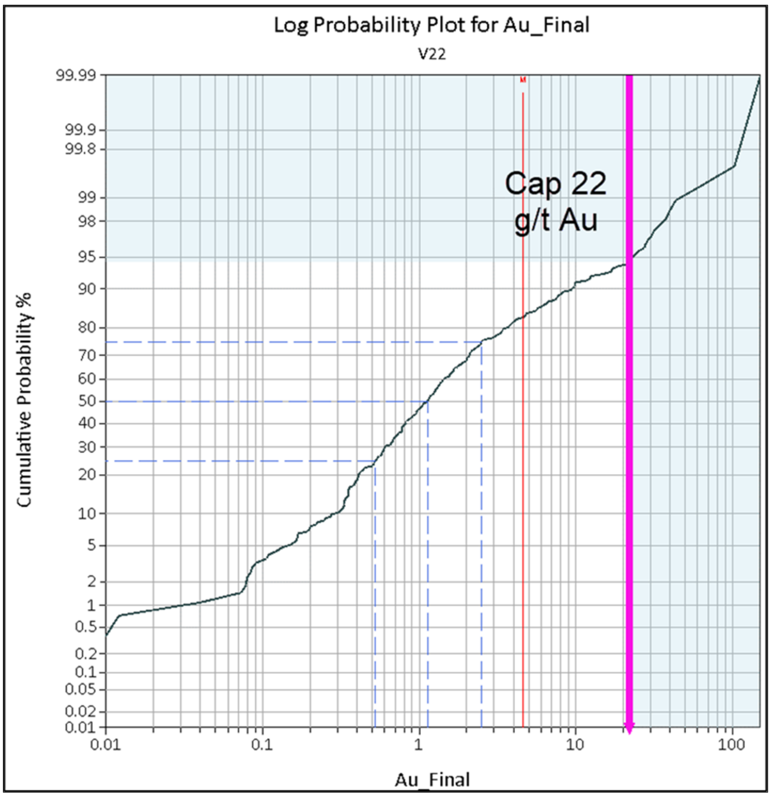
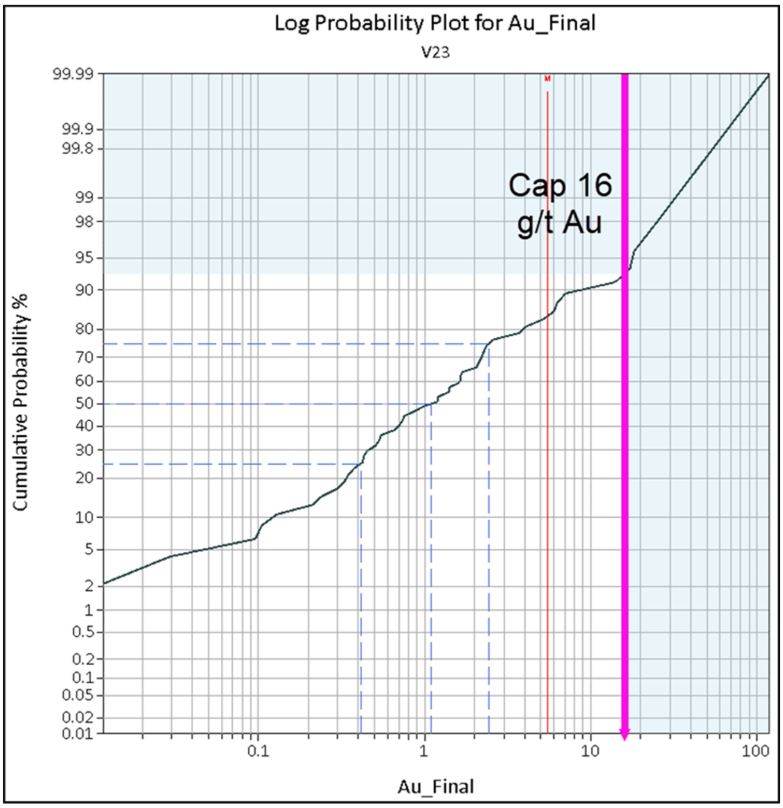
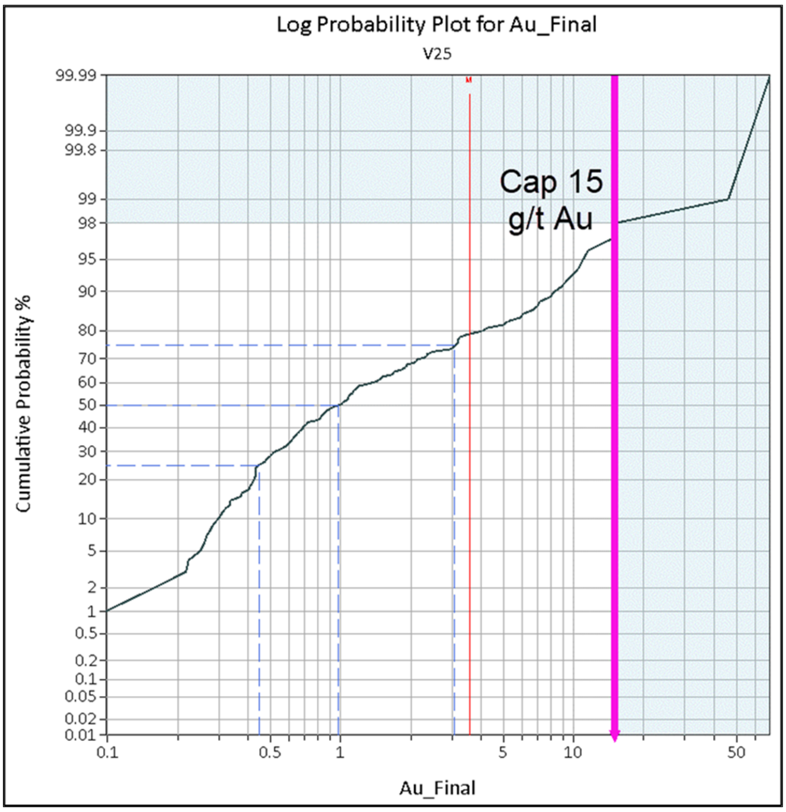
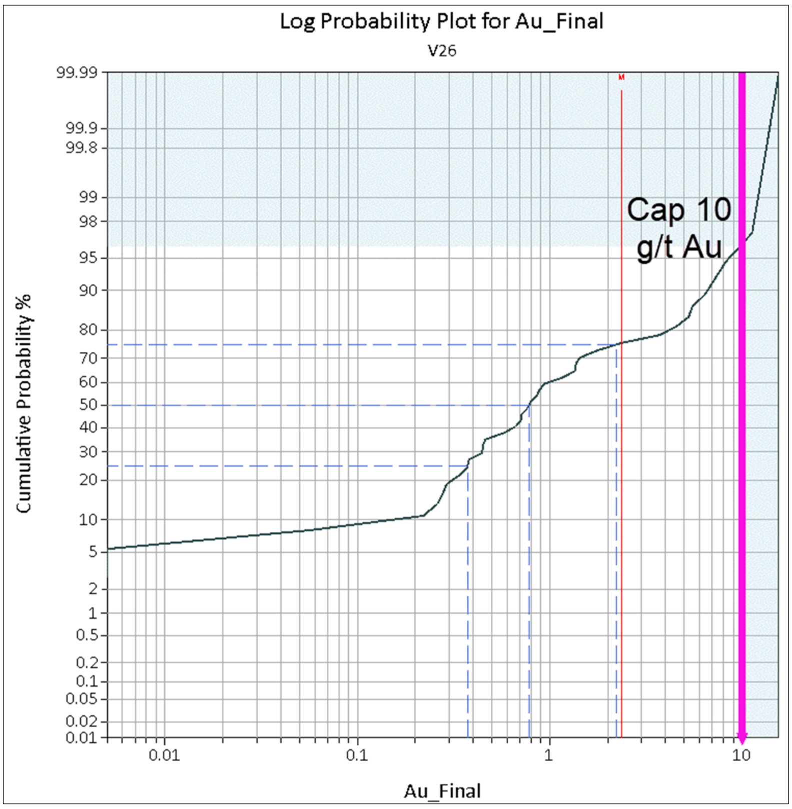
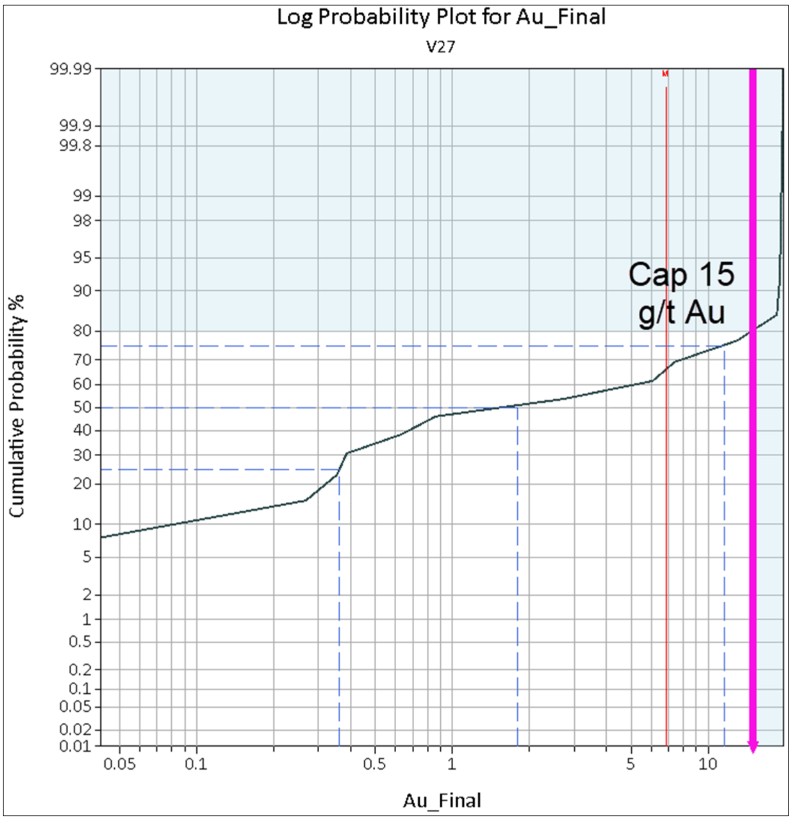
99.8
99
98
95
90
80
70
60
50
40
30
20
10
5
2
1
0.5
0.2
0.1
0.05
0.02
0.01
0.01
0.1
1
10

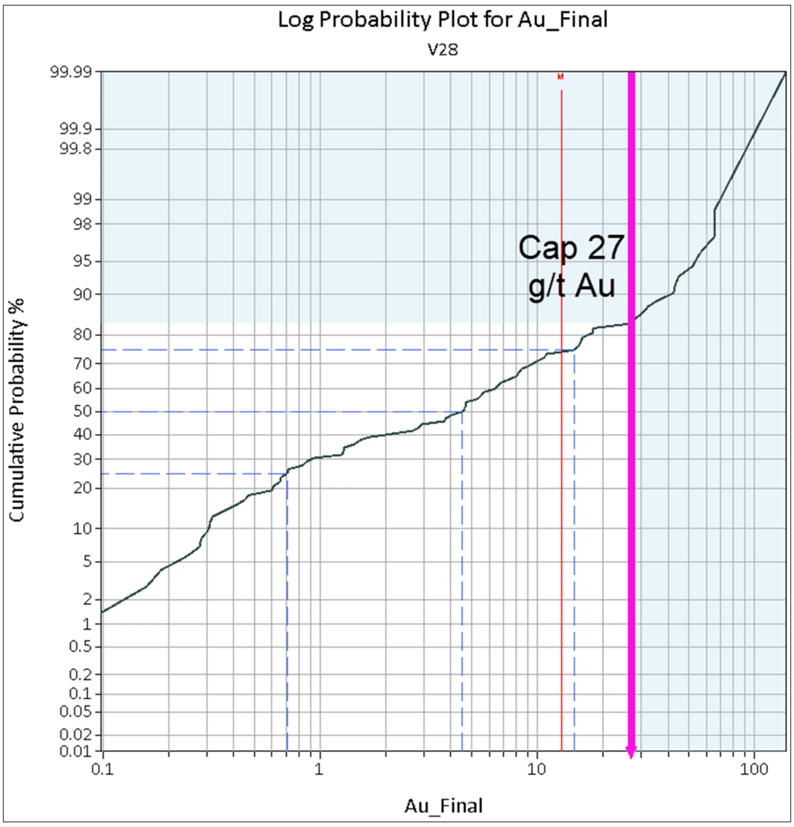

Log Probability Plot for Au_Final
V22
Cap 22
g/t Au
Cumulative Probability %
Au_Final
99.99
99.9
99.8
99
98
95
90
80
70
60
50
40
30
20
10
5
2
1
0.5
0.2
0.1
0.05
0.02
0.01
0.01
0.1
1
10
100

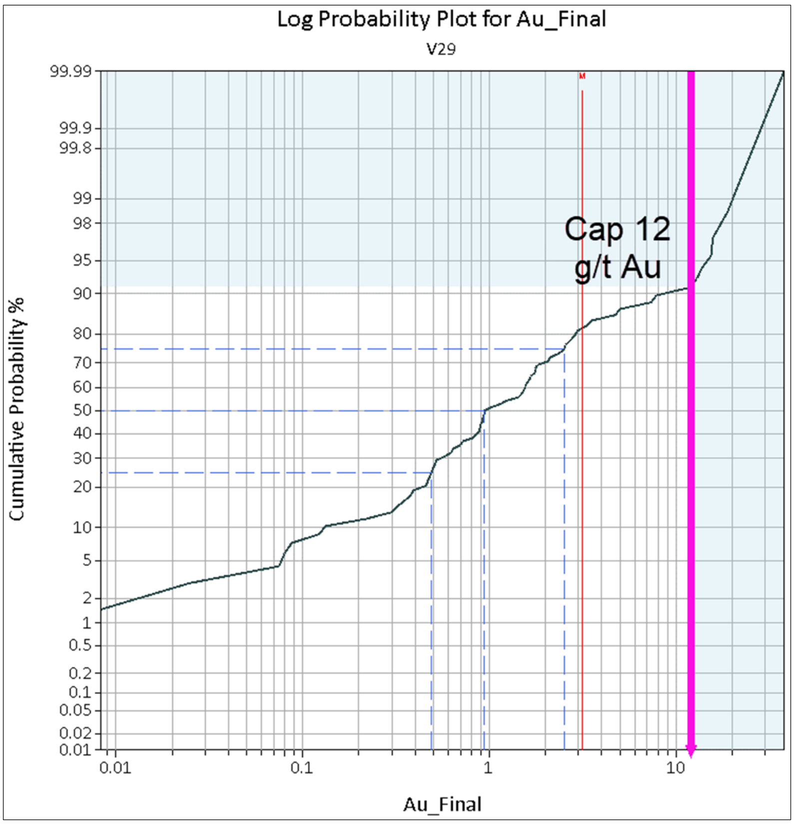

Log Probability Plot for Au_Final
V23
Cap 16
g/t Au
Cumulative Probability %
Au_Final

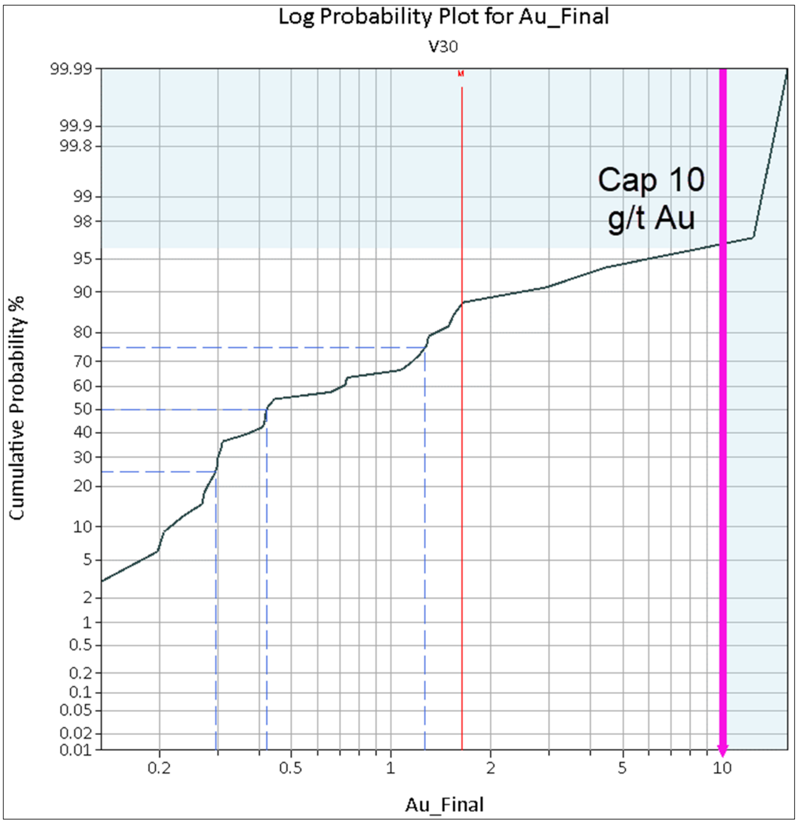

Log Probability Plot for Au_Final
V25
Cap 15
g/t Au
Cumulative Probability %
Au_Final

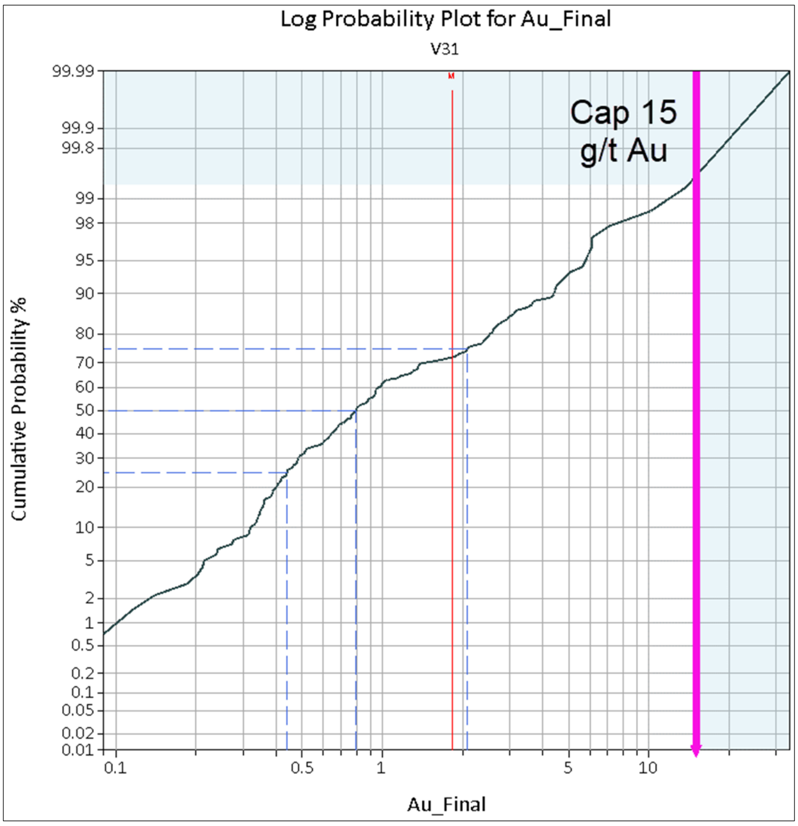

Log Probability Plot for Au_Final
V26
Cap 10 g/t Au
Cumulative Probability %
Au_Final
99.99
99.9
99.8
99
98
95
90
80
70
60
50
40
30
20
10
5
2
1
0.5
0.2
0.1
0.05
0.02
0.01
0.01
0.1
1
10

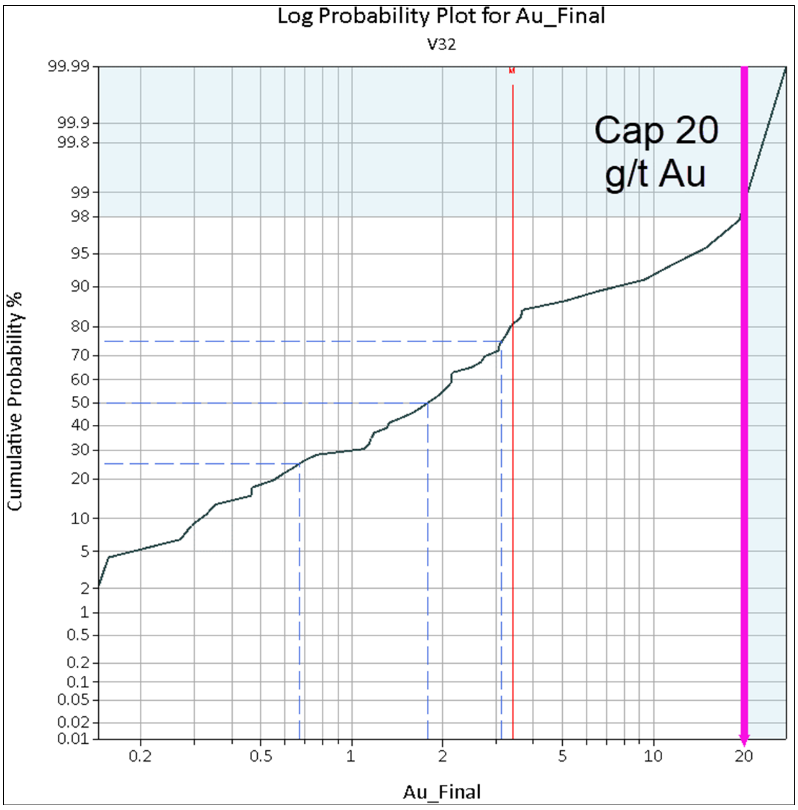

Log Probability Plot for Au_Final
V27
Cap 15 g/t Au
Cumulative Probability %
Au_Final
99.99
99.9
99.8
99
98
95
90
80
70
60
50
40
30
20
10
5
2
1
0.5
0.2
0.1
0.05
0.02
0.01
0.05
0.1
0.5
1
5
10

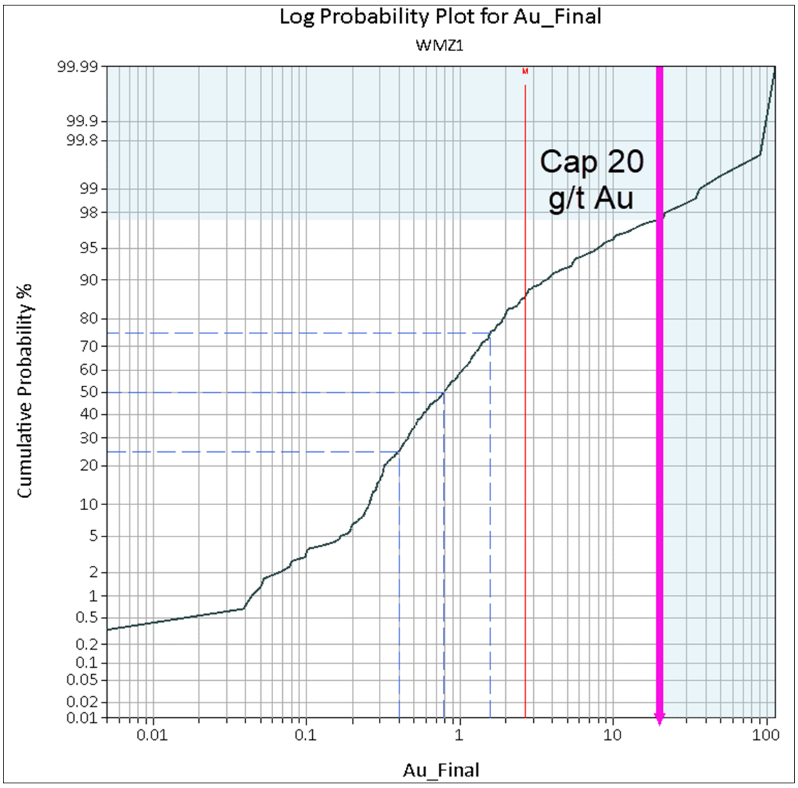

Log Probability Plot for Au_Final
V28
Cap 27
g/t Au
Cumulative Probability %
Au_Final
99.99
99.9
99.8
99
98
95
90
80
70
60
50
40
30
20
10
5
2
1
0.5
0.2
0.1
0.05
0.02
0.01
0.1
1
10
100

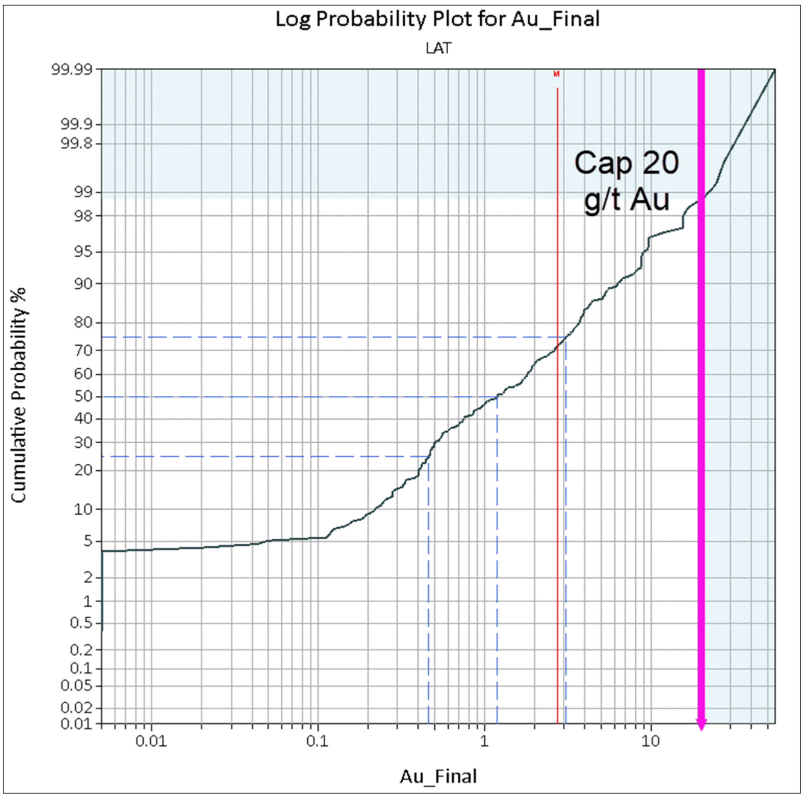

Log Probability Plot for Au_Final
V29
Cap 12
g/t Au
Cumulative Probability %
Au_Final
99.99
99.9
99.8
99
98
95
90
80
70
60
50
40
30
20
10
5
2
1
0.5
0.2
0.1
0.05
0.02
0.01
0.01
0.1
1
10


Log Probability Plot for Au_Final
V30
Cap 10
g/t Au
Cumulative Probability %
99.99
99.9
99.8
99
98
95
90
80
70
60
50
40
30
20
10
5
2
1
0.5
0.2
0.1
0.05
0.02
0.01
0.2
0.5
1
2
5
10
Au_Final


Log Probability Plot for Au_Final
V31
Cap 15
g/t Au
Cumulative Probability %
99.99
99.9
99.8
99
98
95
90
80
70
60
50
40
30
20
10
5
2
1
0.5
0.2
0.1
0.05
0.02
0.01
0.1
0.5
1
5
10
Au_Final

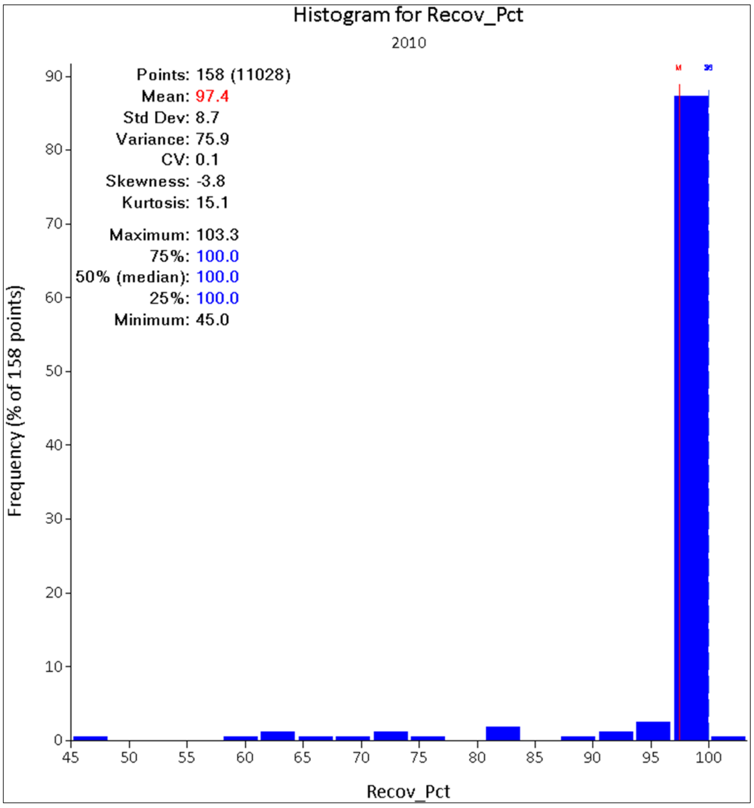

Log Probability Plot for Au_Final
V32
Cumulative Probability %
Au_Final
Cap 20 g/t Au


Log Probability Plot for Au_Final
WMZ1
Cumulative Probability %
Au_Final
Cap 20 g/t Au

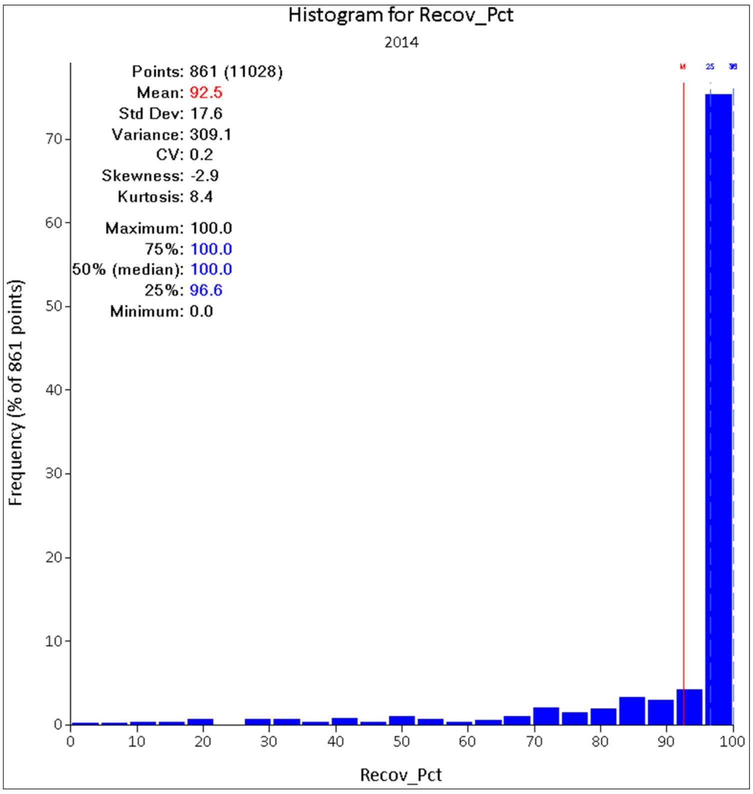

Log Probability Plot for Au_Final
LAT
Cap 20 g/t Au
Cumulative Probability %
99.99
99.9
99.8
99
98
95
90
80
70
60
50
40
30
20
10
5
2
1
0.5
0.2
0.1
0.05
0.02
0.01
0.01
0.1
1
10
Au_Final

APPENDIX B DRILLING RECOVERY

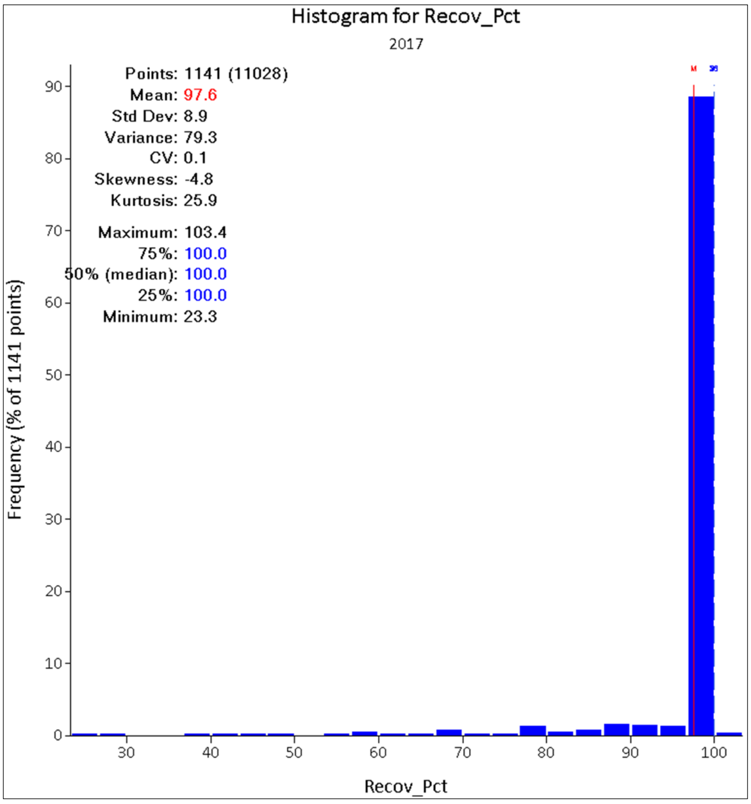
2010 Drilling:



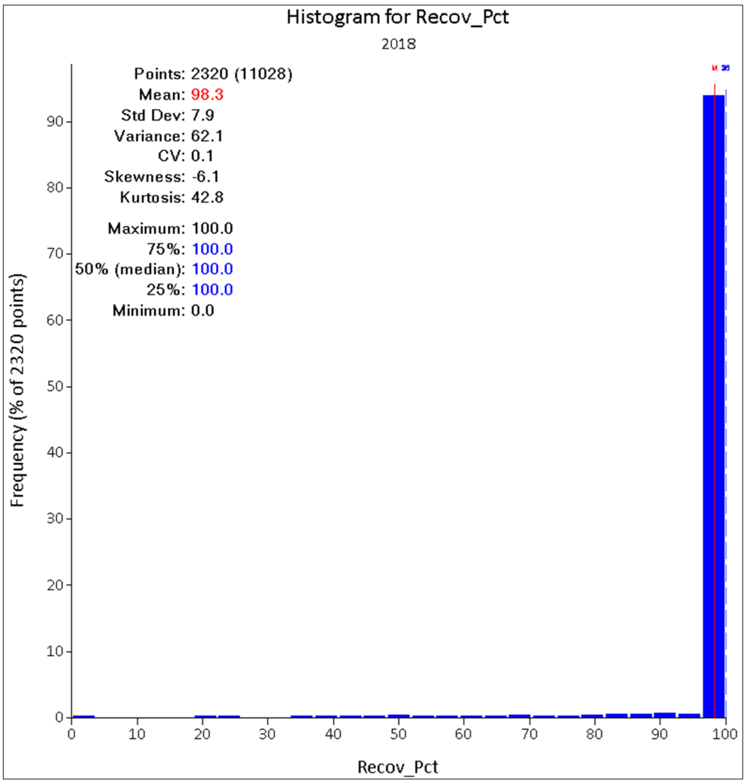
2014 Drilling:



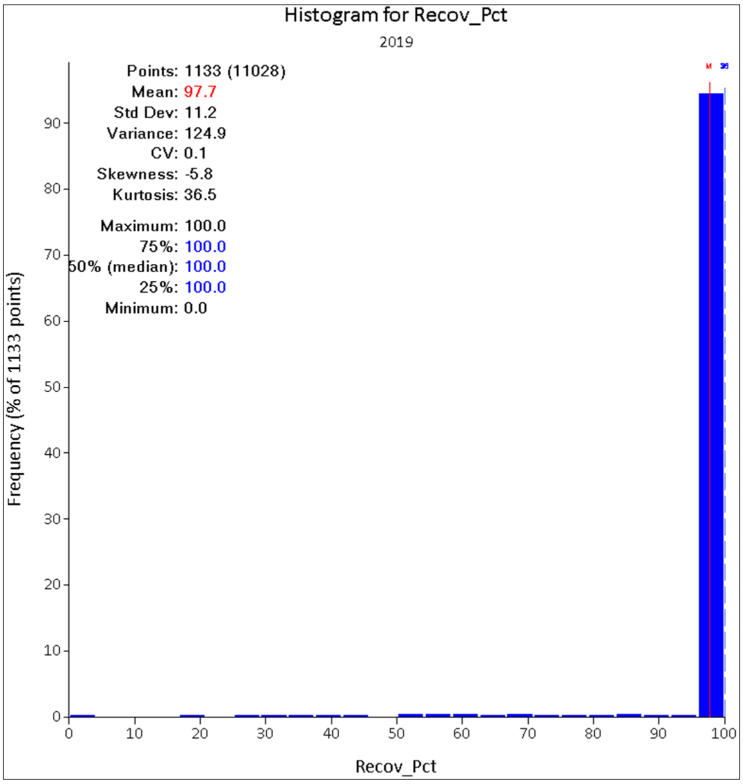
2017 Drilling:



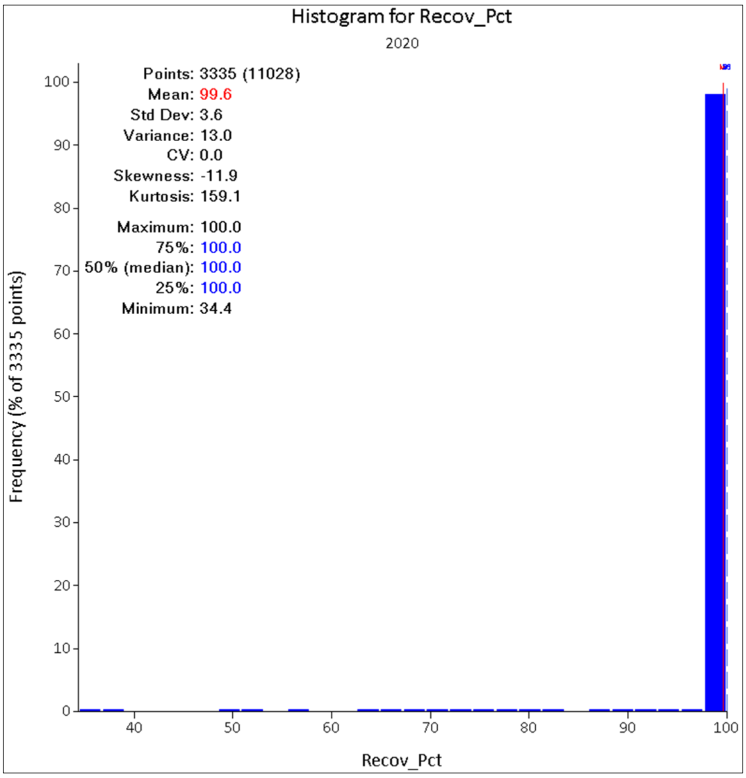
2018 Drilling:



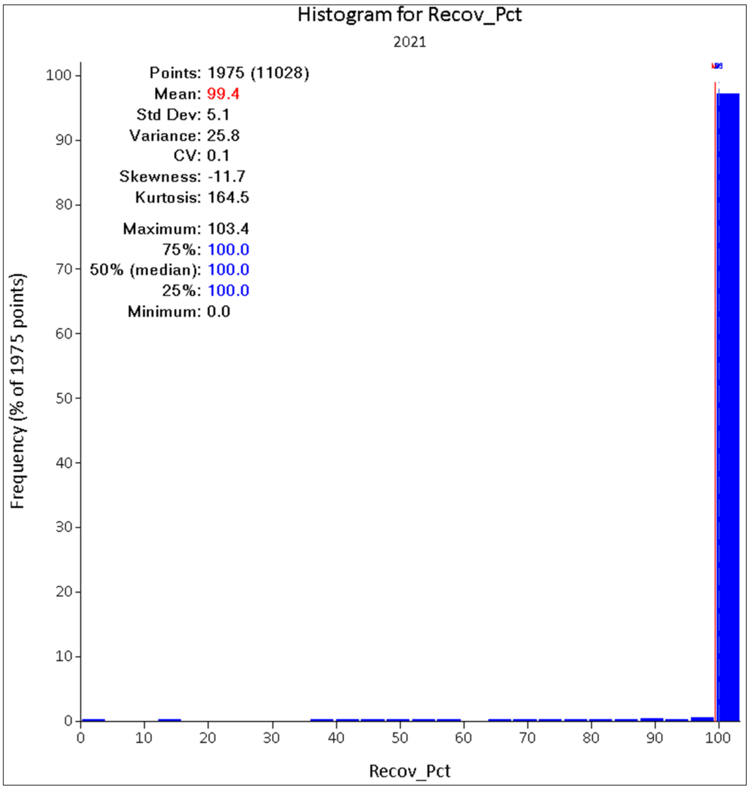
2019 Drilling:



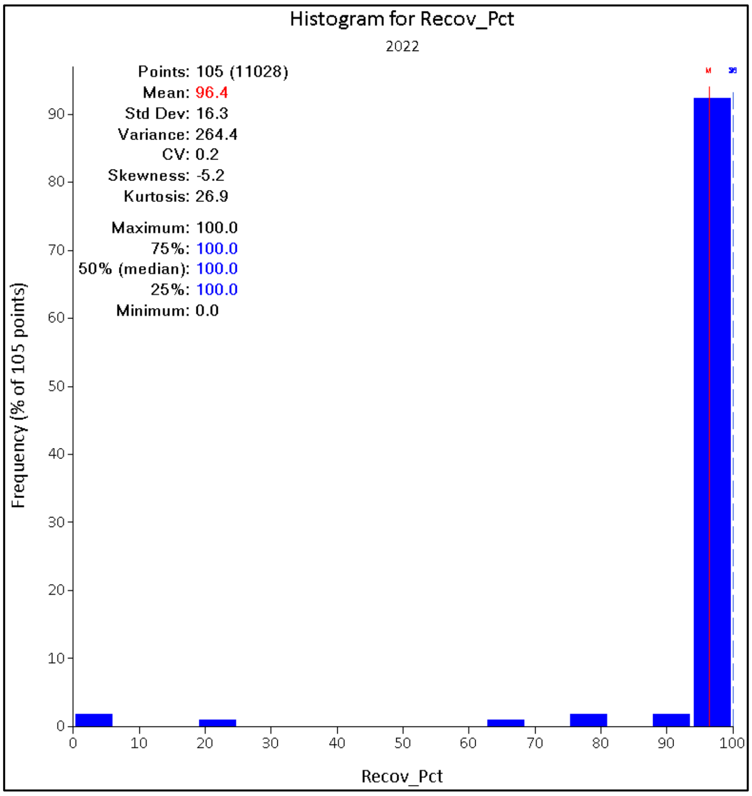
2020 Drilling:



2021 Drilling:



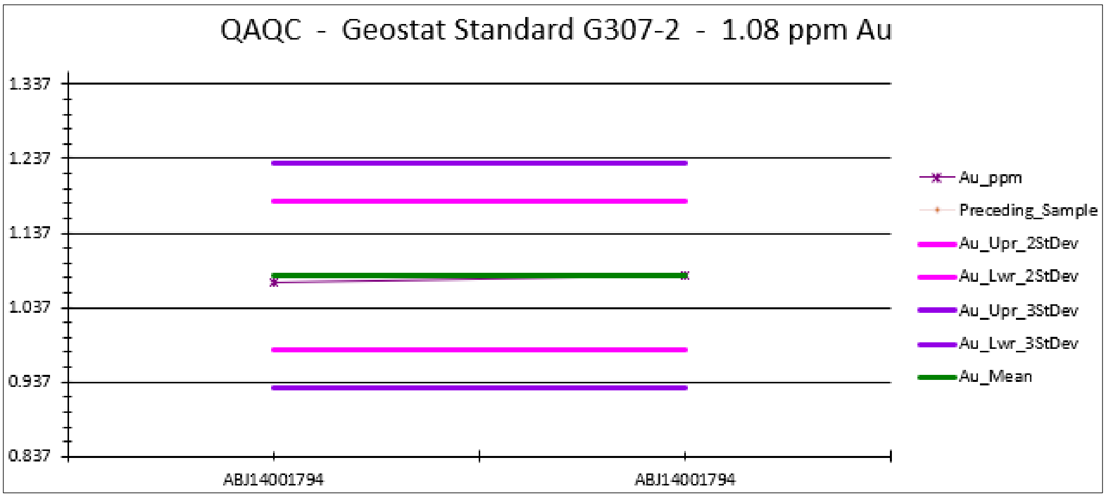
2022 Drilling:

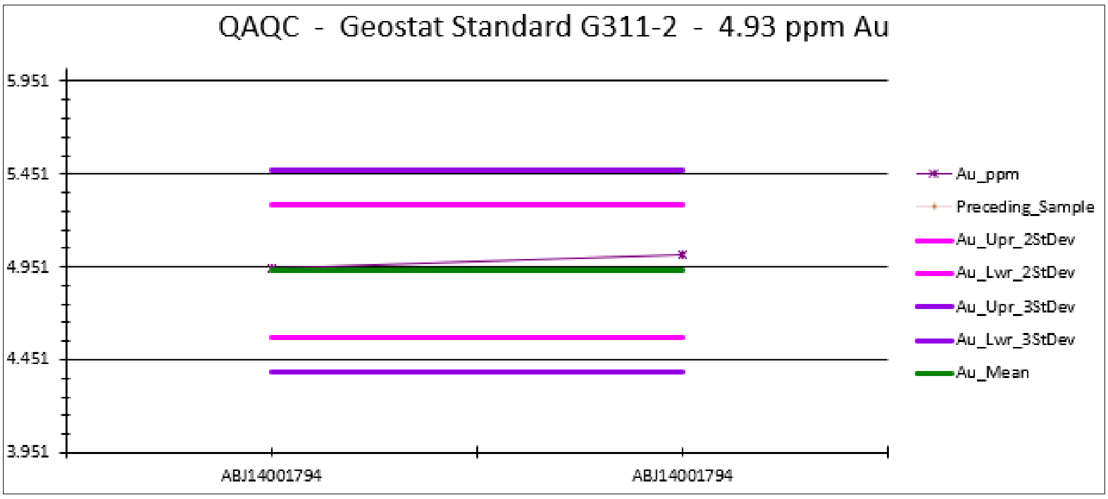


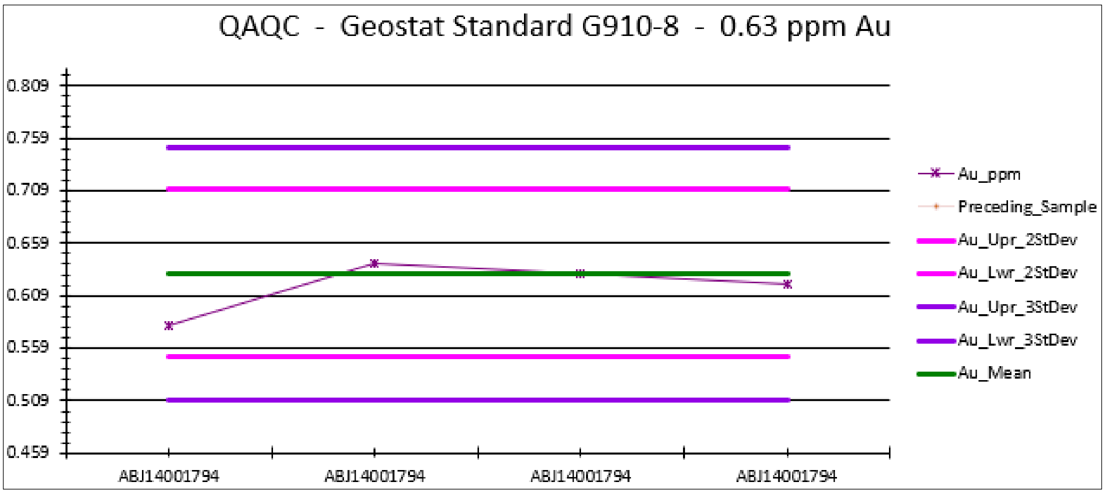
APPENDIX
C QAQC PLOTS

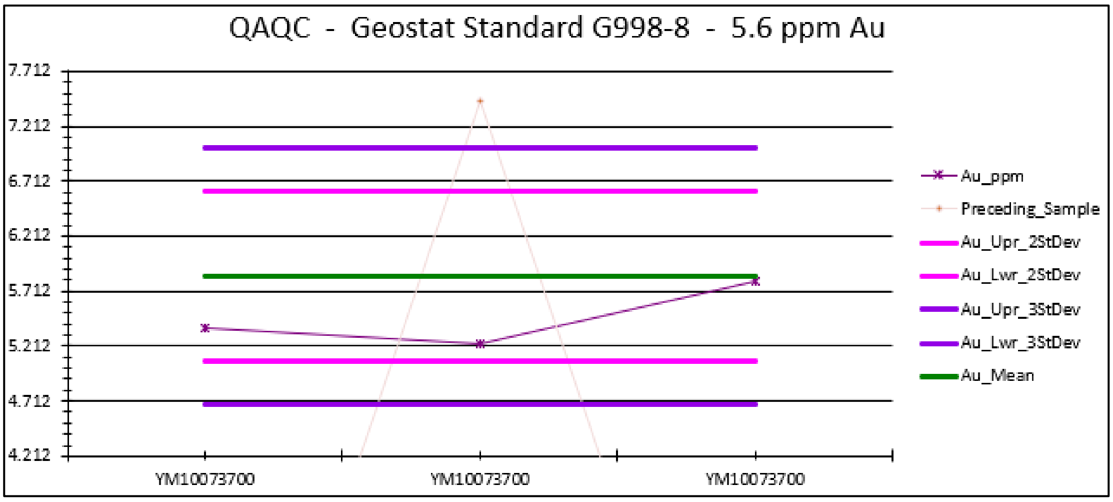
CRM Plots:

2010 CRM Plots - DD:

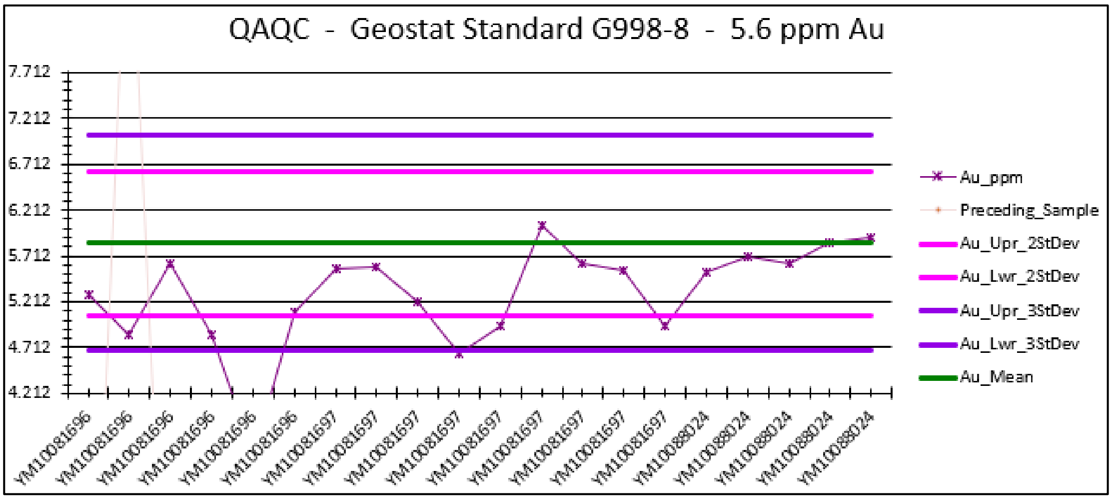




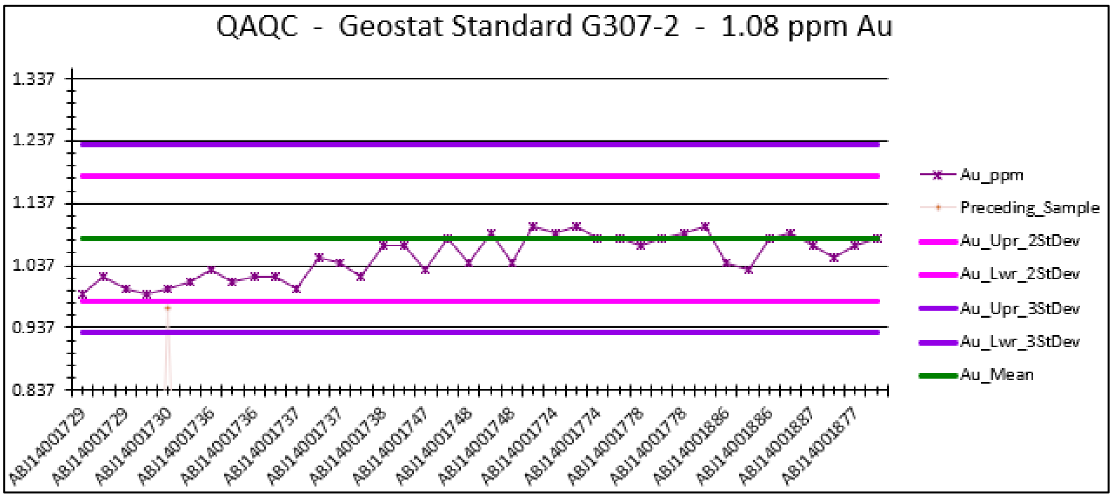


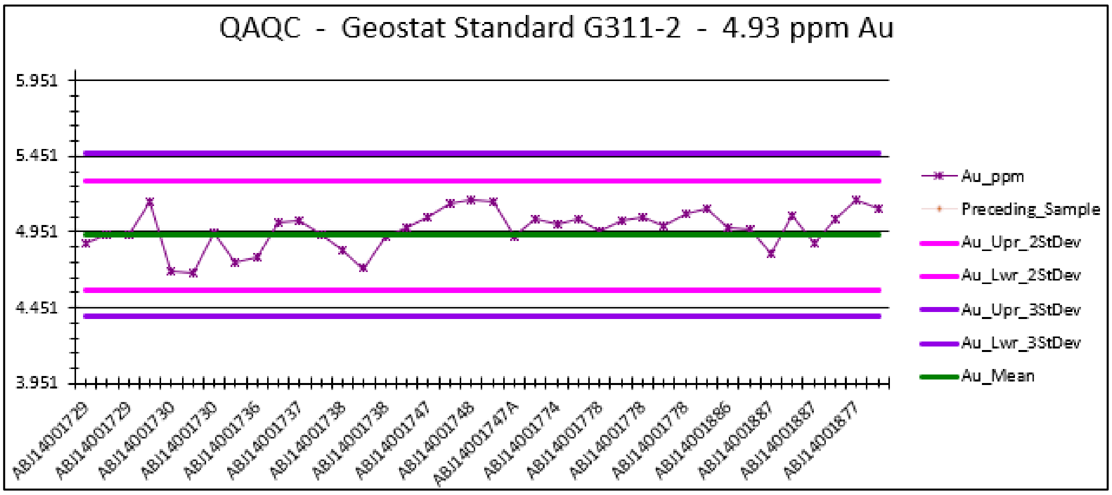


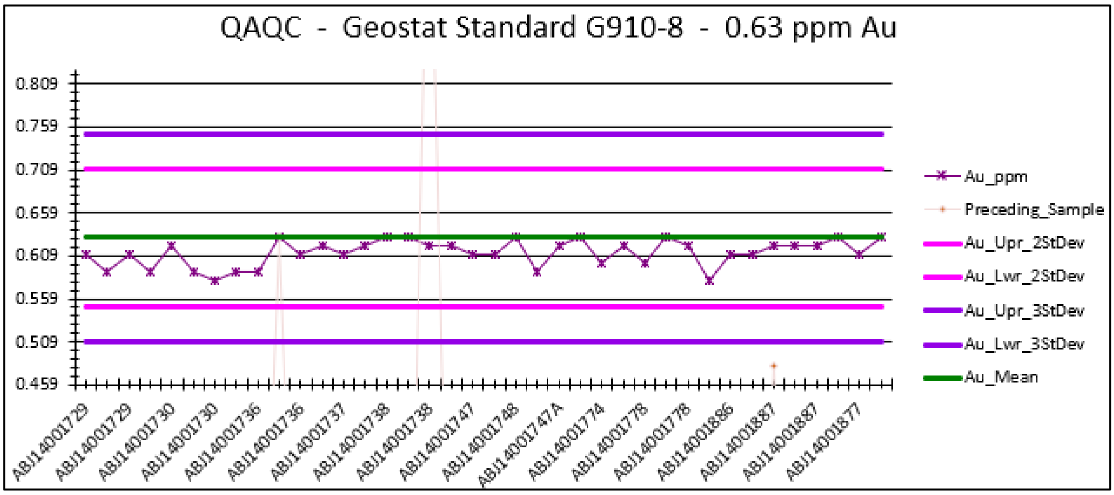
2010 CRM Plots - RC:



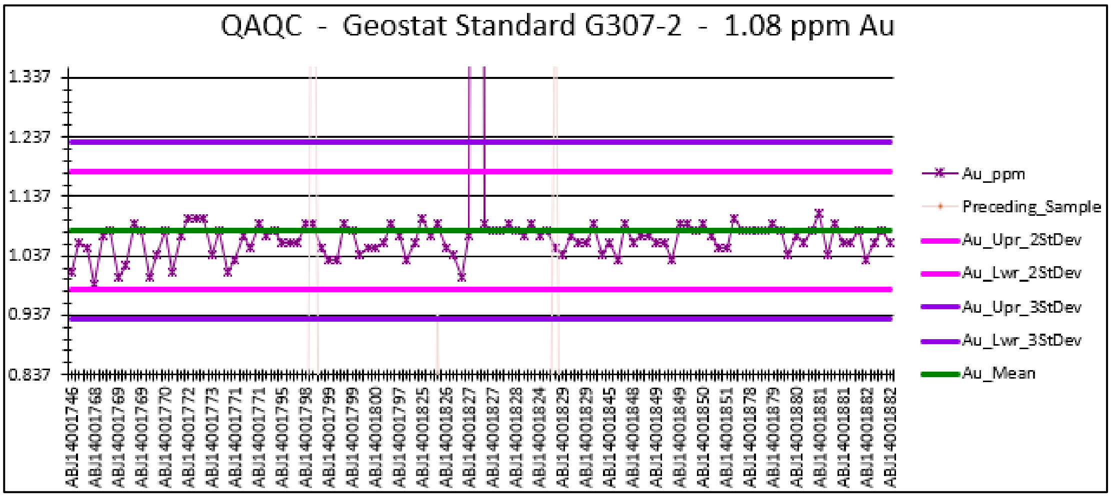
2014 CRM Plots – DD:

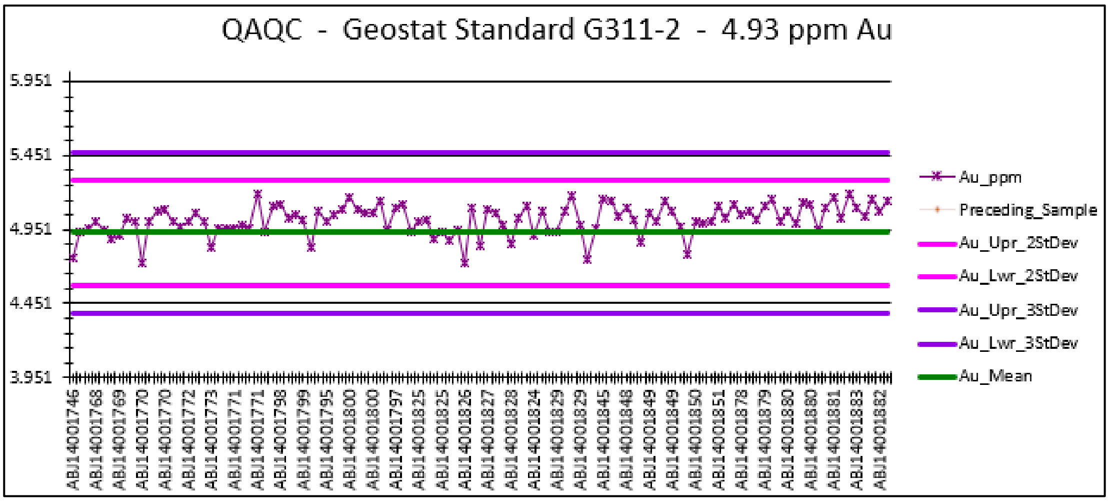


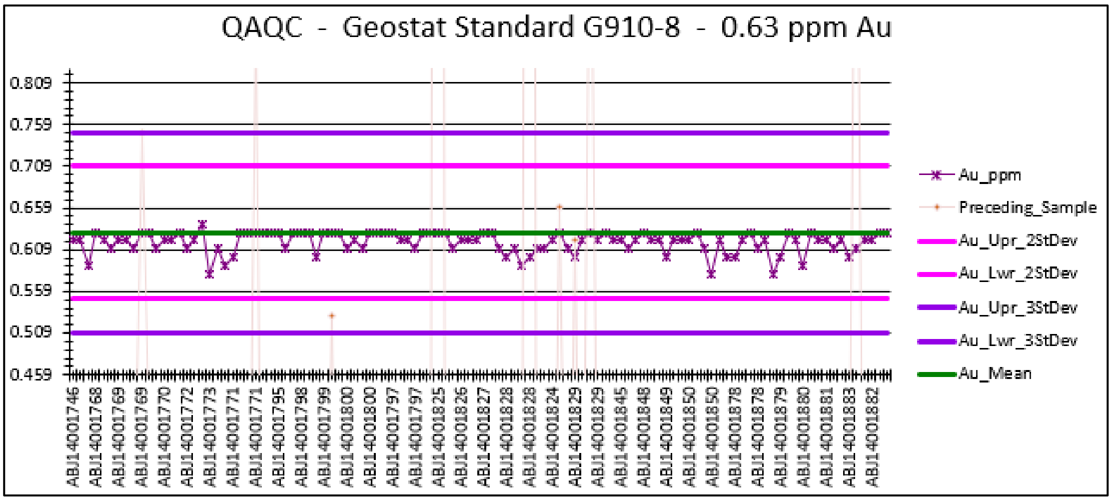




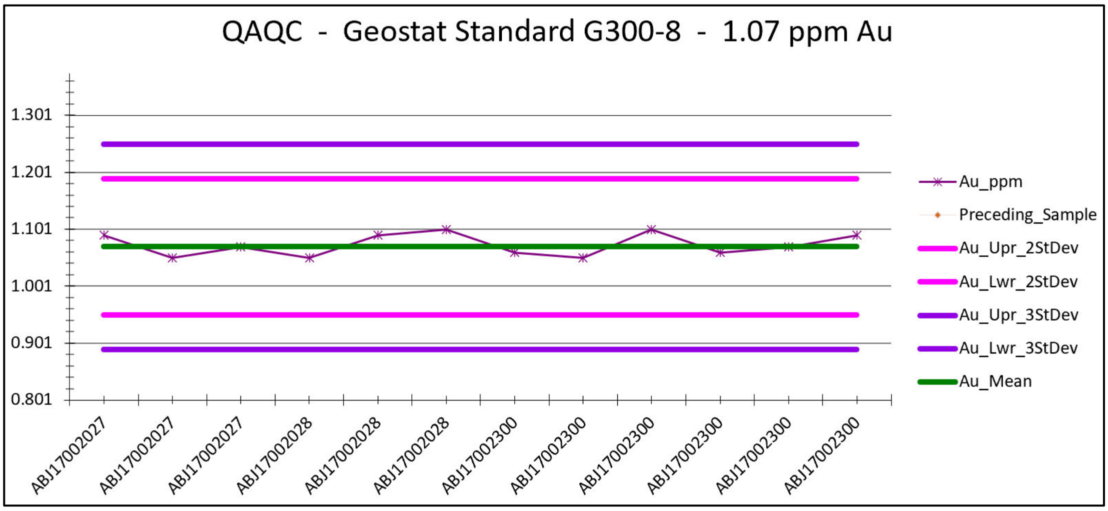
2014 CRM Plots – RC:

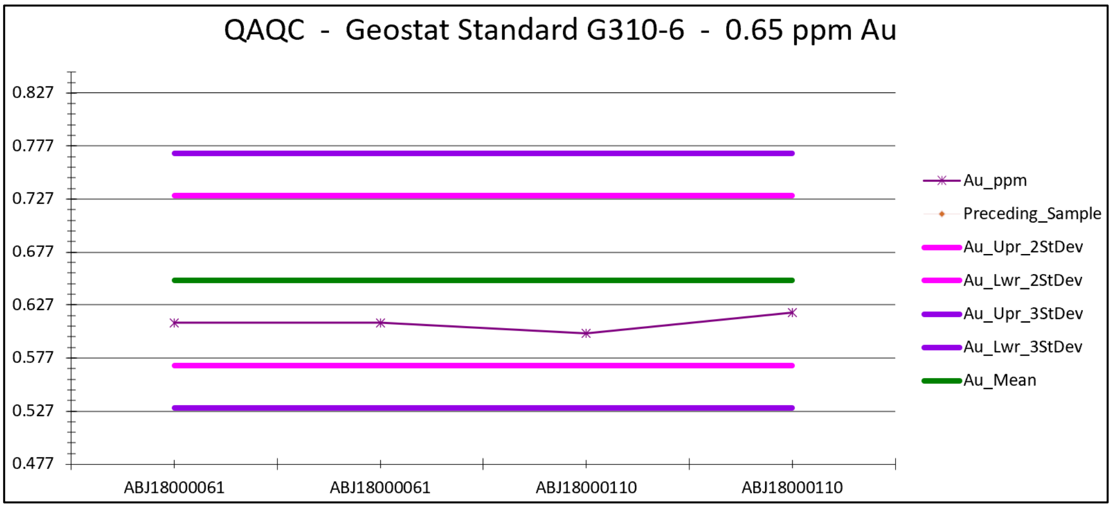


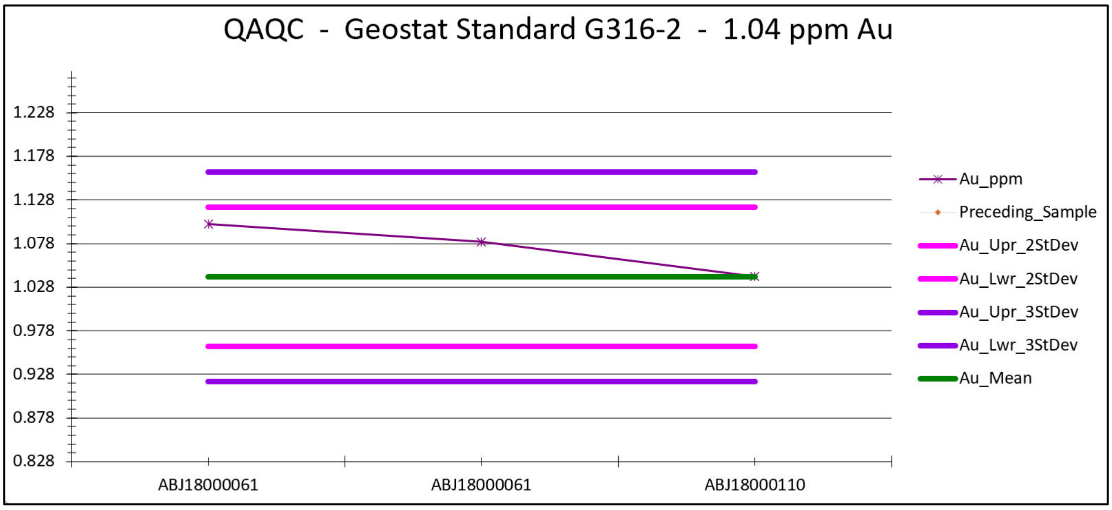


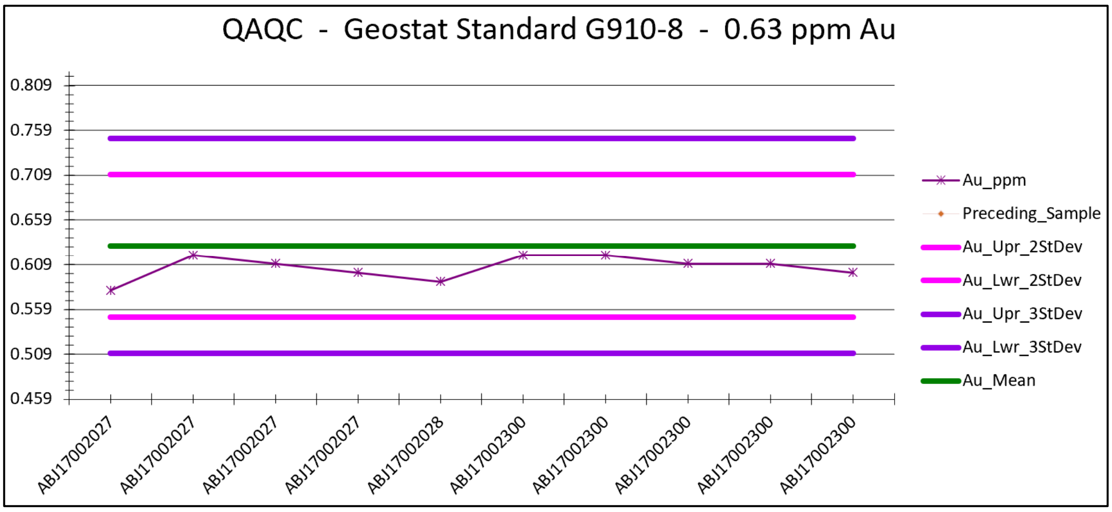


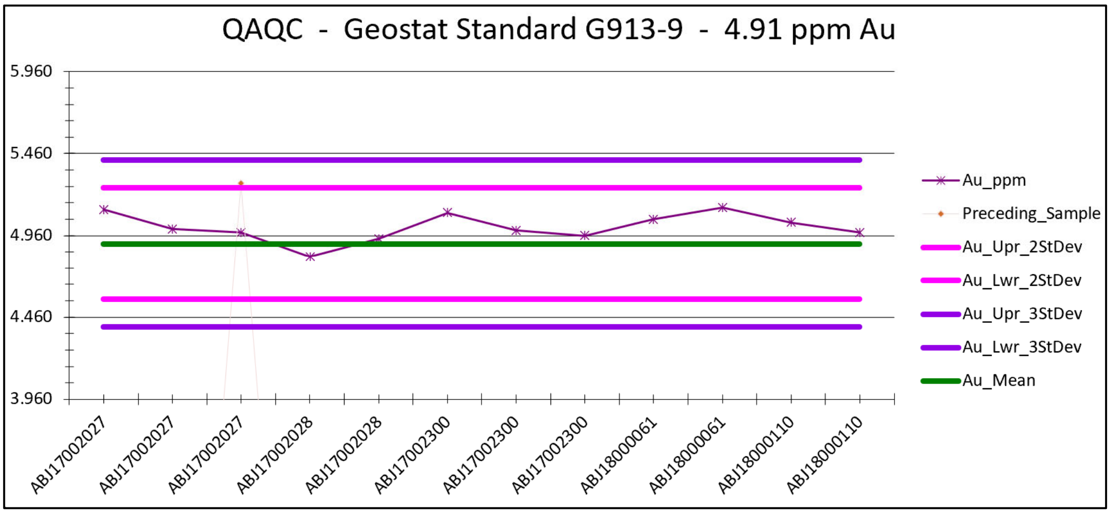
2017 CRM Plots – DD:



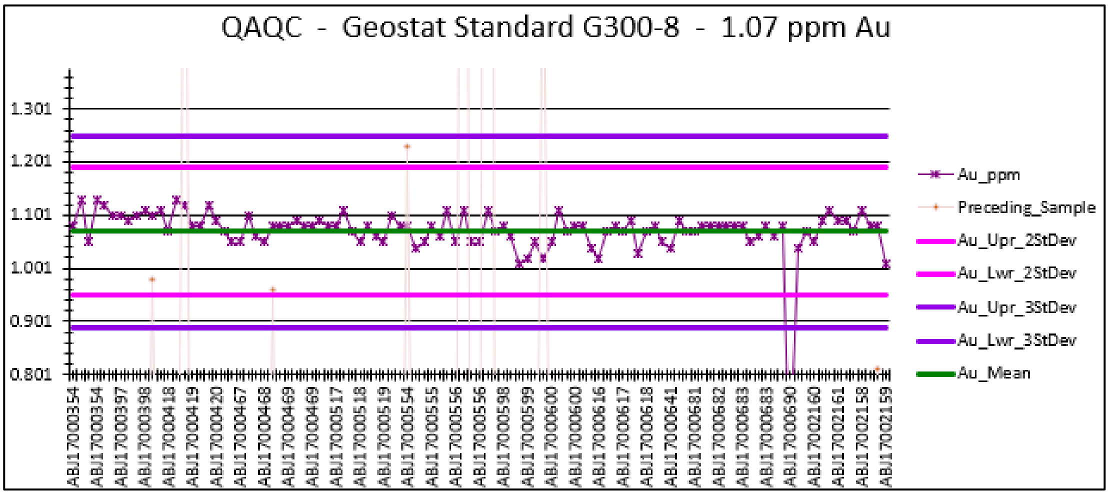

QAQC - Geostat Standard G310-6 - 0.65 ppm Au
0.827
0.777
0.727
0.677
0.627
0.577
0.527
0.477
ABJ18000061
ABJ18000061
ABJ18000110
ABJ18000110
Au_ppm
Preceding_Sample
Au_Upr_2StDev
Au_Lwr_2StDev
Au_Upr_3StDev
Au_Lwr_3StDev
Au_Mean

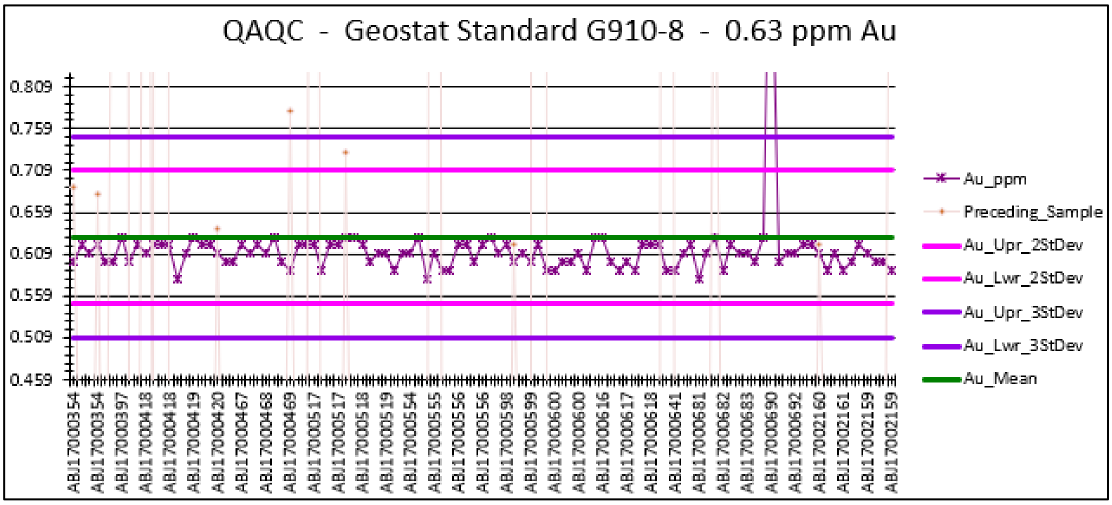

QAQC - Geostat Standard G316-2 - 1.04 ppm Au
1.228
1.178
1.128
1.078
1.028
0.978
0.928
0.878
0.828
ABJ18000061
ABJ18000061
ABJ18000110
Au_ppm
Preceding_Sample
Au_Upr_2StDev
Au_Lwr_2StDev
Au_Upr_3StDev
Au_Lwr_3StDev
Au_Mean

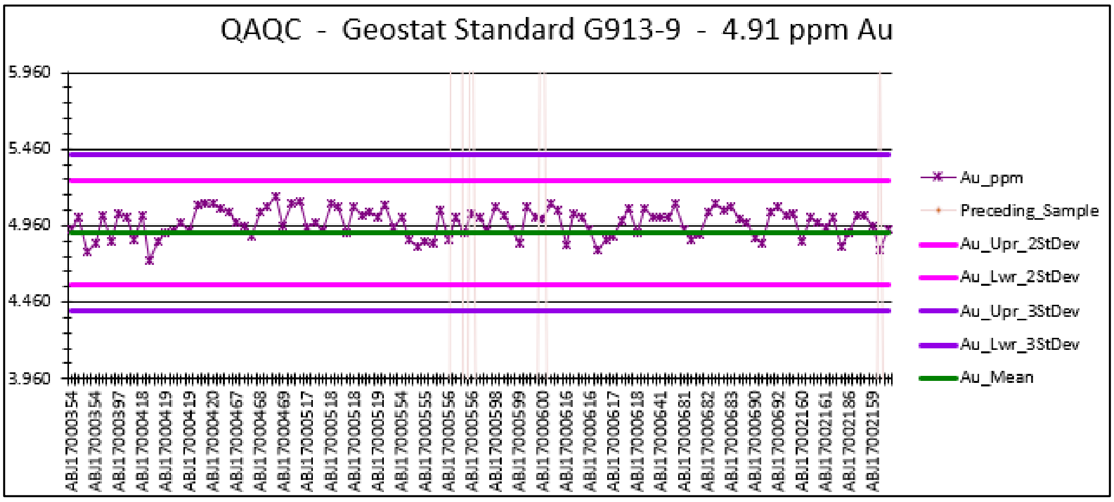

QAQC - Geostat Standard G910-8 - 0.63 ppm Au
0.809
0.759
0.709
0.659
0.609
0.559
0.509
0.459
ABJ17002027
ABJ17002027
ABJ17002027
ABJ17002027
ABJ17002028
ABJ17002300
ABJ17002300
ABJ17002300
ABJ17002300
ABJ17002300
Au_ppm
Preceding_Sample
Au_Upr_2StDev
Au_Lwr_2StDev
Au_Upr_3StDev
Au_Lwr_3StDev
Au_Mean



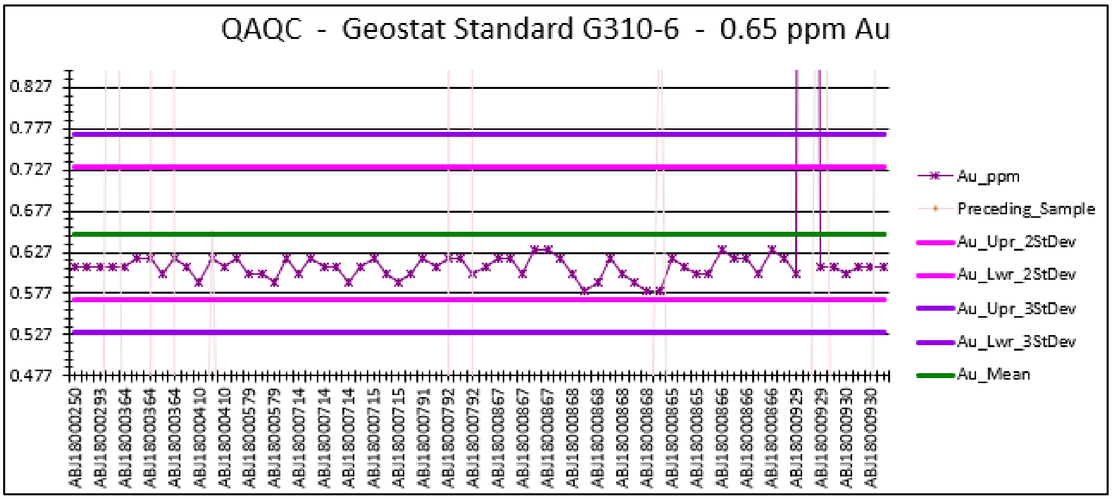
2017 CRM Plots – RC:

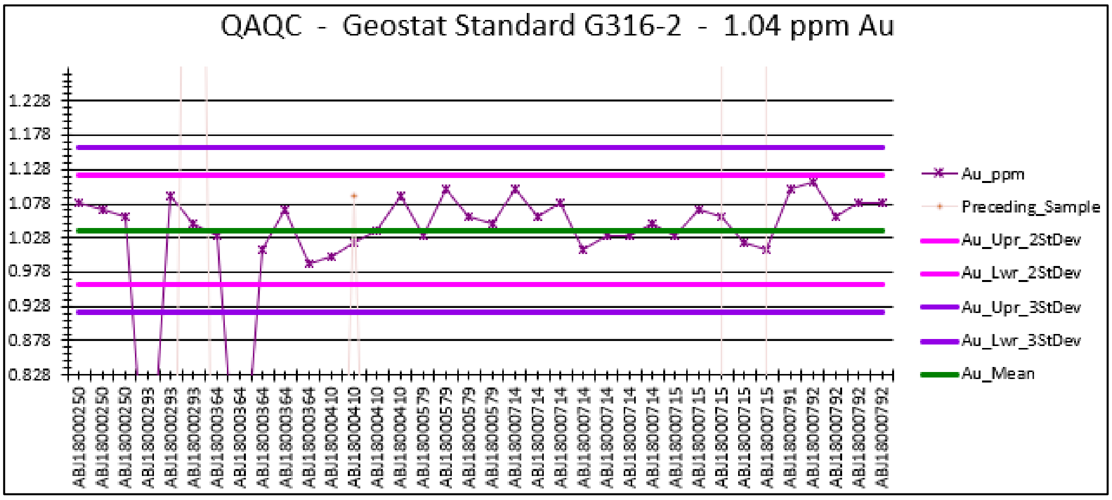


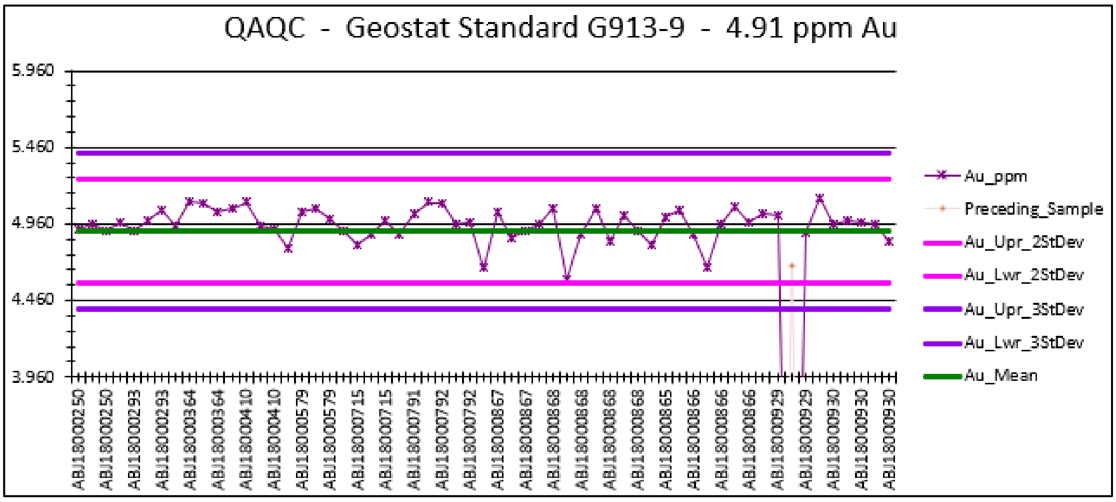




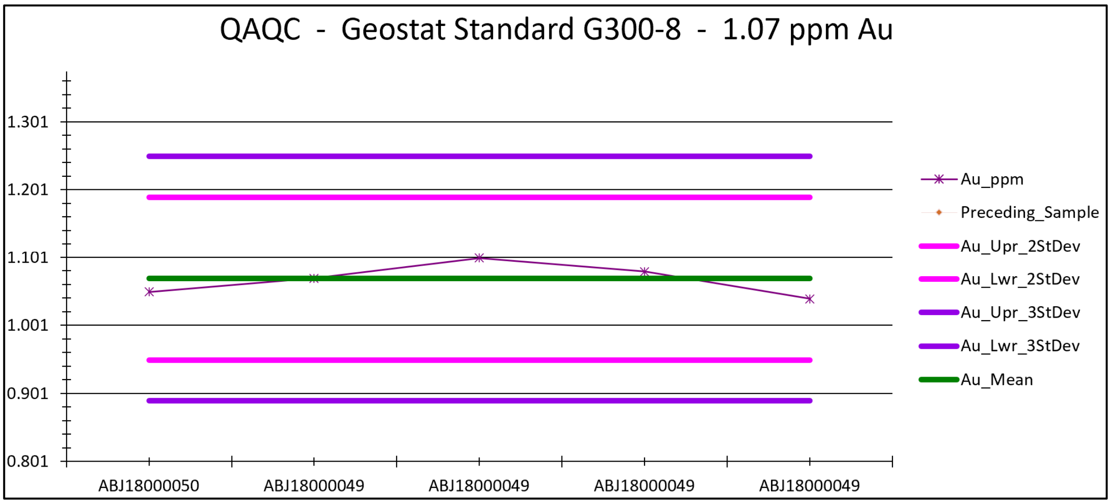
2018 CRM Plots – DD:

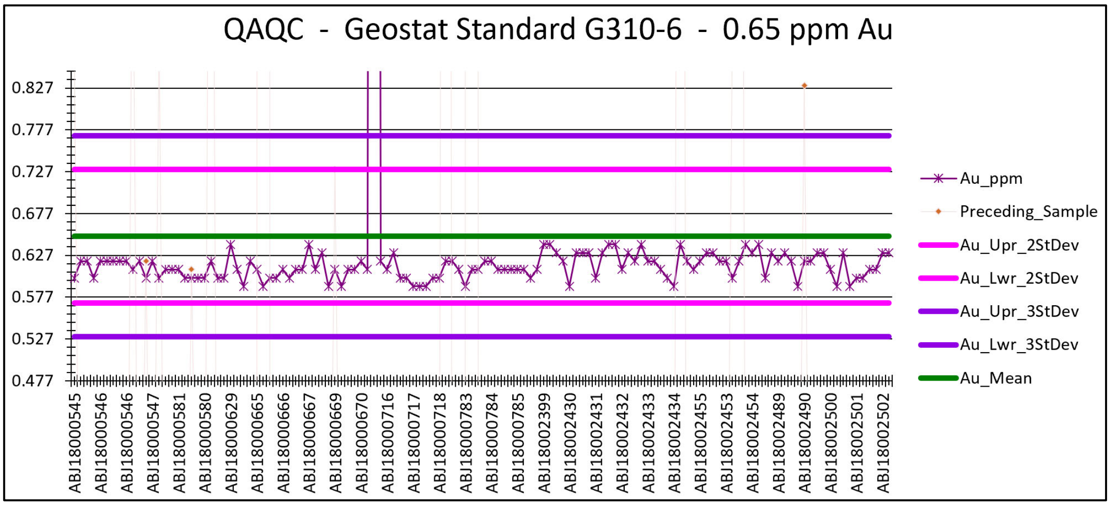


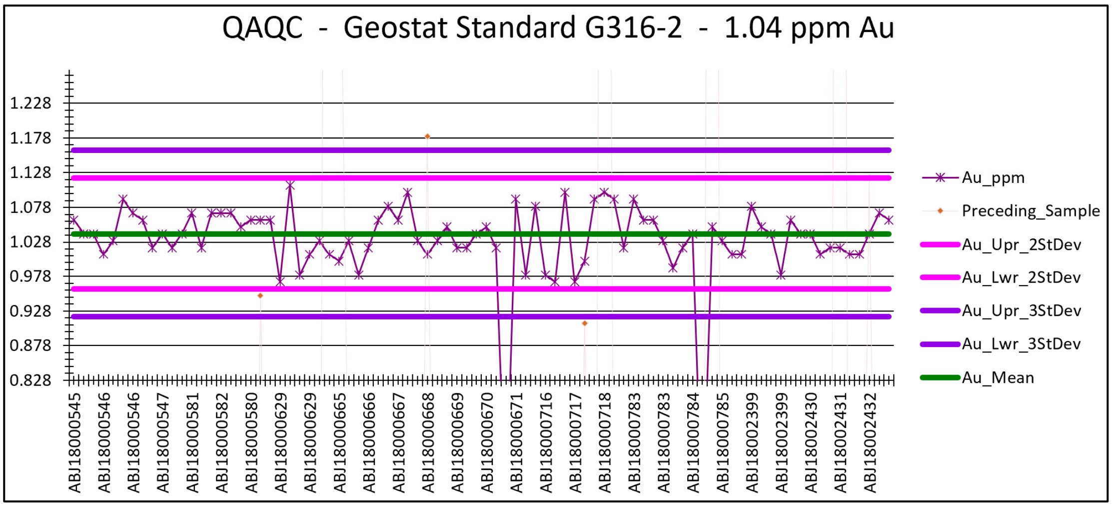




2018 CRM Plots – RC:






QAQC - Geostat Standard G316-2 - 1.04 ppm Au
1.228
1.178
1.128
1.078
1.028
0.978
0.928
0.878
0.828
ABJ18000545
ABJ18000546
ABJ18000546
ABJ18000547
ABJ18000581
ABJ18000582
ABJ18000580
ABJ18000629
ABJ18000629
ABJ18000665
ABJ18000666
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ABJ18000668
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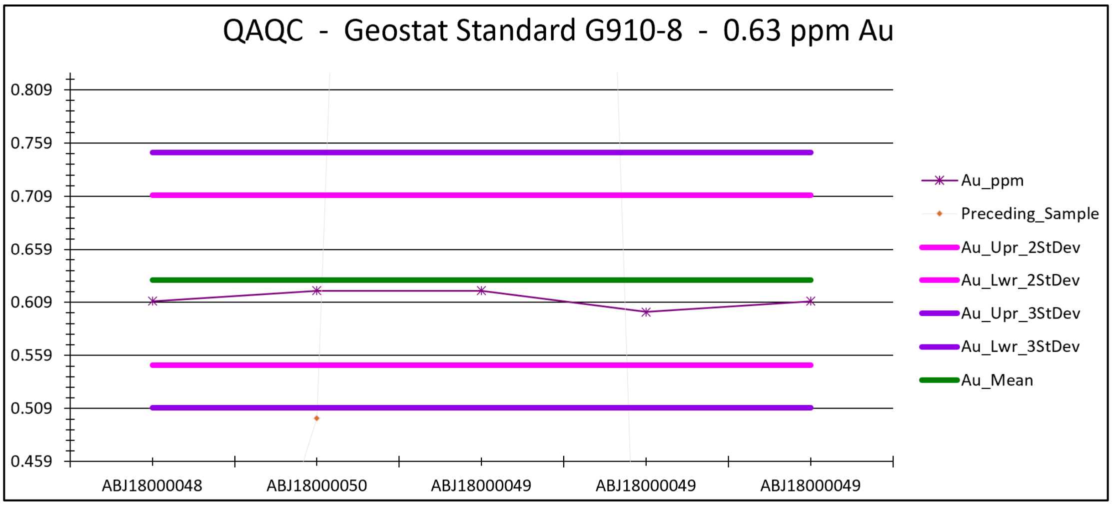
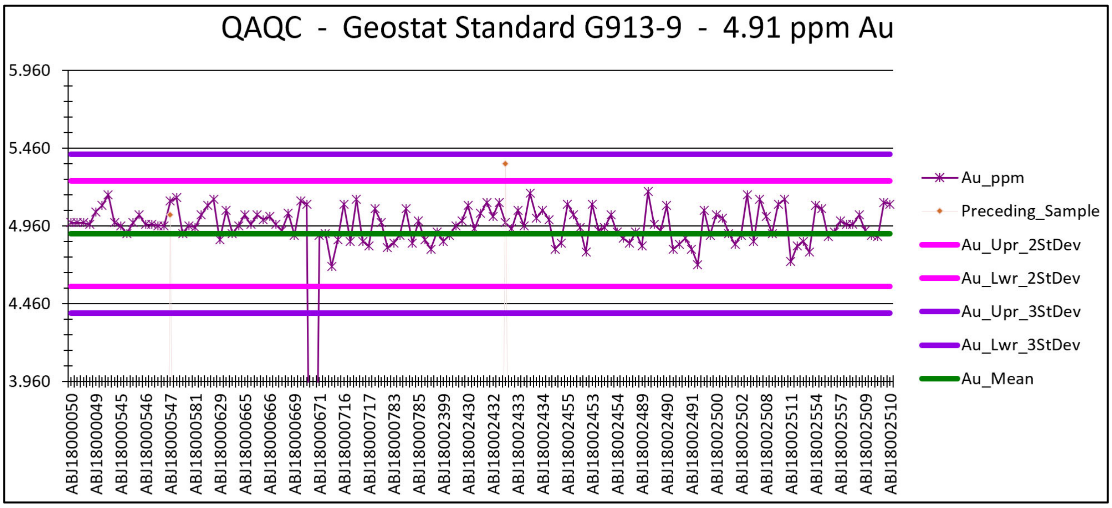
ABJ18000717
ABJ18000718
ABJ18000783
ABJ18000783
ABJ18000784
ABJ18000785
ABJ18002399
ABJ18002399
ABJ18002430
ABJ18002431
ABJ18002432
Au_ppm
Preceding_Sample
Au_Upr_2StDev
Au_Lwr_2StDev
Au_Upr_3StDev
Au_Lwr_3StDev
Au_Mean


QAQC - Geostat Standard G910-8 - 0.63 ppm Au
0.809
0.759
0.709
0.659
0.609
0.559
0.509
0.459
ABJ18000048
ABJ18000050
ABJ18000049
ABJ18000049
ABJ18000049
Au_ppm
Preceding_Sample
Au_Upr_2StDev
Au_Lwr_2StDev
Au_Upr_3StDev
Au_Lwr_3StDev
Au_Mean


QAQC - Geostat Standard G913-9 - 4.91 ppm Au
5.960
5.460
4.960
4.460
3.960
ABJ18000050
ABJ18000049
ABJ18000545
ABJ18000546
ABJ18000547
ABJ18000581
ABJ18000629
ABJ18000665
ABJ18000666
ABJ18000669
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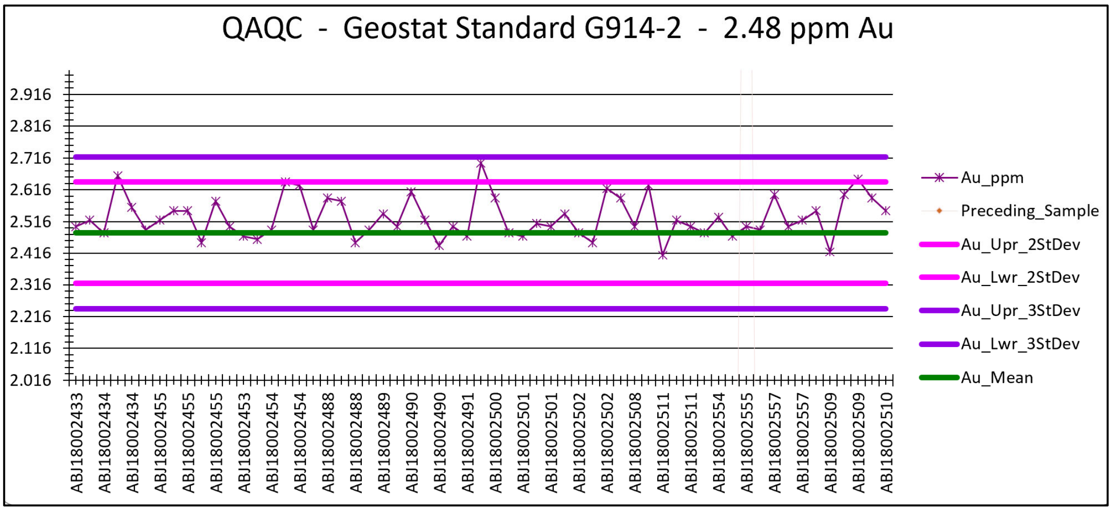
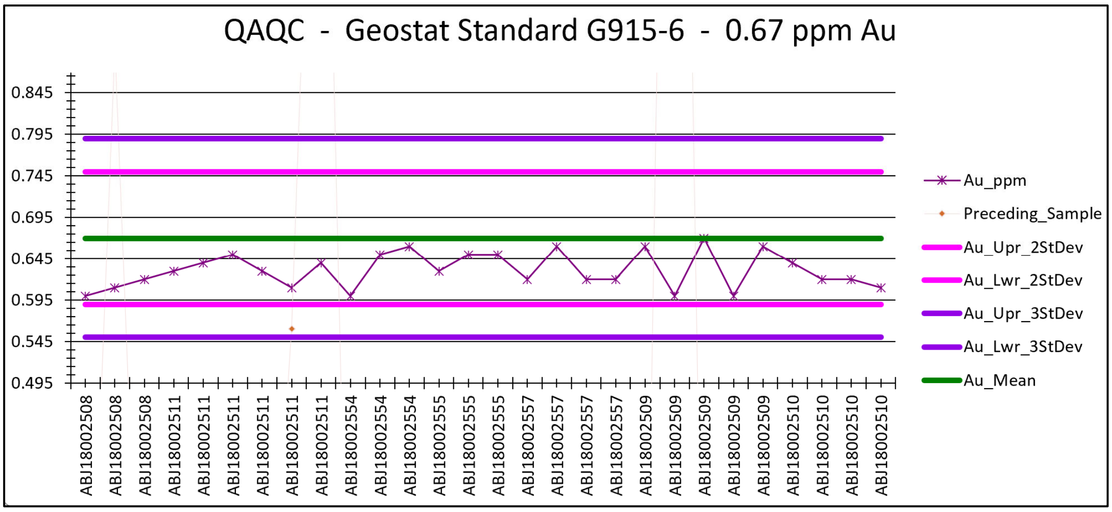
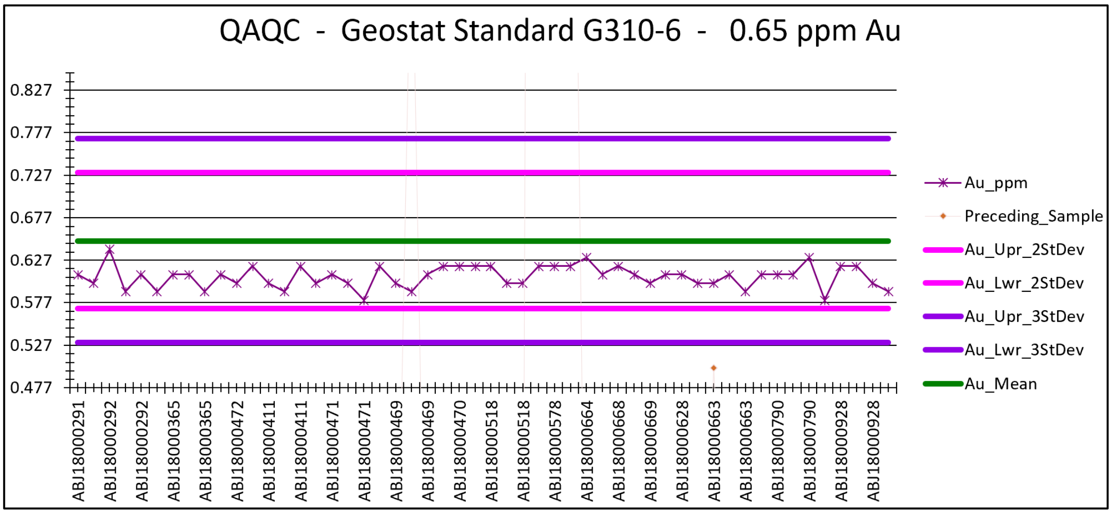
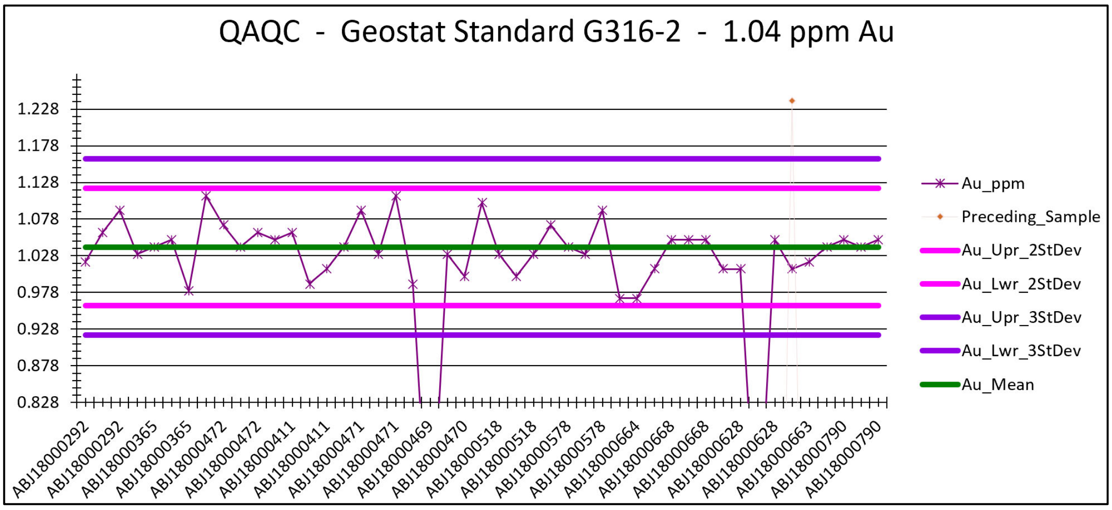
ABJ18002491
ABJ18002500
ABJ18002502
ABJ18002508
ABJ18002511
ABJ18002554
ABJ18002557
ABJ18002509
ABJ18002510
Au_ppm
Preceding_Sample
Au_Upr_2StDev
Au_Lwr_2StDev
Au_Upr_3StDev
Au_Lwr_3StDev
Au_Mean

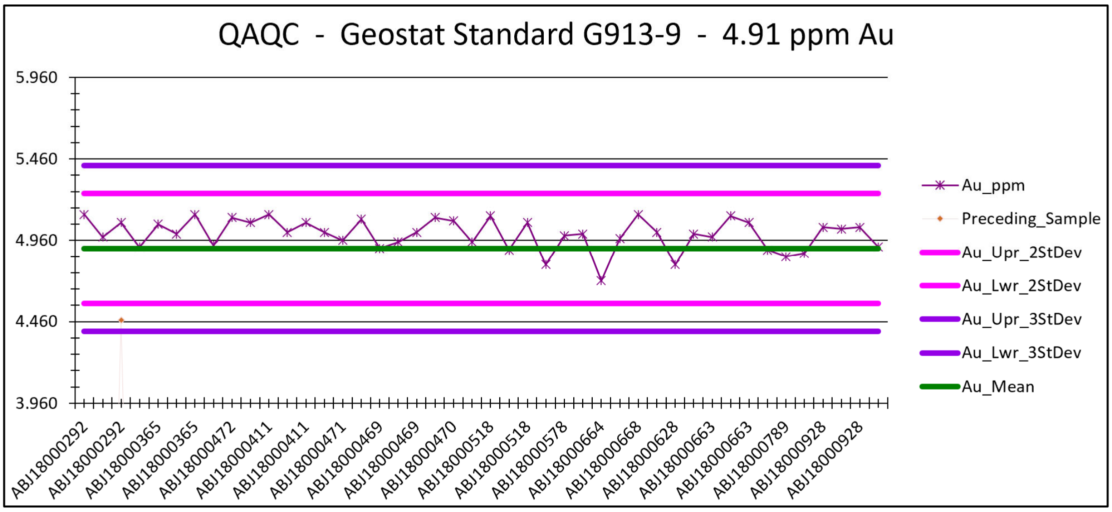




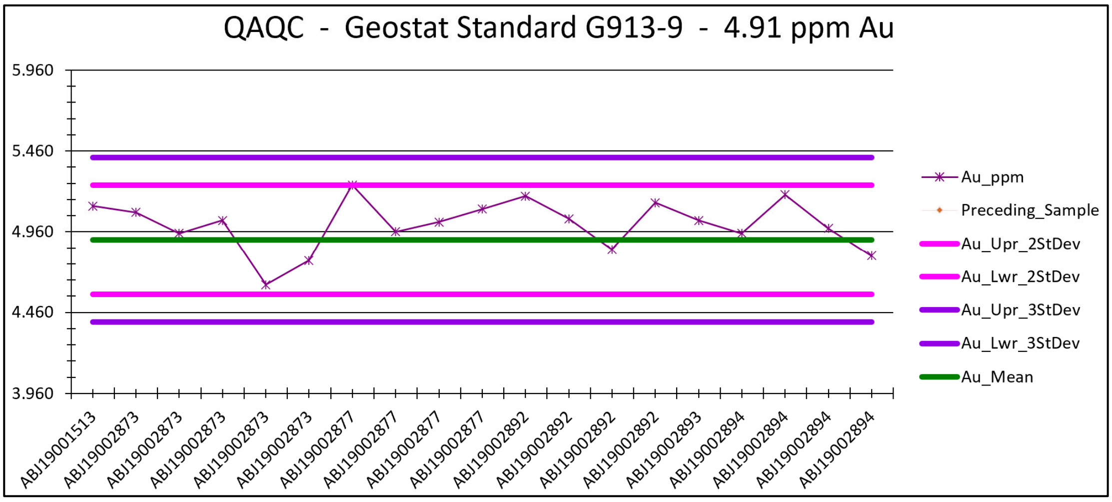
2018 CRM Plots – RC-DD:

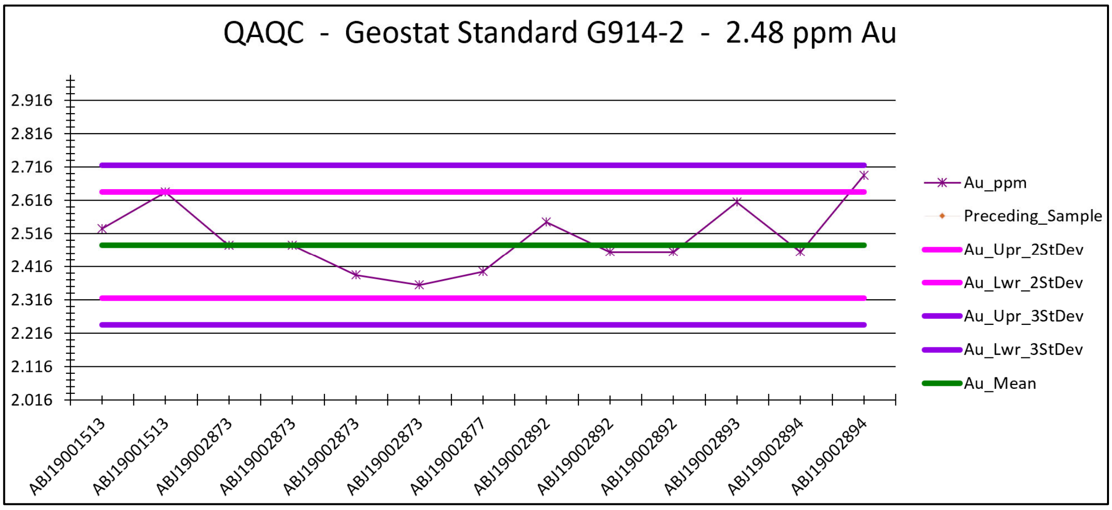


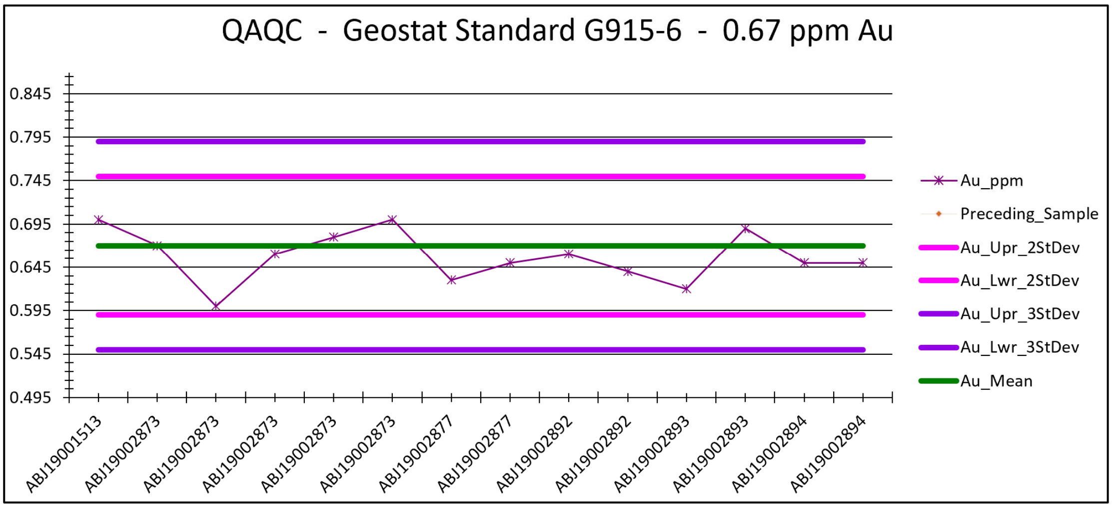




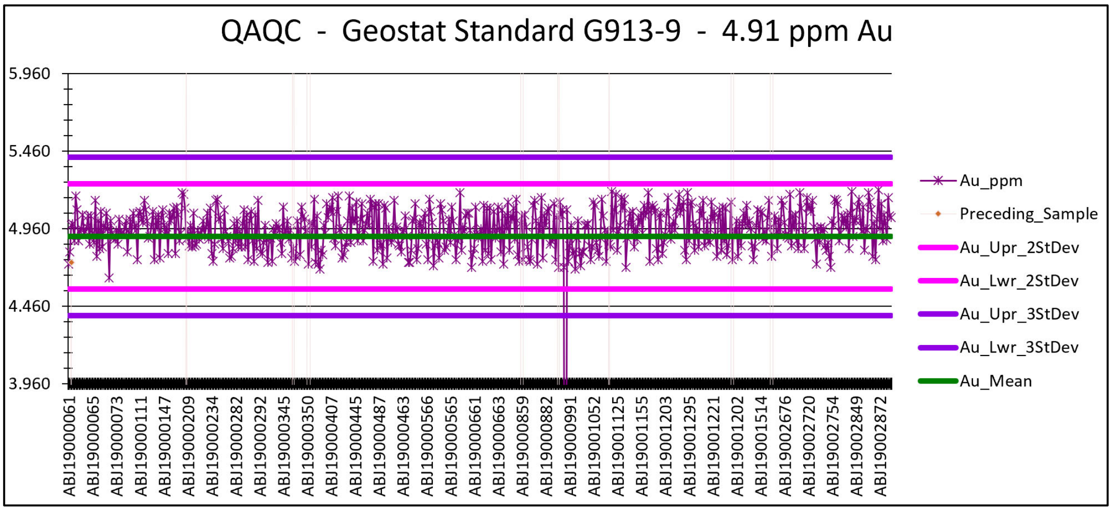
2019 CRM Plots – DD:

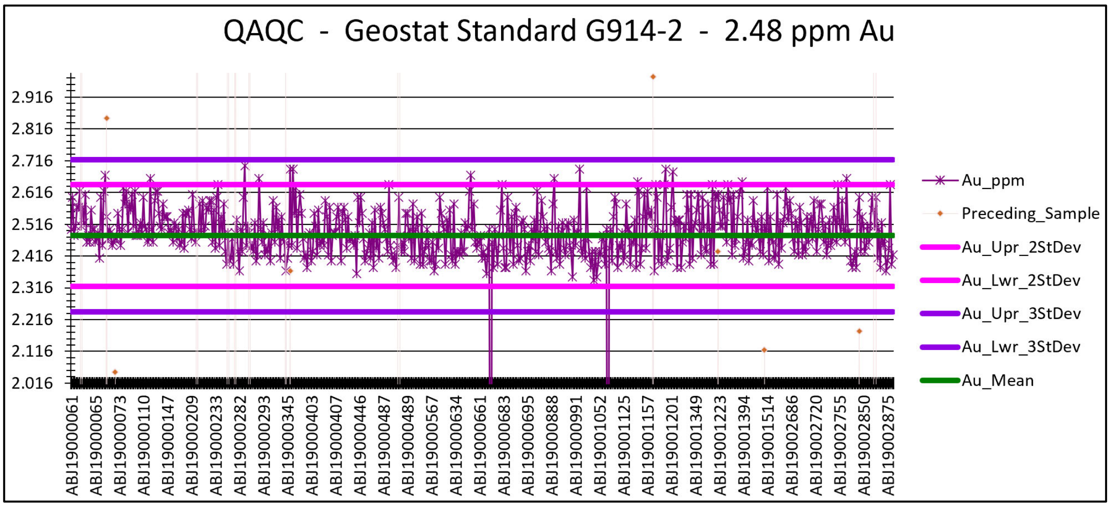


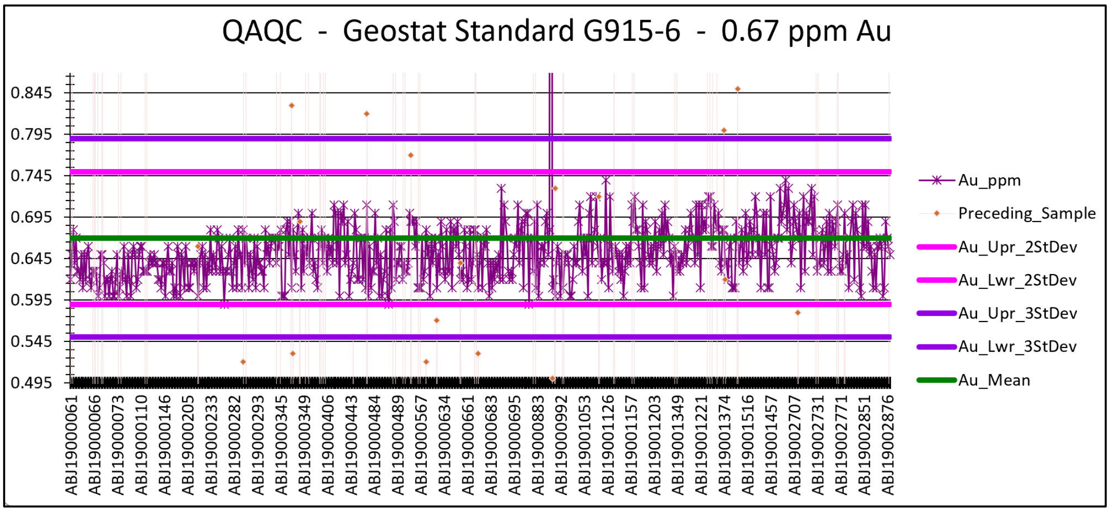




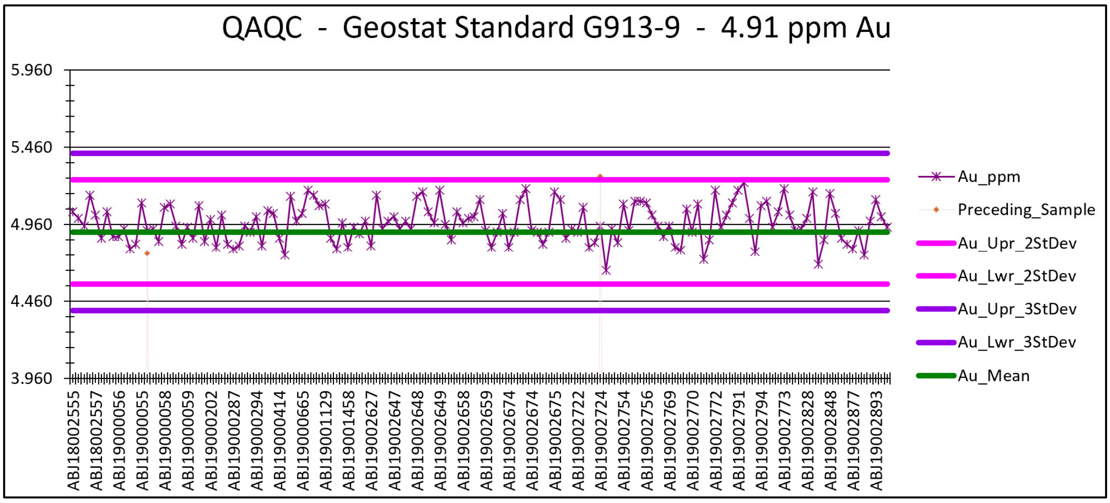
2019 CRM Plots – RC:

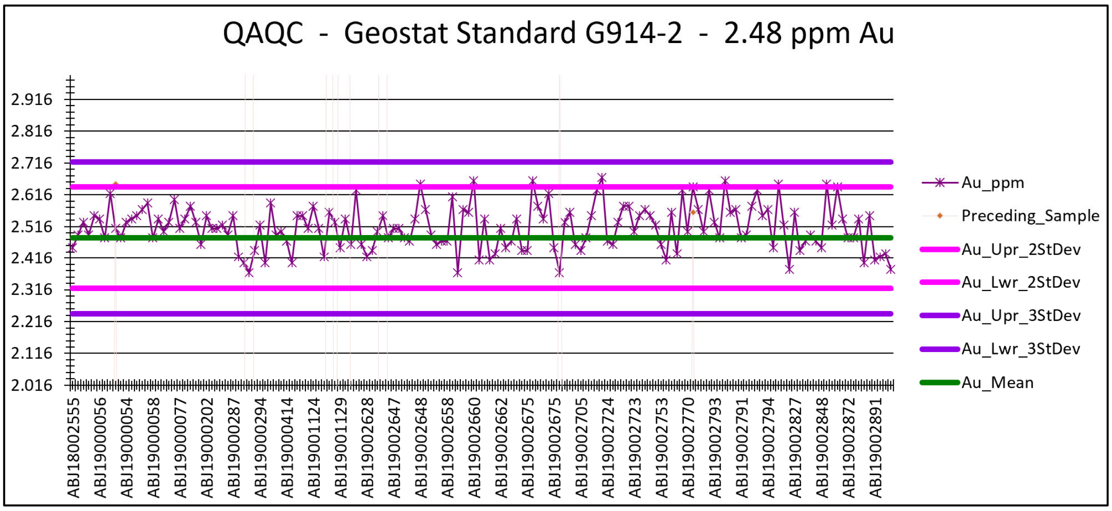


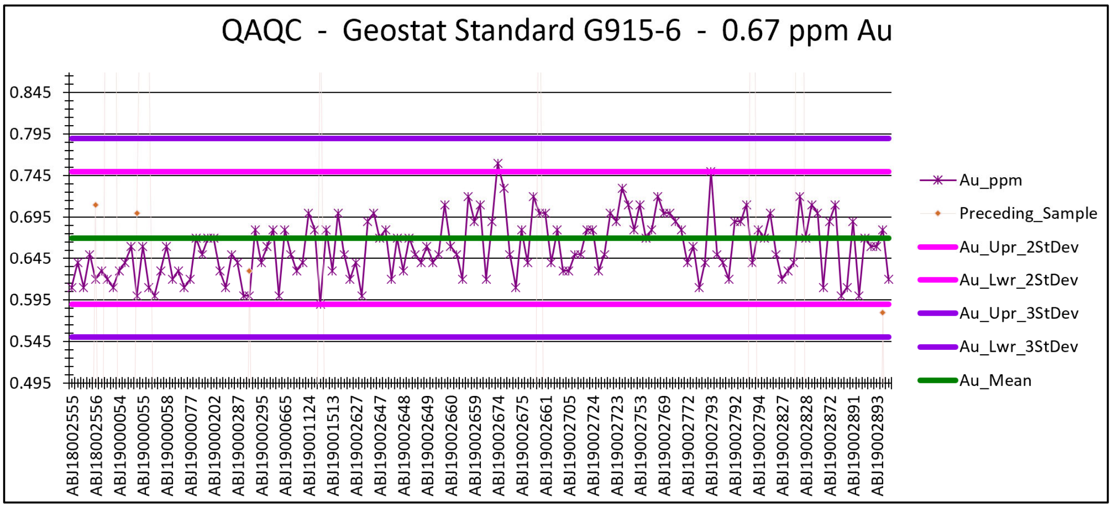




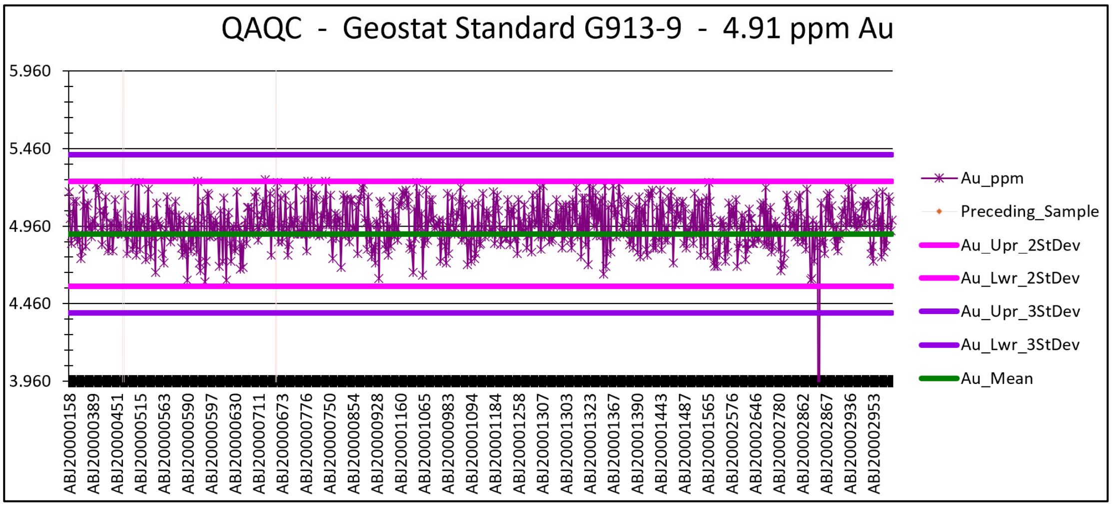
2019 CRM Plots – RC-DD:

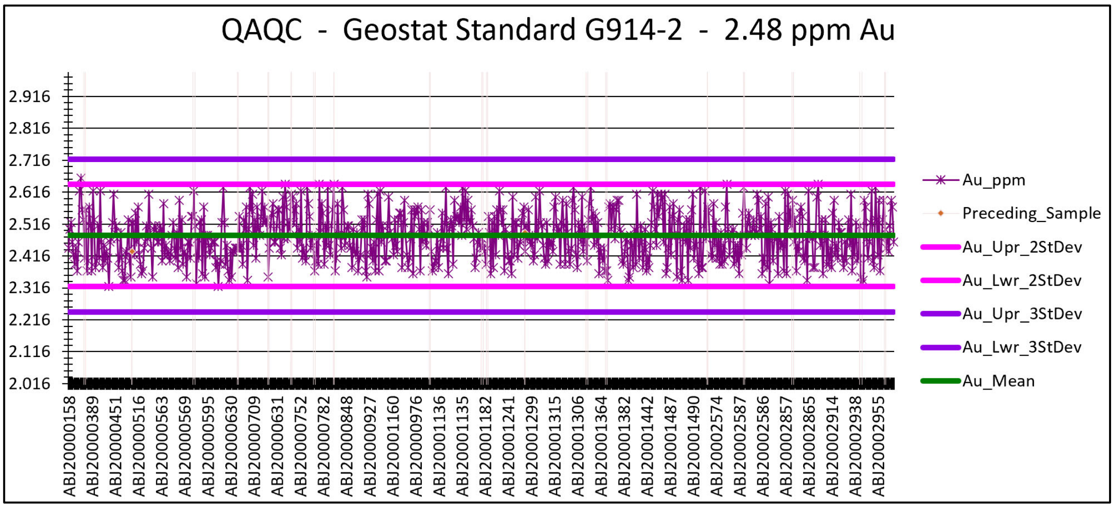


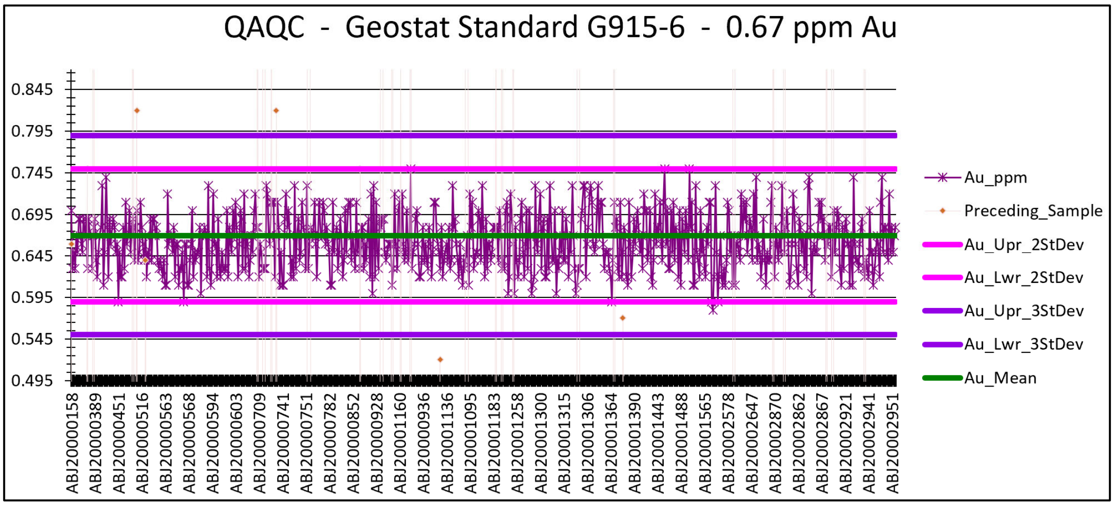




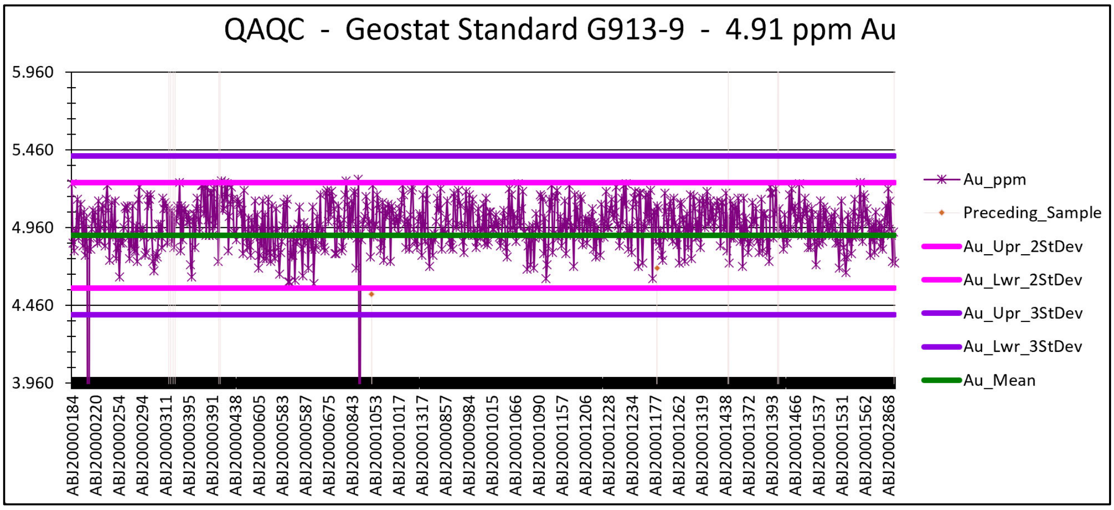
2020 CRM Plots – RC:

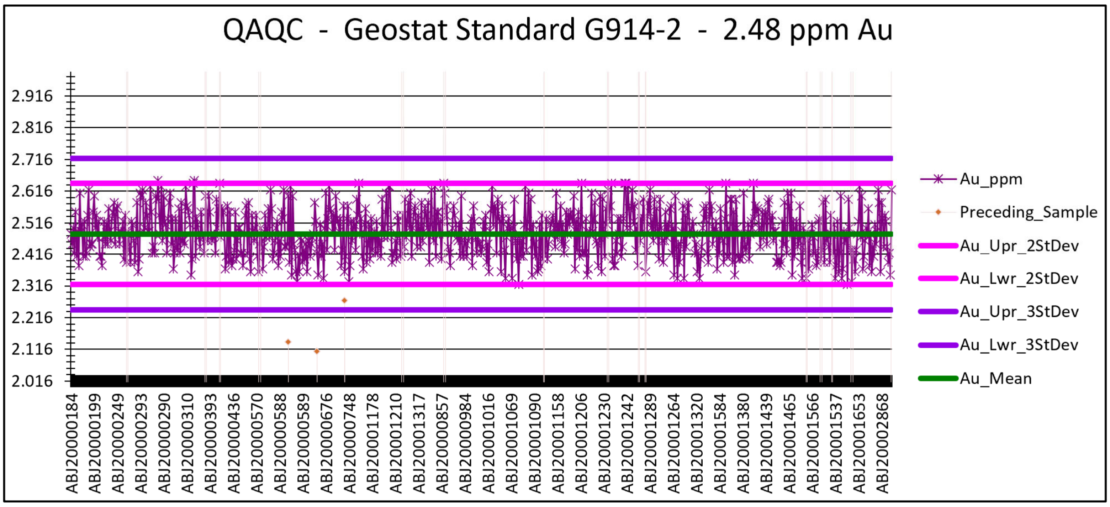


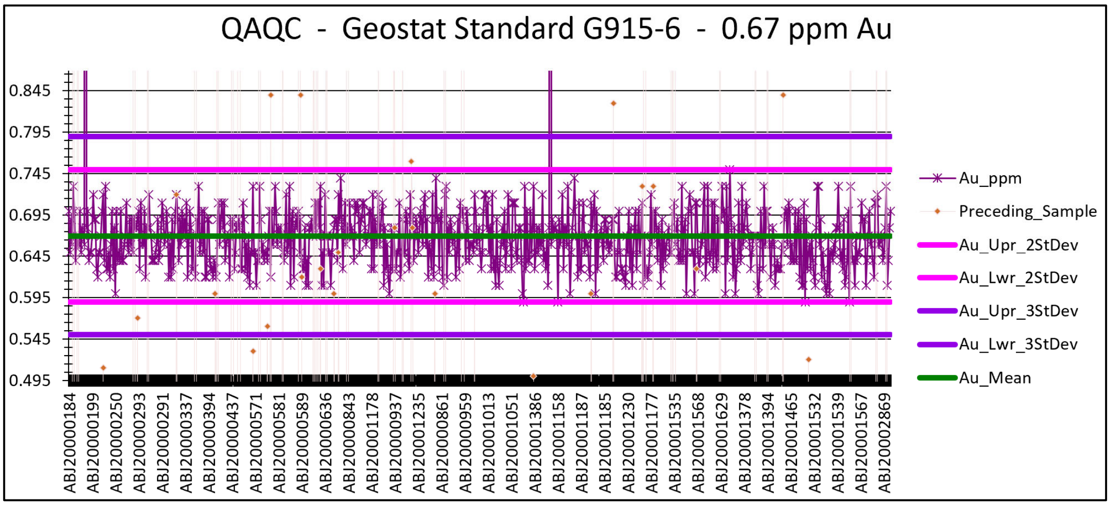




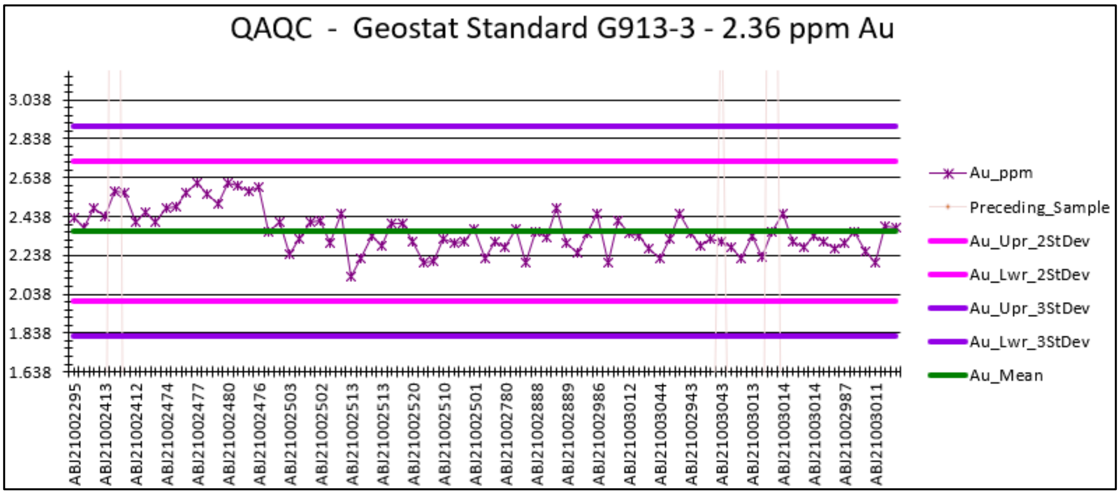
2020 CRM Plots – RC-DD:

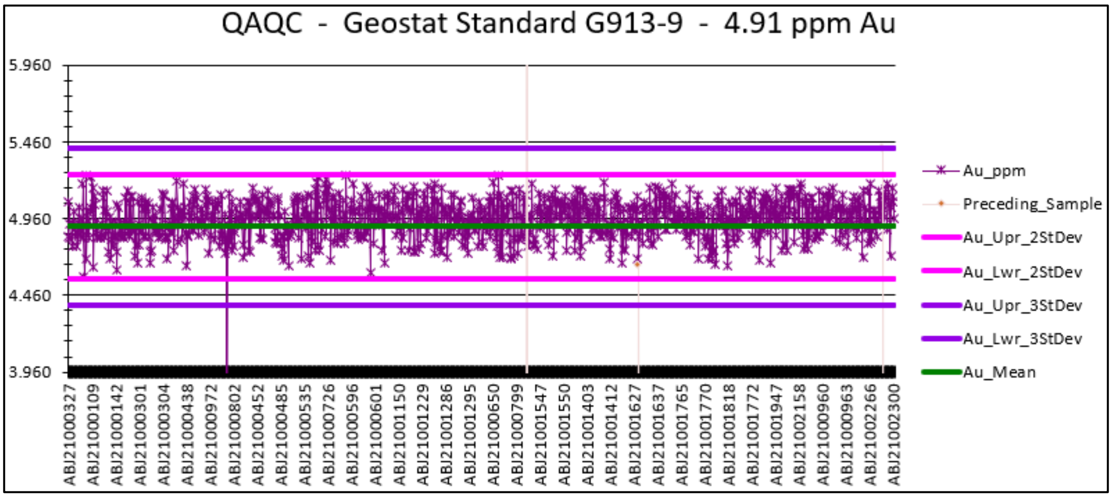


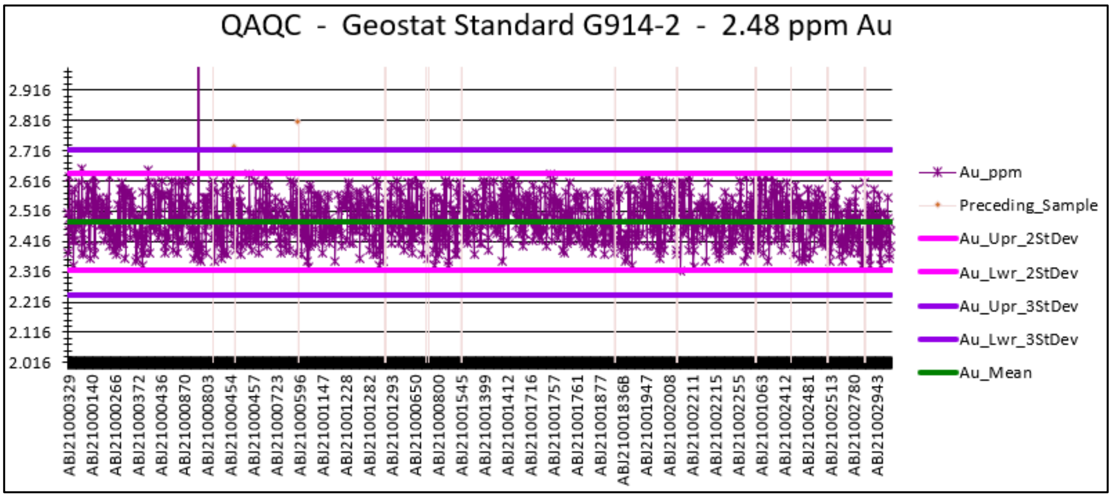


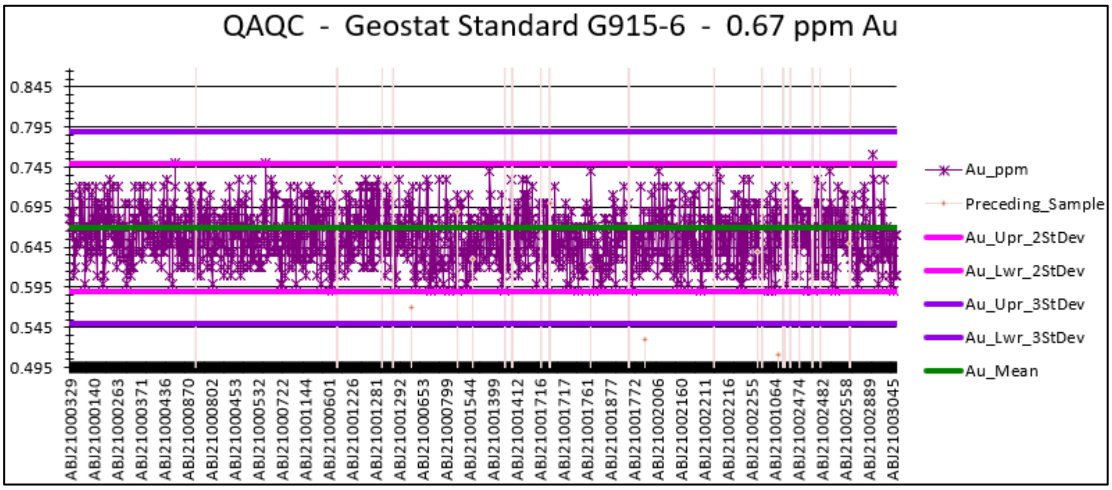


2021 CRM Plots – RC:

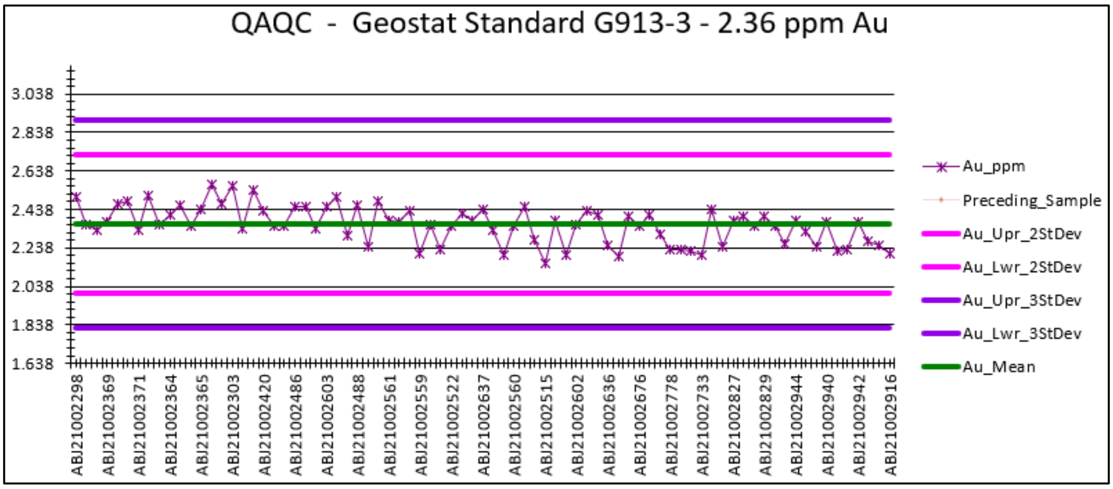


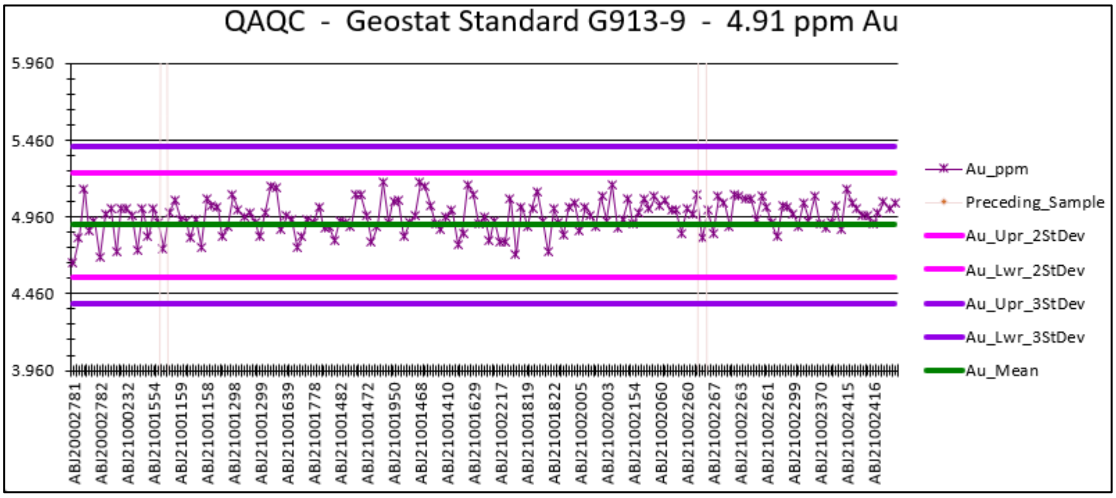

QAQC - Geostat Standard G913-9 - 4.91 ppm Au
5.960
5.460
4.960
4.460
3.960
Au_ppm
Preceding_Sample
Au_Upr_2StDev
Au_Lwr_2StDev
Au_Upr_3StDev
Au_Lwr_3StDev
Au_Mean

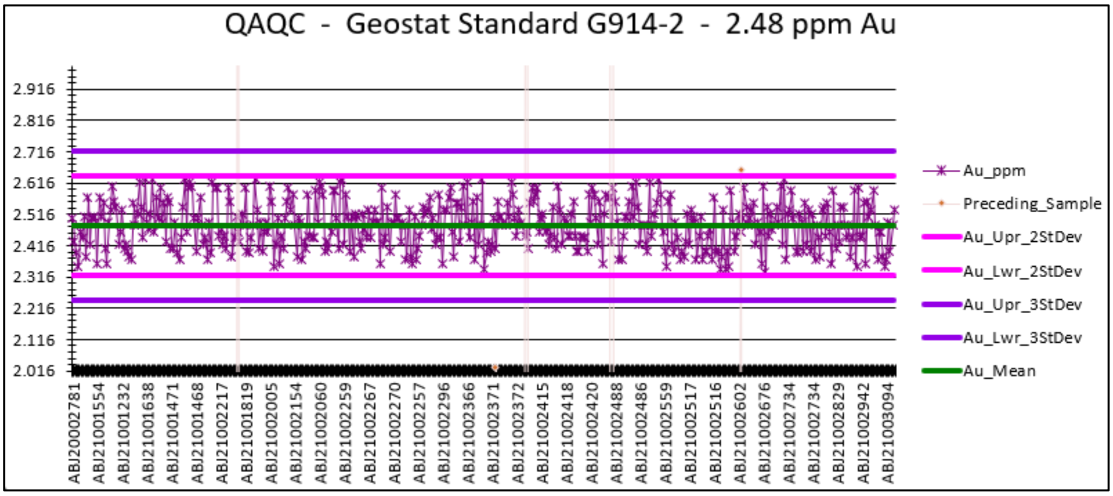

QAQC - Geostat Standard G914-2 - 2.48 ppm Au
2.916
2.816
2.716
2.616
2.516
2.416
2.316
2.216
2.116
2.016
Au_ppm
Preceding_Sample
Au_Upr_2StDev
Au_Lwr_2StDev
Au_Upr_3StDev
Au_Lwr_3StDev
Au_Mean

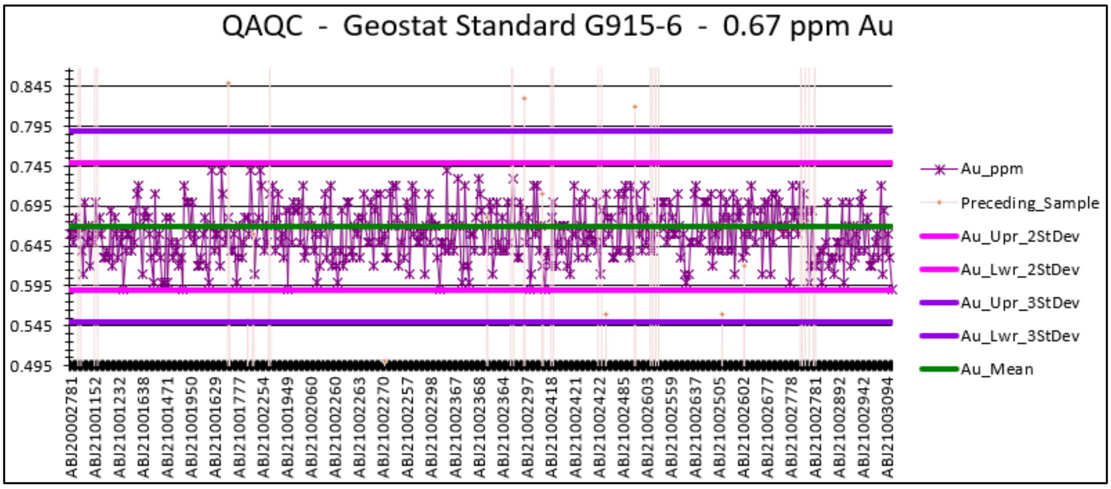

QAQC - Geostat Standard G915-6 - 0.67 ppm Au
0.845
0.795
0.745
0.695
0.645
0.595
0.545
0.495
Au_ppm
Preceding_Sample
Au_Upr_2StDev
Au_Lwr_2StDev
Au_Upr_3StDev
Au_Lwr_3StDev
Au_Mean

2021 CRM Plots – RC-DD:

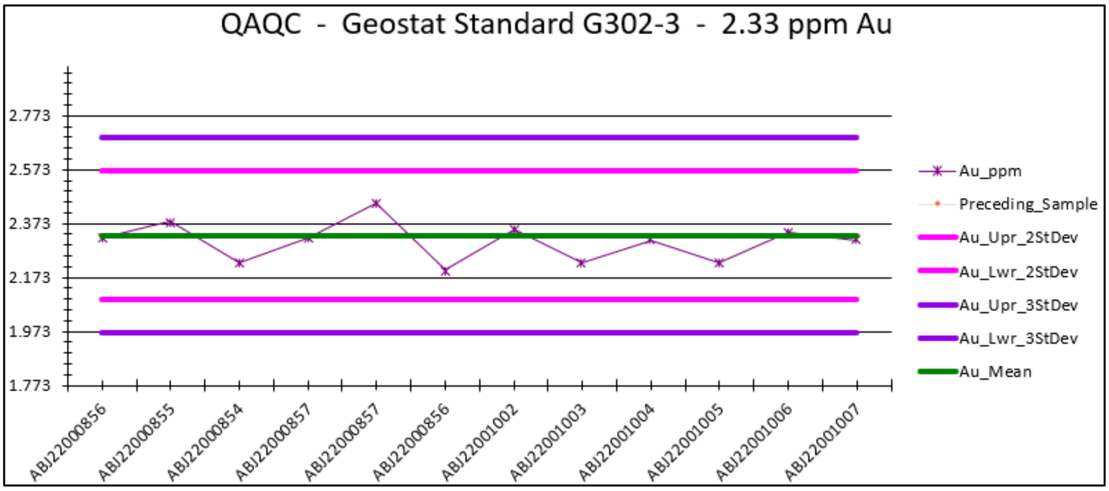


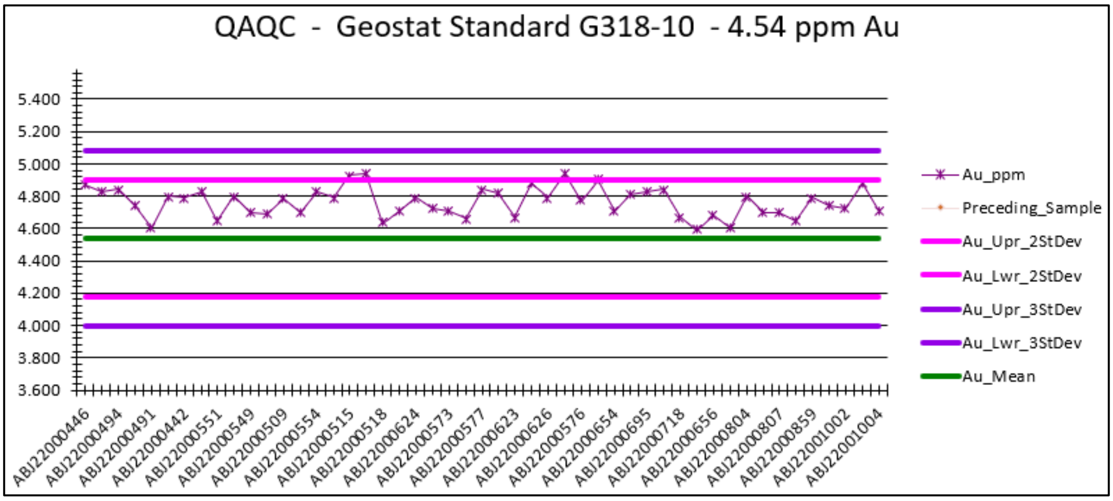


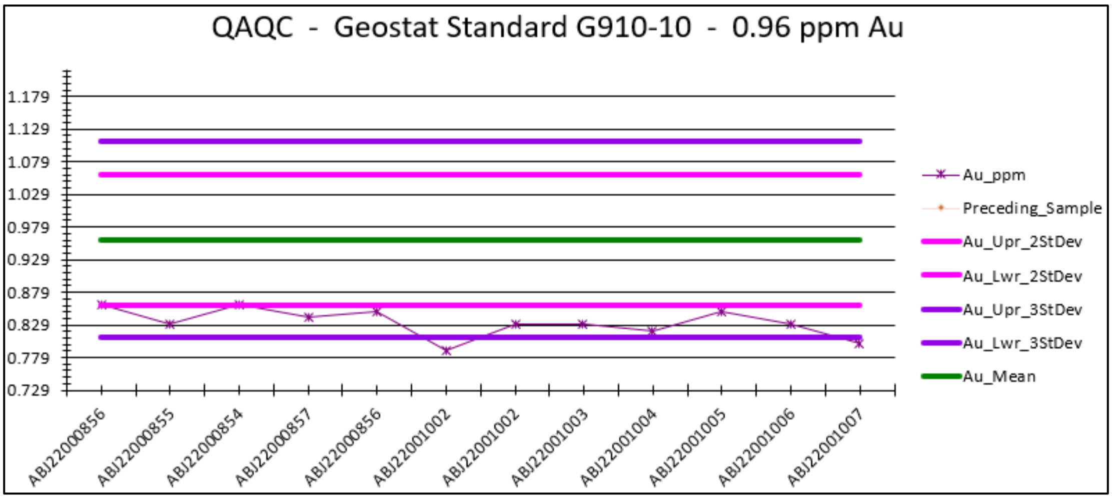


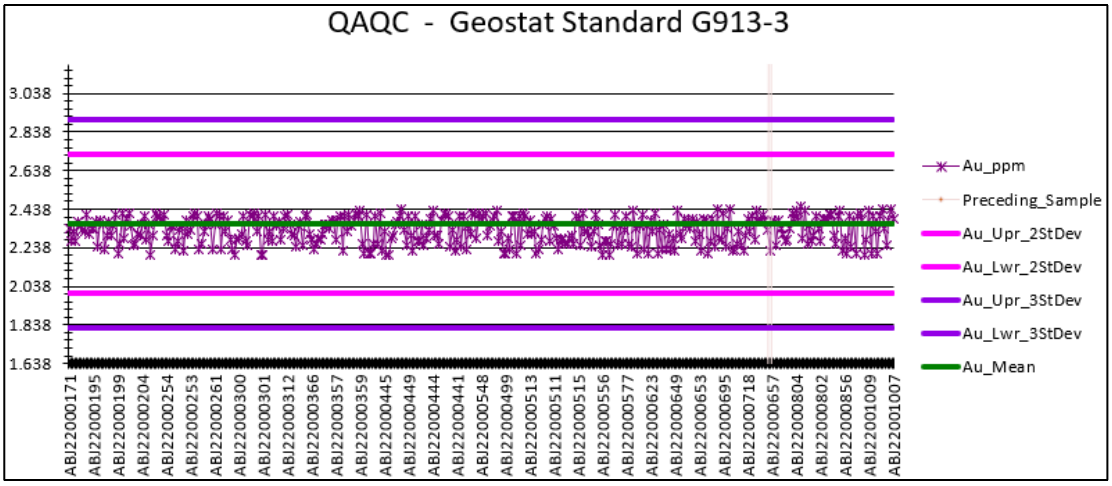


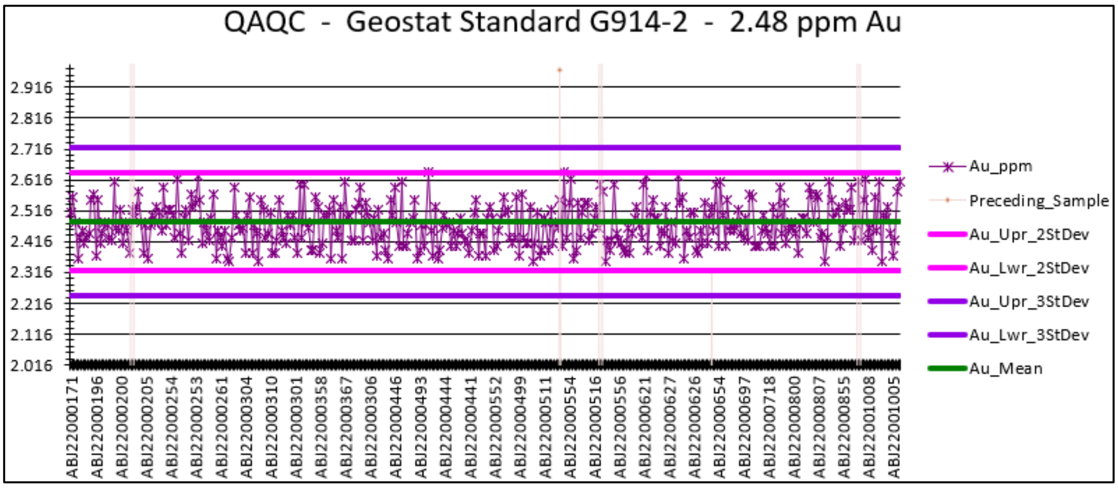
2022 CRM Plots – RC:

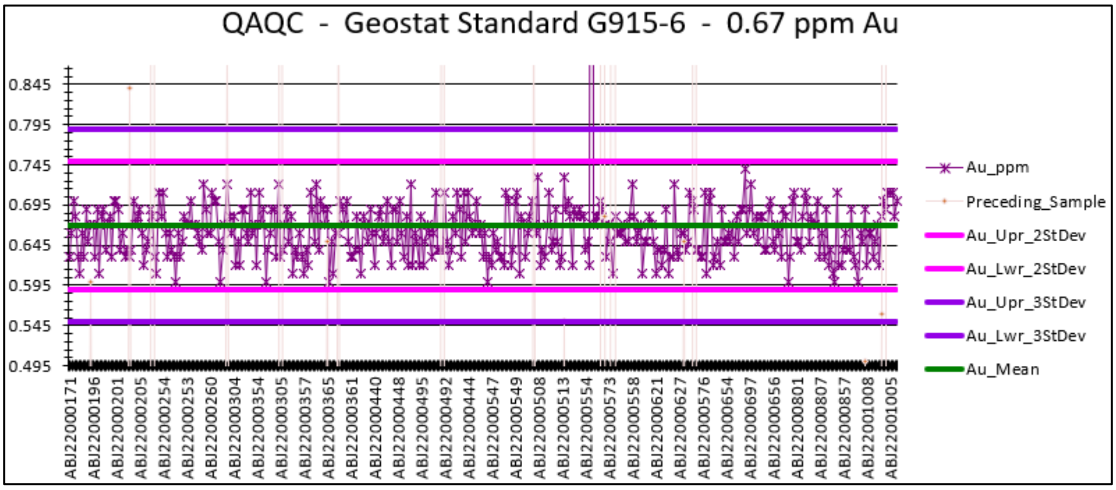




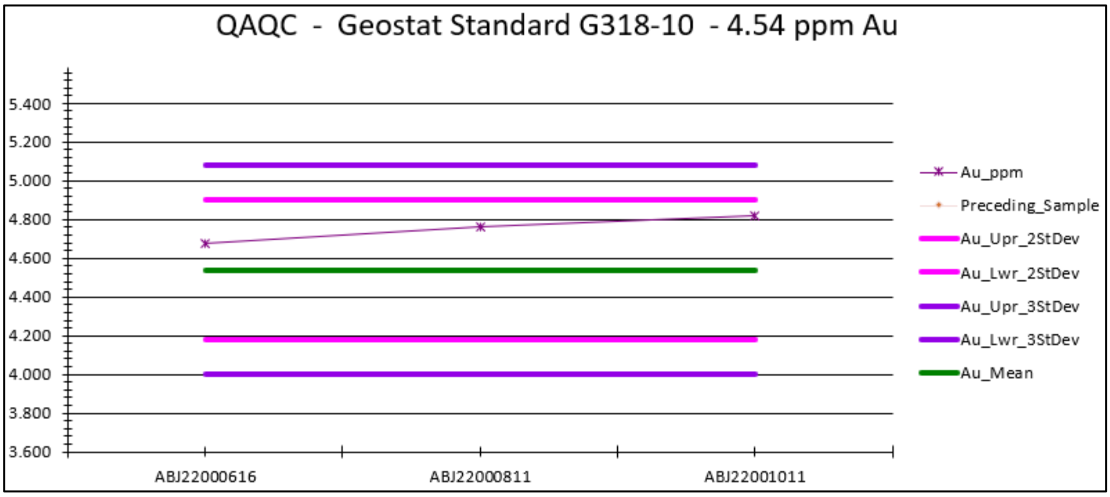

QAQC - Geostat Standard G910-10 - 0.96 ppm Au
1.179
1.129
1.079
1.029
0.979
0.929
0.879
0.829
0.779
0.729
ABJ22000856
ABJ22000855
ABJ22000854
ABJ22000857
ABJ22000856
ABJ22001002
ABJ22001002
ABJ22001003
ABJ22001004
ABJ22001005
ABJ22001006
ABJ22001007
Au_ppm
Preceding_Sample
Au_Upr_2StDev
Au_Lwr_2StDev
Au_Upr_3StDev
Au_Lwr_3StDev
Au_Mean

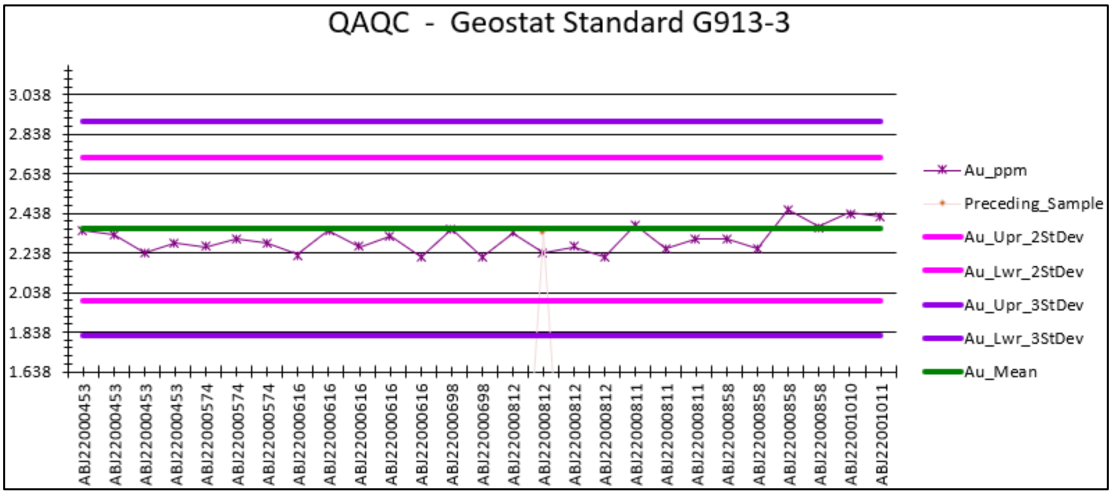

QAQC - Geostat Standard G913-3
3.038
2.838
2.638
2.438
2.238
2.038
1.838
1.638
ABJ22000171
ABJ22000195
ABJ22000199
ABJ22000204
ABJ22000254
ABJ22000253
ABJ22000261
ABJ22000300
ABJ22000301
ABJ22000312
ABJ22000366
ABJ22000357
ABJ22000359
ABJ22000445
ABJ22000449
ABJ22000444
ABJ22000441
ABJ22000548
ABJ22000499
ABJ22000513
ABJ22000511
ABJ22000515
ABJ22000556
ABJ22000577
ABJ22000623
ABJ22000649
ABJ22000653
ABJ22000695
ABJ22000718
ABJ22000657
ABJ22000804
ABJ22000802
ABJ22000856
ABJ22001009
ABJ22001007
Au_ppm
Preceding_Sample
Au_Upr_2StDev
Au_Lwr_2StDev
Au_Upr_3StDev
Au_Lwr_3StDev
Au_Mean

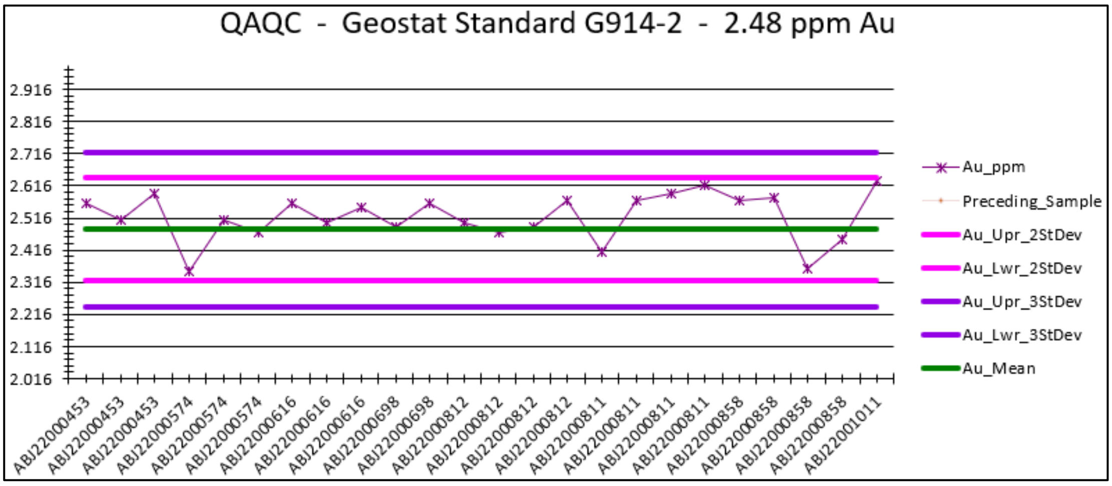

QAQC - Geostat Standard G914-2 - 2.48 ppm Au
2.916
2.816
2.716
2.616
2.516
2.416
2.316
2.216
2.116
2.016
ABJ22000171
ABJ22000196
ABJ22000200
ABJ22000205
ABJ22000254
ABJ22000253
ABJ22000261
ABJ22000304
ABJ22000310
ABJ22000301
ABJ22000358
ABJ22000367
ABJ22000306
ABJ22000446
ABJ22000493
ABJ22000444
ABJ22000441
ABJ22000552
ABJ22000499
ABJ22000511
ABJ22000554
ABJ22000516
ABJ22000556
ABJ22000621
ABJ22000627
ABJ22000626
ABJ22000654
ABJ22000697
ABJ22000718
ABJ22000800
ABJ22000807
ABJ22000855
ABJ22001008
ABJ22001005
Au_ppm
Preceding_Sample
Au_Upr_2StDev
Au_Lwr_2StDev
Au_Upr_3StDev
Au_Lwr_3StDev
Au_Mean



2022 CRM Plots – RC-DD:






QAQC - Geostat Standard G914-2 - 2.48 ppm Au
2.916
2.816
2.716
2.616
2.516
2.416
2.316
2.216
2.116
2.016
ABJ22000453
ABJ22000453
ABJ22000453
ABJ22000574
ABJ22000574
ABJ22000574
ABJ22000616
ABJ22000616
ABJ22000616
ABJ22000698
ABJ22000698
ABJ22000812
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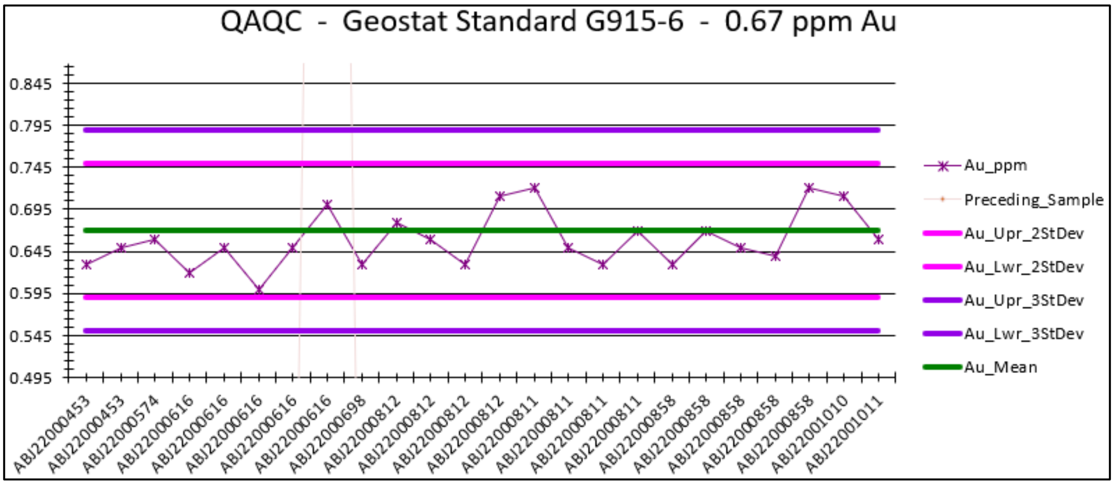
ABJ22000811
ABJ22000811
ABJ22000811
ABJ22000811
ABJ22000858
ABJ22000858
ABJ22000858
ABJ22000858
ABJ22001011
Au_ppm
Preceding_Sample
Au_Upr_2StDev
Au_Lwr_2StDev
Au_Upr_3StDev
Au_Lwr_3StDev
Au_Mean


QAQC - Geostat Standard G915-6 - 0.67 ppm Au
0.845
0.795
0.745
0.695
0.645
0.595
0.545
0.495
ABJ22000453
ABJ22000453
ABJ22000574
ABJ22000616
ABJ22000616
ABJ22000616
ABJ22000616
ABJ22000616
ABJ22000698
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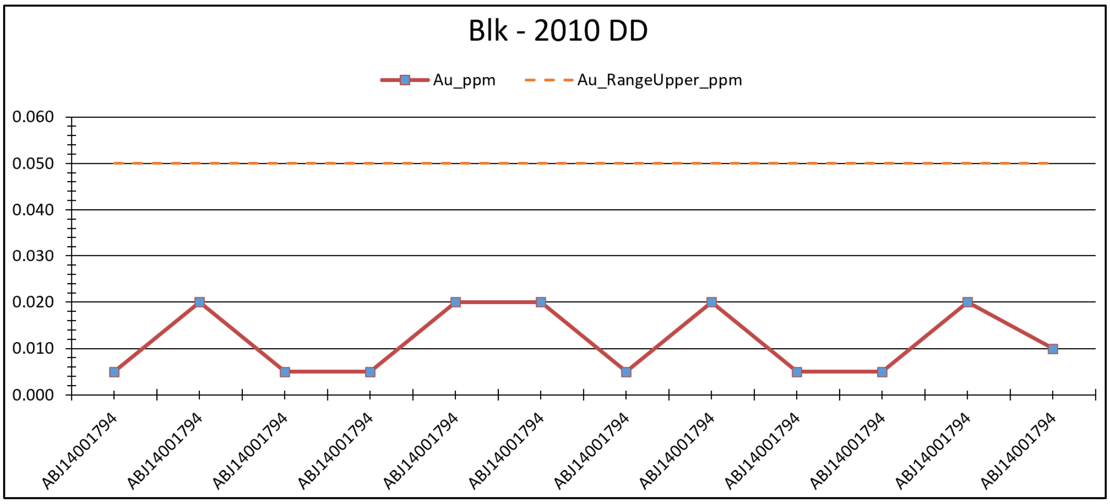
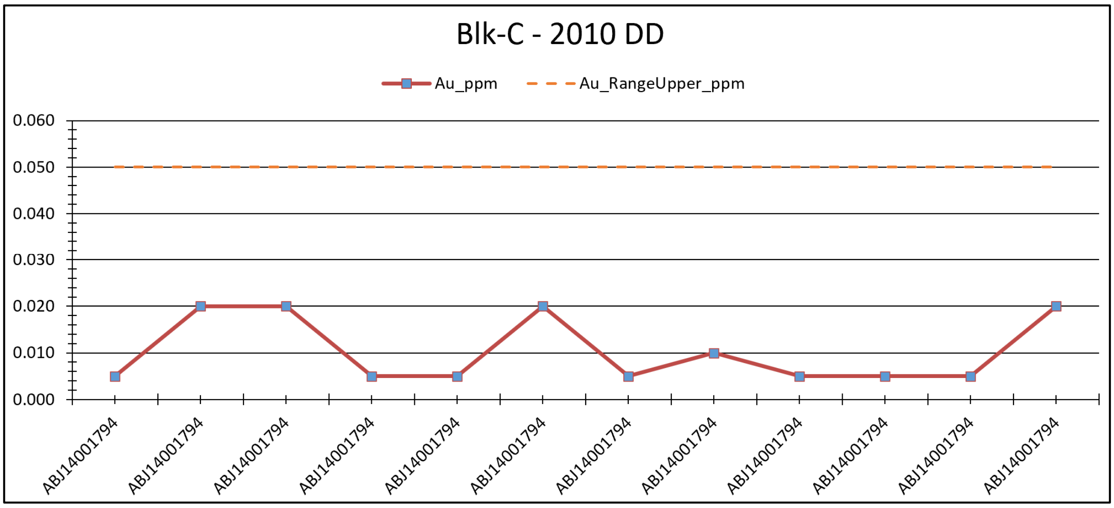
ABJ22000811
ABJ22000811
ABJ22000858
ABJ22000858
ABJ22000858
ABJ22000858
ABJ22000858
ABJ22001010
ABJ22001011
Au_ppm
Preceding_Sample
Au_Upr_2StDev
Au_Lwr_2StDev
Au_Upr_3StDev
Au_Lwr_3StDev
Au_Mean

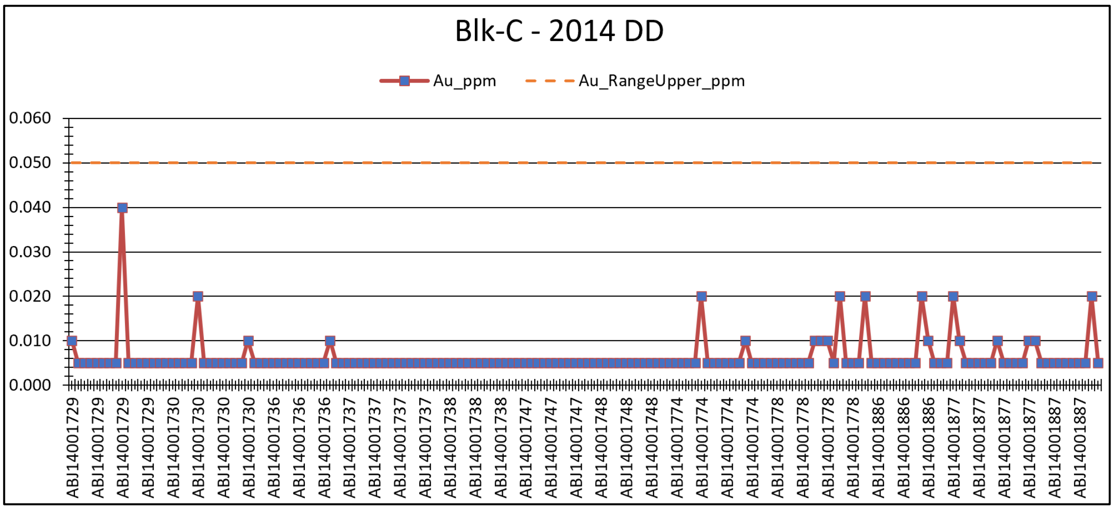
Blank Plots:

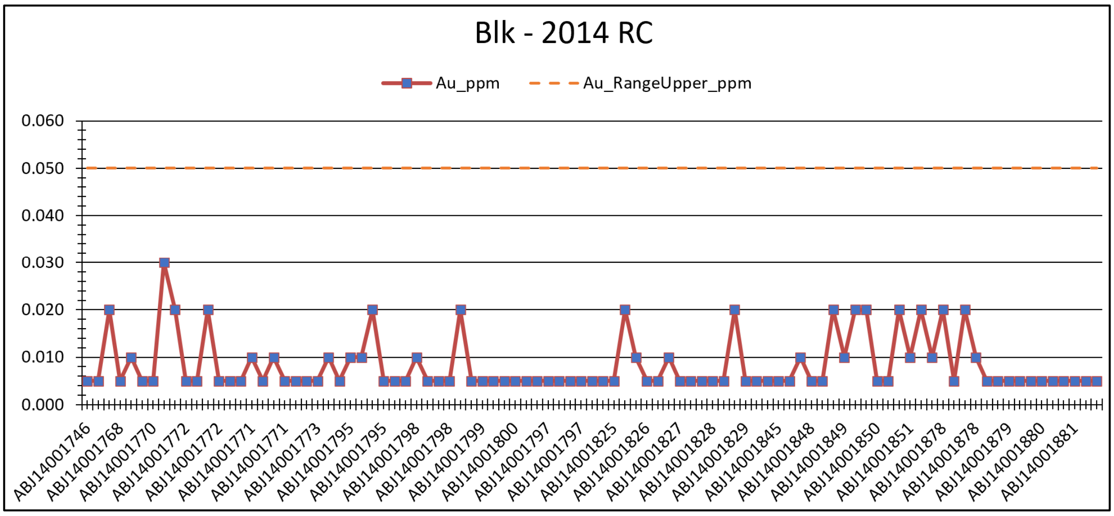
2010 Blank Plots:

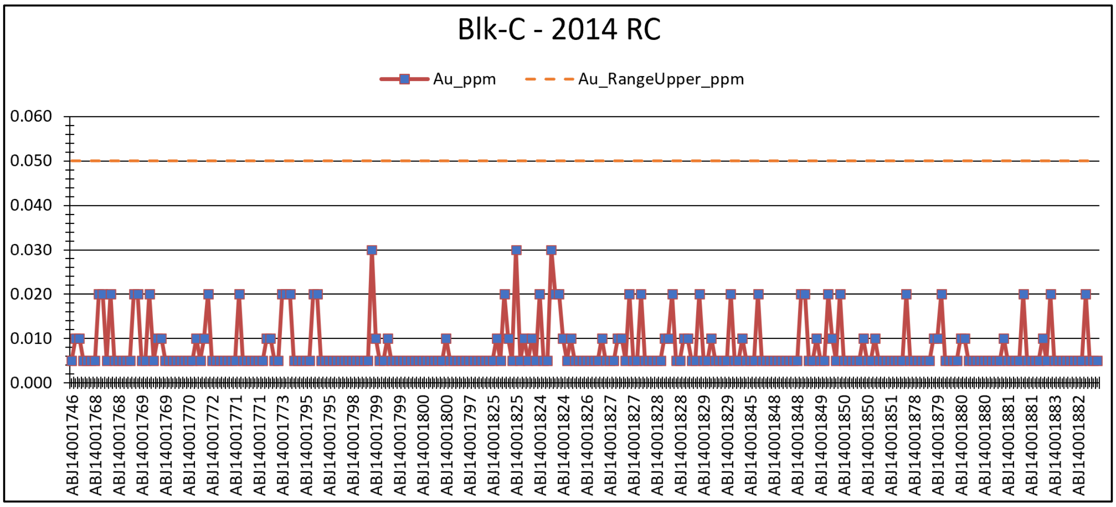




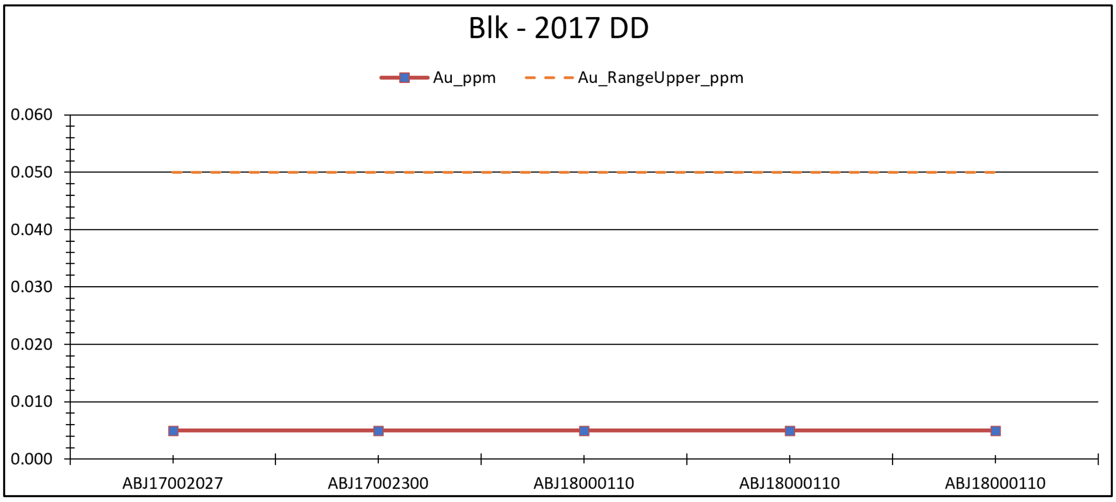
2014 Blank Plots:

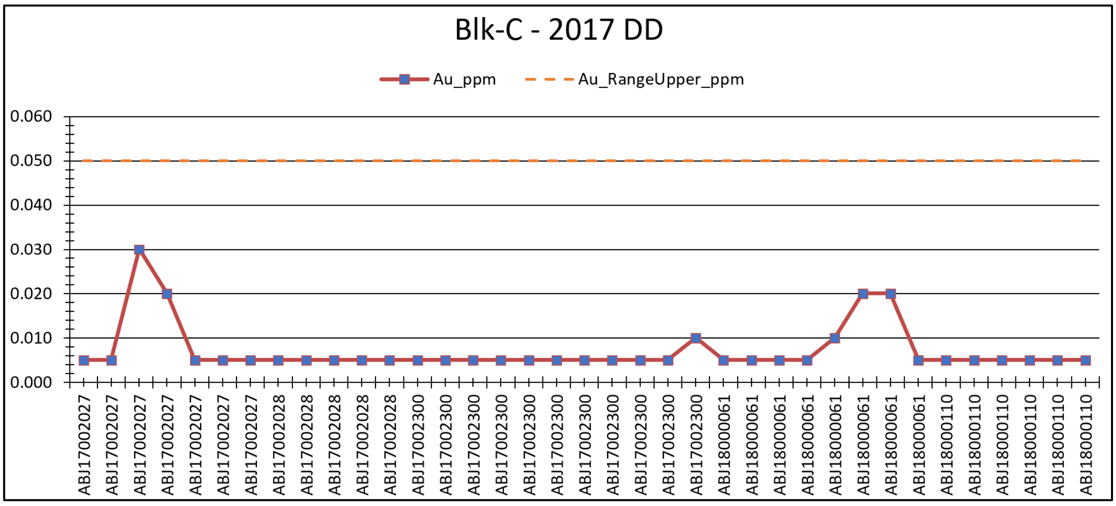






2017 Blank Plots:




Blk-C - 2017 DD
Au_ppm
Au_RangeUpper_ppm
0.060
0.050
0.040
0.030
0.020
0.010
0.000
ABJ17002027
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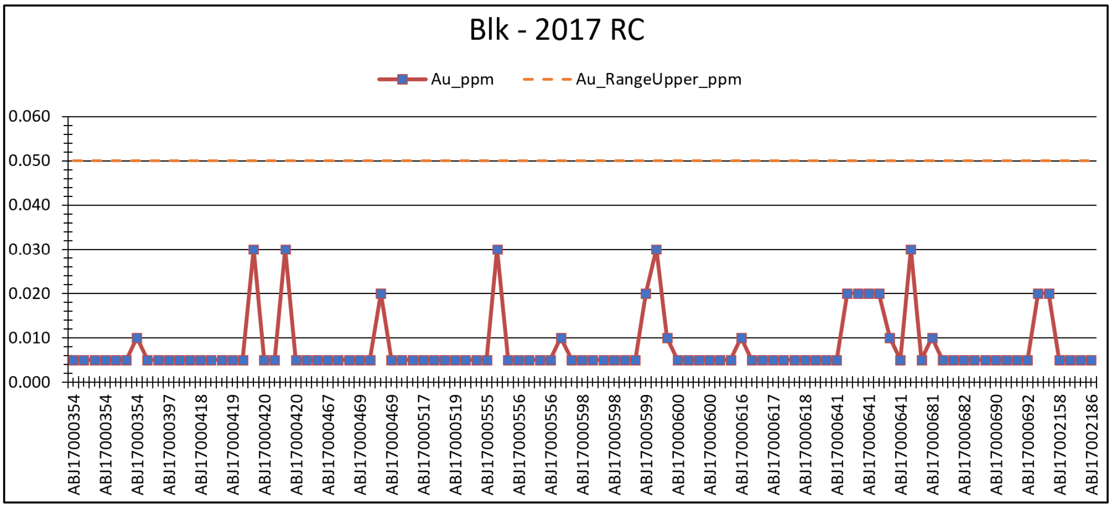
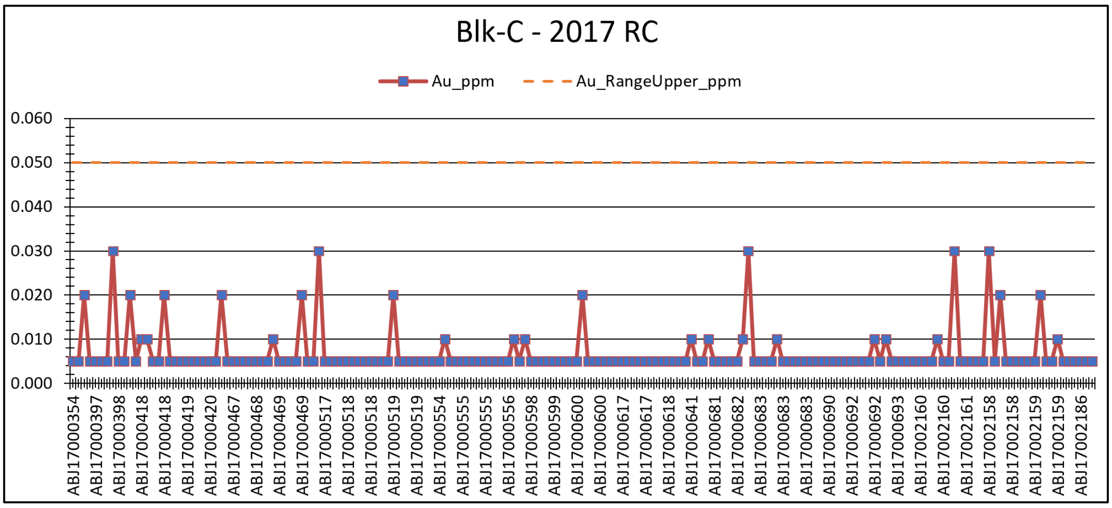
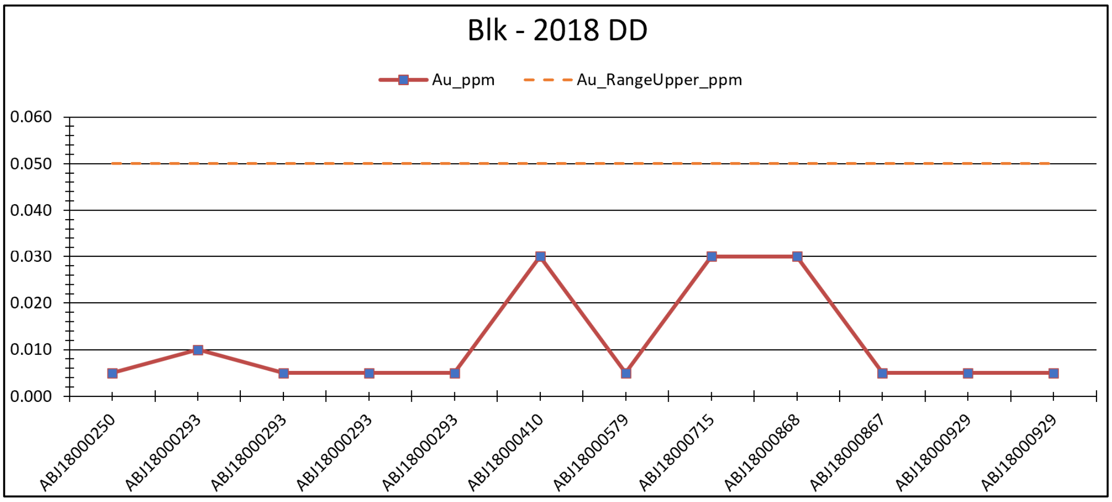
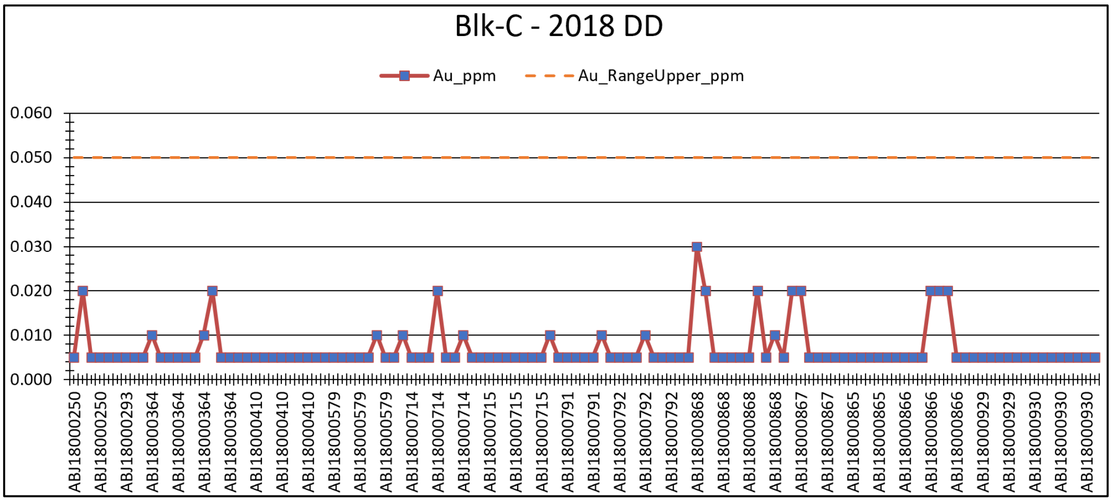
ABJ17002300
ABJ18000061
ABJ18000110

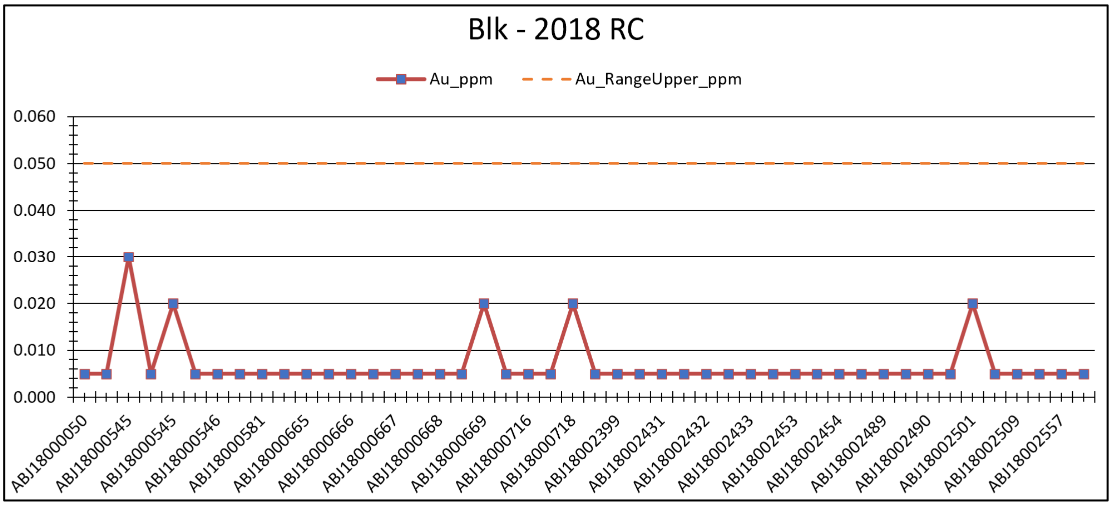

Blk - 2017 RC
Au_ppm
Au_RangeUpper_ppm
0.060
0.050
0.040
0.030
0.020
0.010
0.000

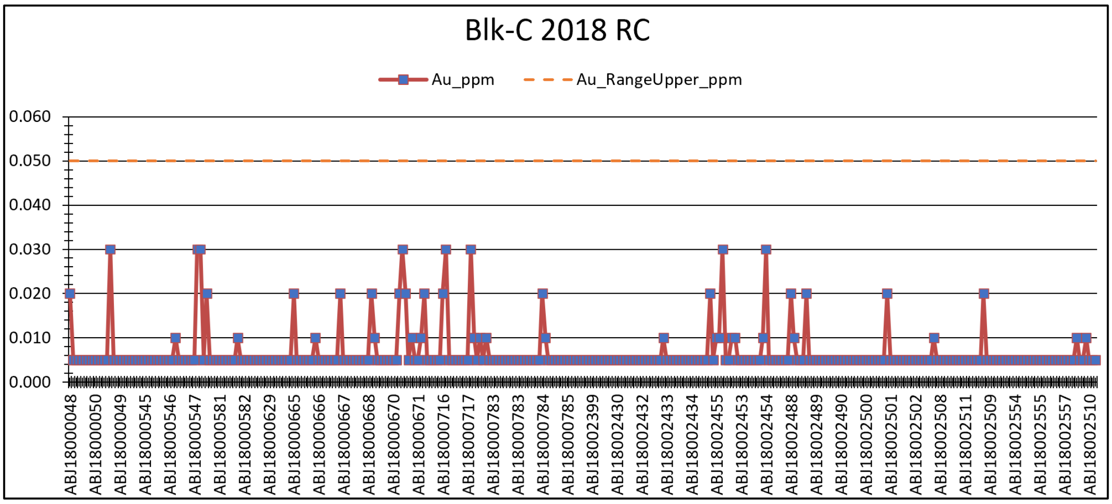

Blk-C - 2017 RC
Au_ppm
Au_RangeUpper_ppm
0.060
0.050
0.040
0.030
0.020
0.010
0.000

2018 Blank Plots:








Blk-C 2018 RC
Au_ppm
Au_RangeUpper_ppm
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0.050
0.040
0.030
0.020
0.010
0.000
ABJ18000048
ABJ18000050
ABJ18000049
ABJ18000545
ABJ18000546
ABJ18000547
ABJ18000581
ABJ18000582
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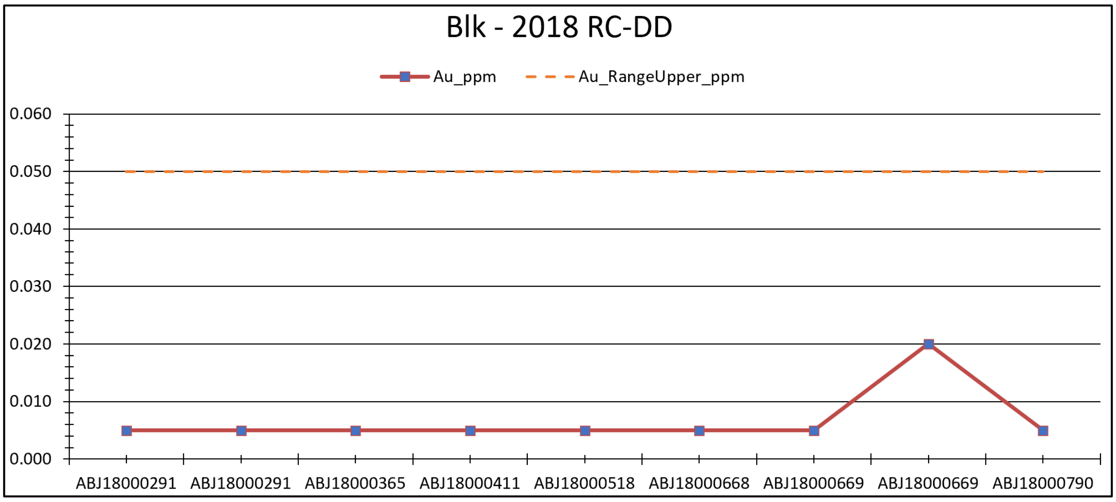
ABJ18002510


Blk - 2018 RC-DD
Au_ppm
Au_RangeUpper_ppm
0.060
0.050
0.040
0.030
0.020
0.010
0.000
ABJ18000291
ABJ18000291
ABJ18000365
ABJ18000411
ABJ18000518
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ABJ18000669
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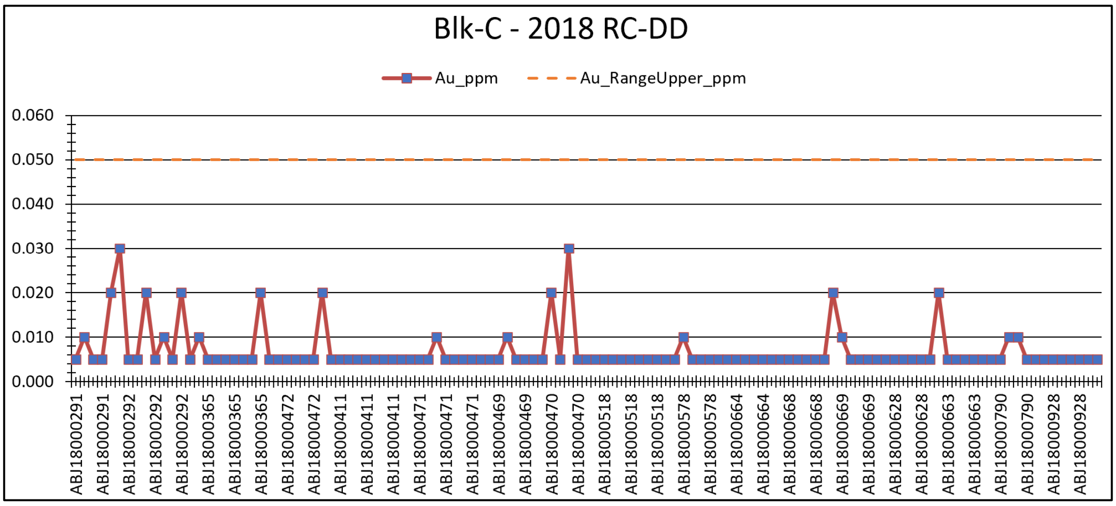
ABJ18000790


Blk-C - 2018 RC-DD
Au_ppm
Au_RangeUpper_ppm
0.060
0.050
0.040
0.030
0.020
0.010
0.000
ABJ18000291
ABJ18000291
ABJ18000292
ABJ18000292
ABJ18000292
ABJ18000365
ABJ18000365
ABJ18000365
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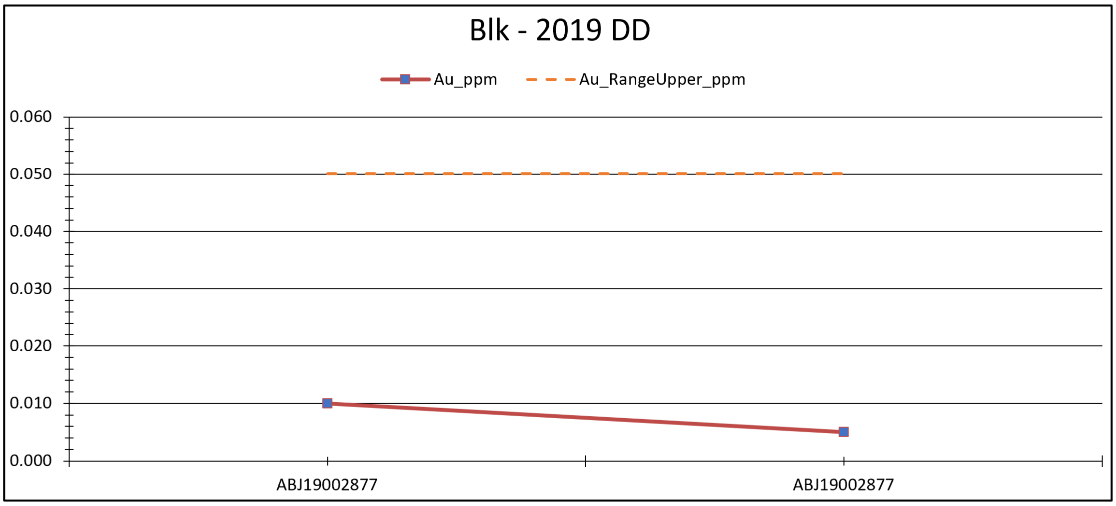
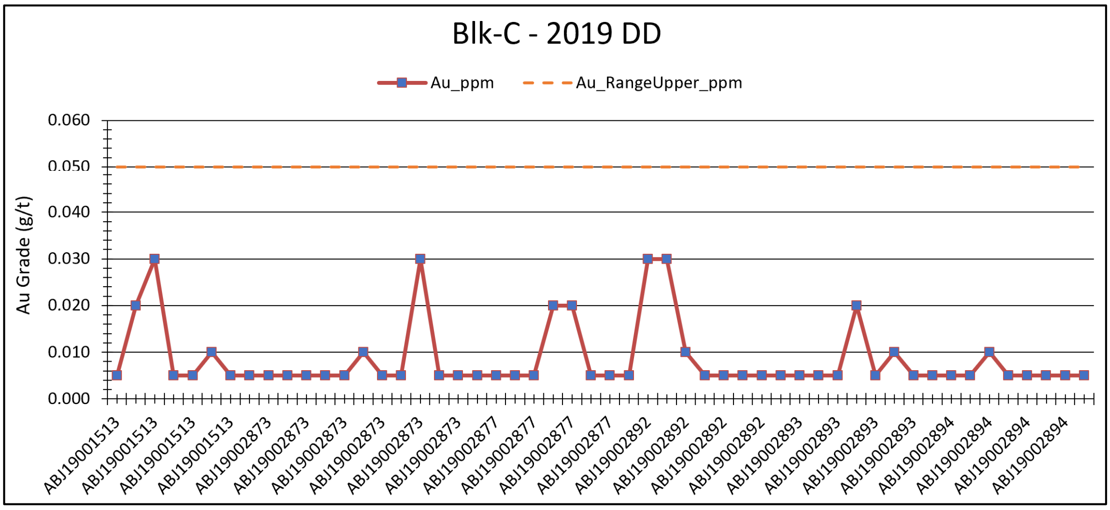
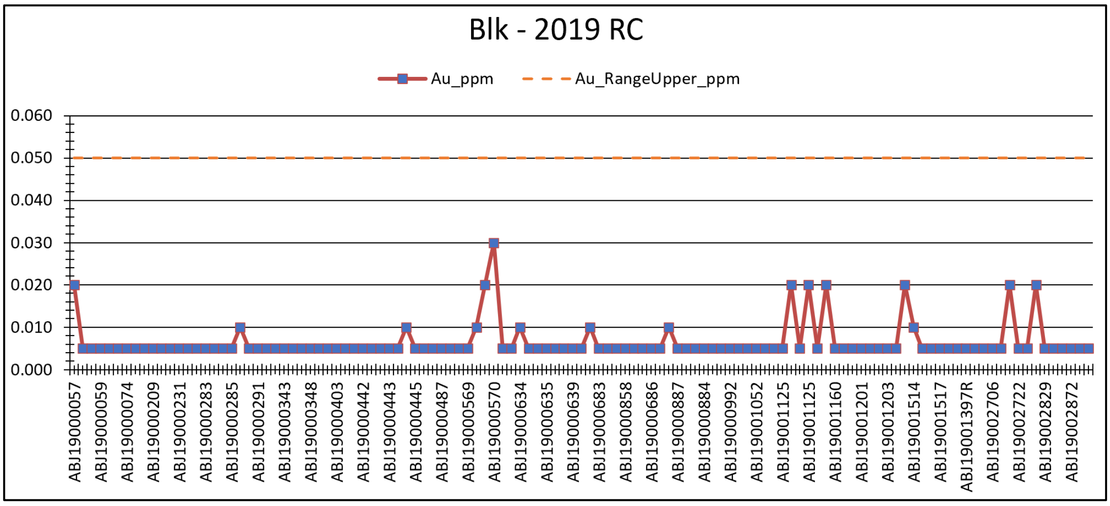
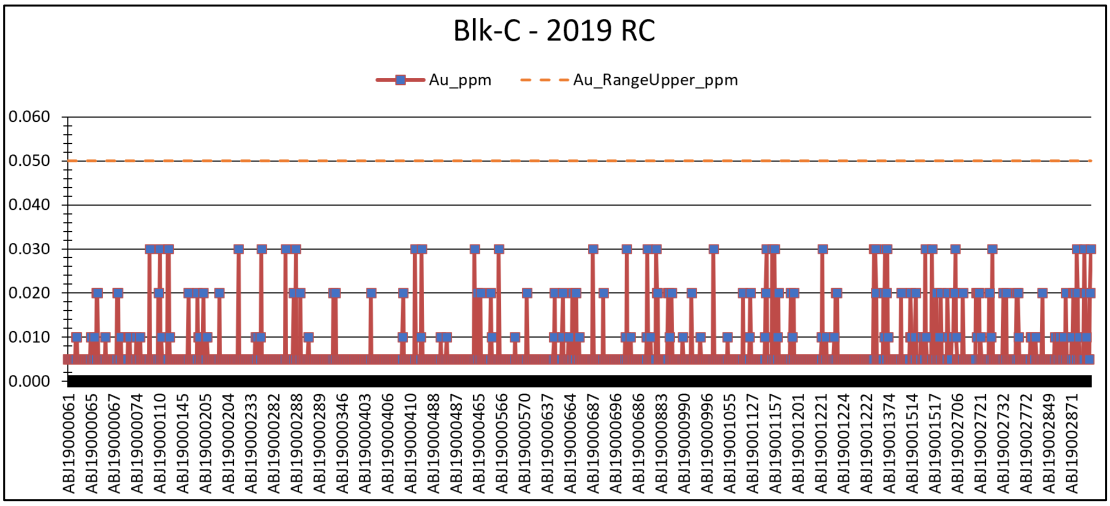
ABJ18000790
ABJ18000928
ABJ18000928

2019 Blank Plots:








Blk-C - 2019 RC
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Au_RangeUpper_ppm
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0.050
0.040
0.030
0.020
0.010
0.000
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ABJ19000110
ABJ19000145
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ABJ19000233
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ABJ19000403
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ABJ19000410
ABJ19000488
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ABJ19000465
ABJ19000566
ABJ19000570
ABJ19000637
ABJ19000664
ABJ19000687
ABJ19000696
ABJ19000686
ABJ19000883
ABJ19000990
ABJ19000996
ABJ19001055
ABJ19001127
ABJ19001157
ABJ19001201
ABJ19001221
ABJ19001224
ABJ19001222
ABJ19001374
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ABJ19001517
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ABJ19002721
ABJ19002732
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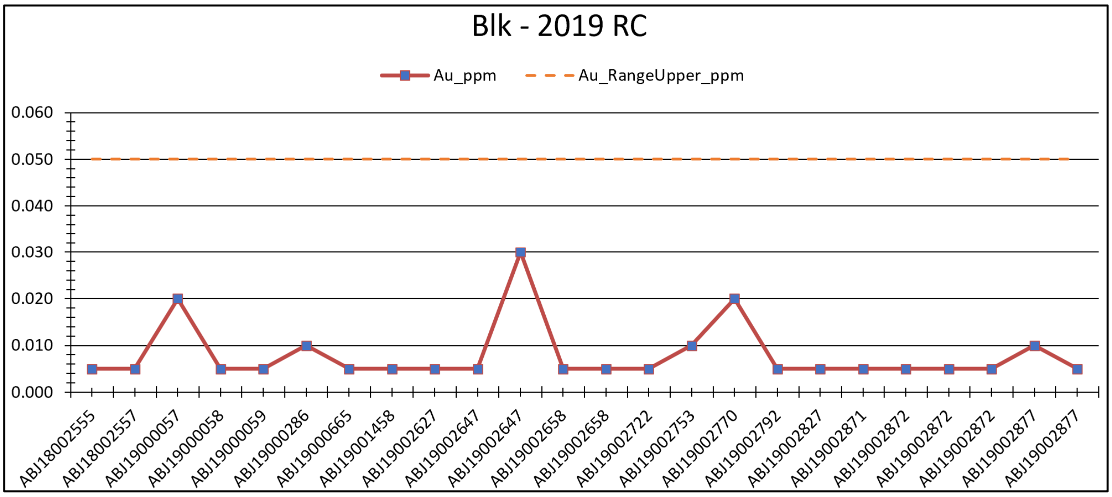
ABJ19002849
ABJ19002871


Blk - 2019 RC
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Au_RangeUpper_ppm
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0.050
0.040
0.030
0.020
0.010
0.000
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ABJ18002557
ABJ19000057
ABJ19000058
ABJ19000059
ABJ19000286
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ABJ19002647
ABJ19002647
ABJ19002658
ABJ19002658
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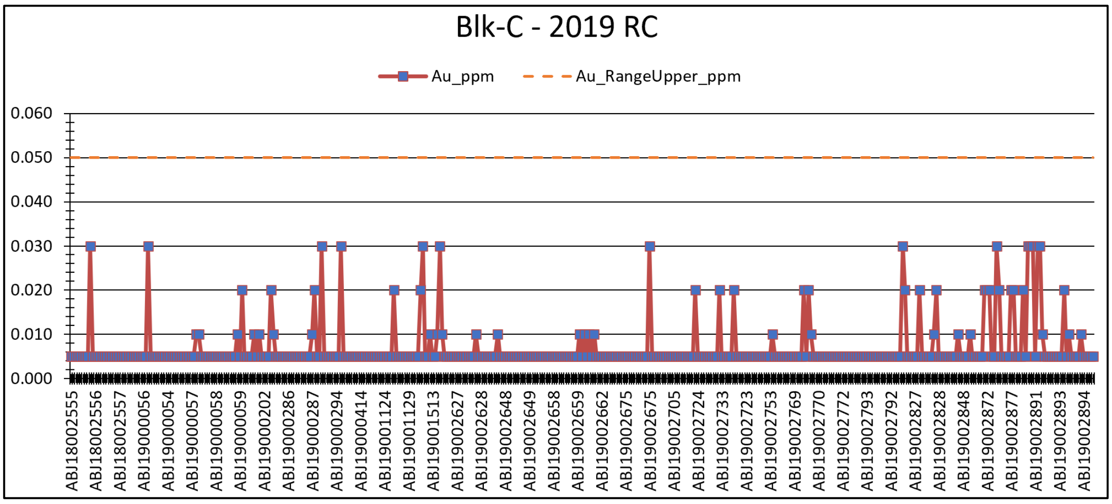
ABJ19002877
ABJ19002877


Blk-C - 2019 RC
Au_ppm
Au_RangeUpper_ppm
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0.050
0.040
0.030
0.020
0.010
0.000
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ABJ18002556
ABJ18002557
ABJ19000056
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ABJ19000058
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ABJ19002659
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ABJ19002894

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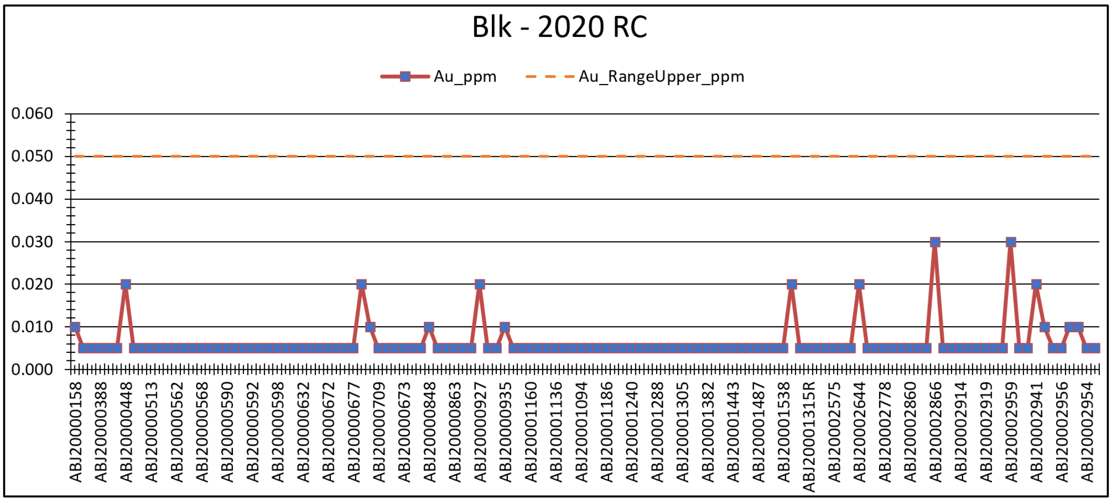

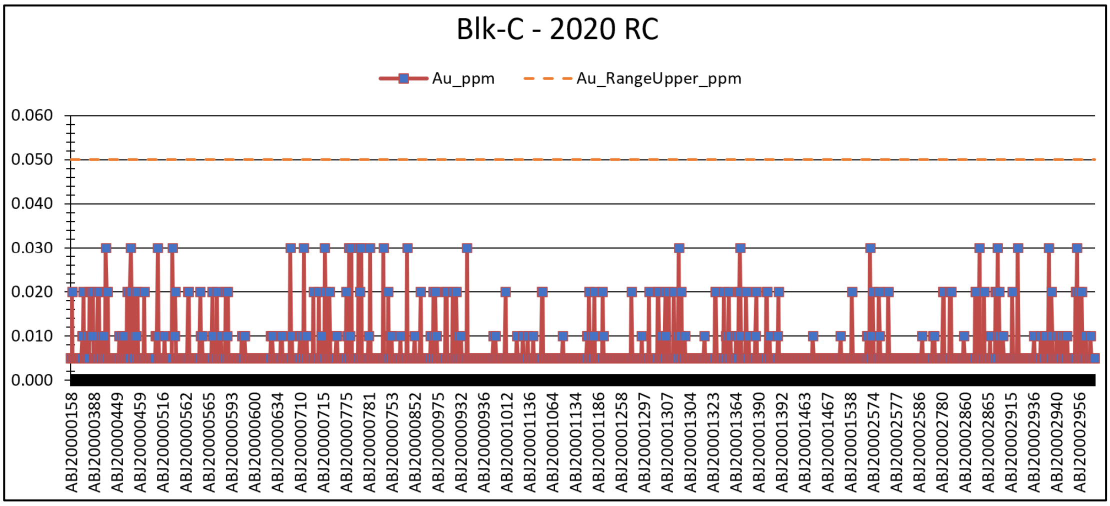

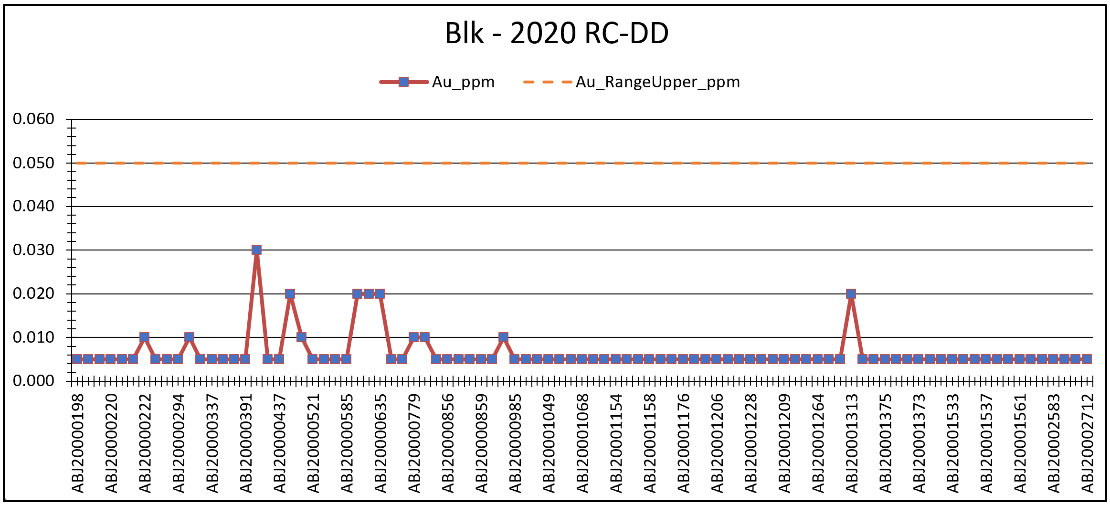

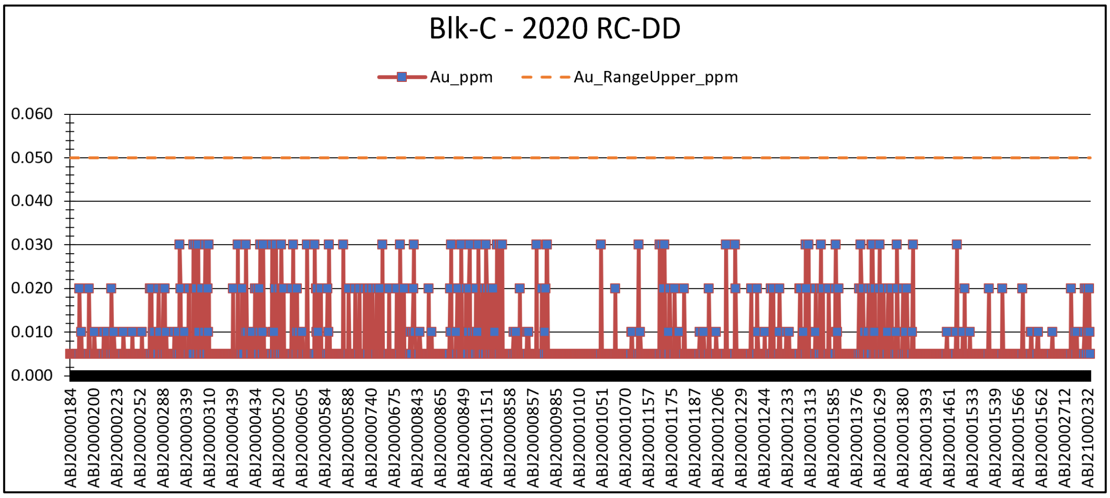

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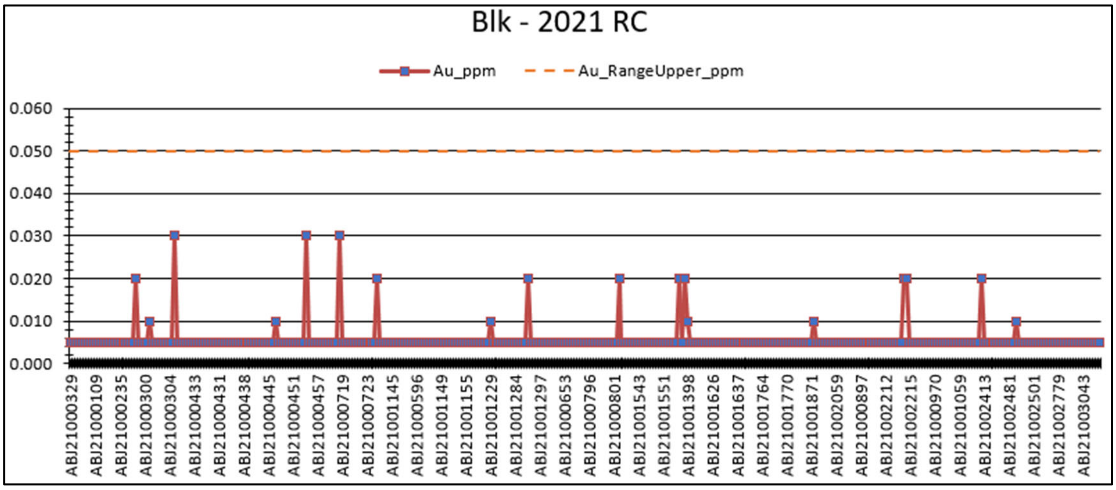

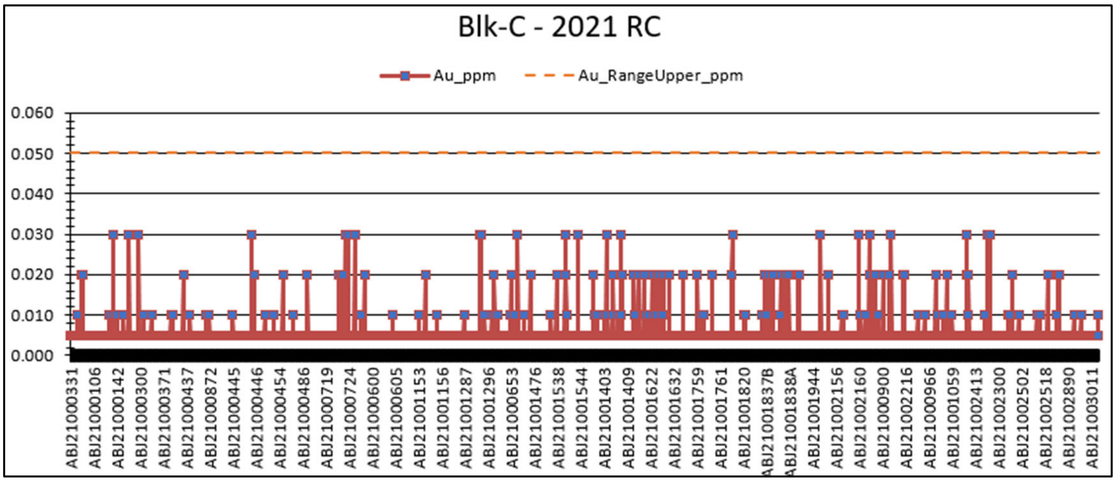

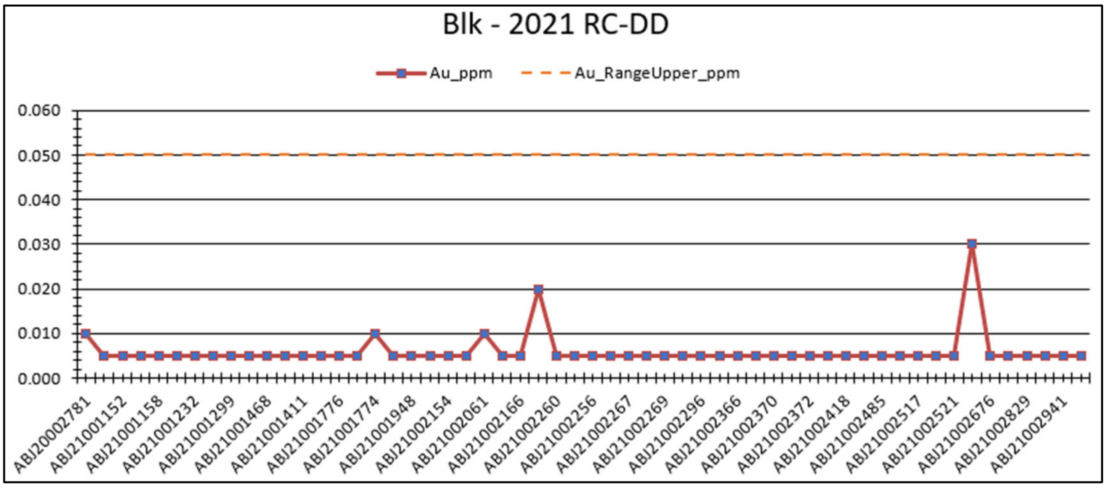
Blk - 2021 RC-DD
Au_ppm
Au_RangeUpper_ppm
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0.050
0.040
0.030
0.020
0.010
0.000
ABJ20002781
ABJ21001152
ABJ21001158
ABJ21001232
ABJ21001299
ABJ21001468
ABJ21001411
ABJ21001776
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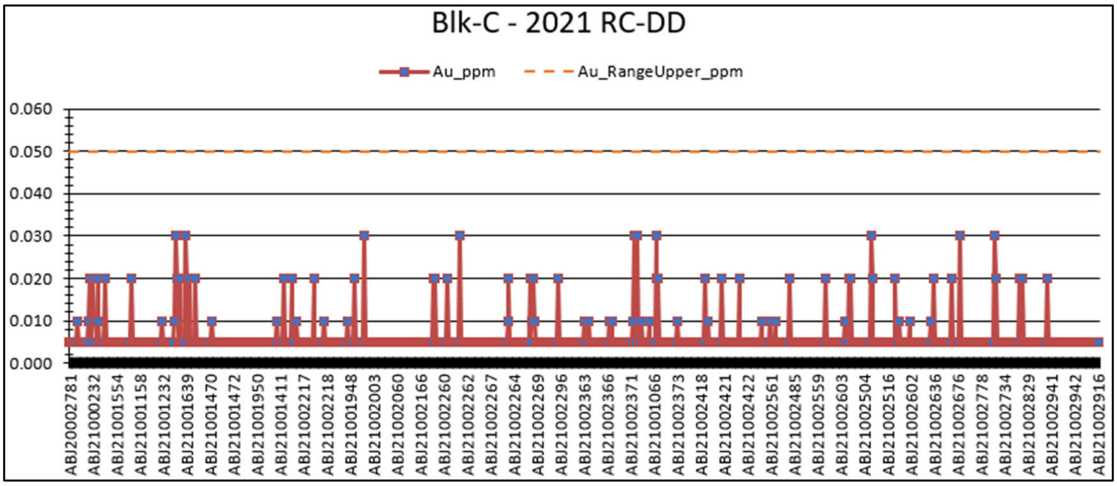
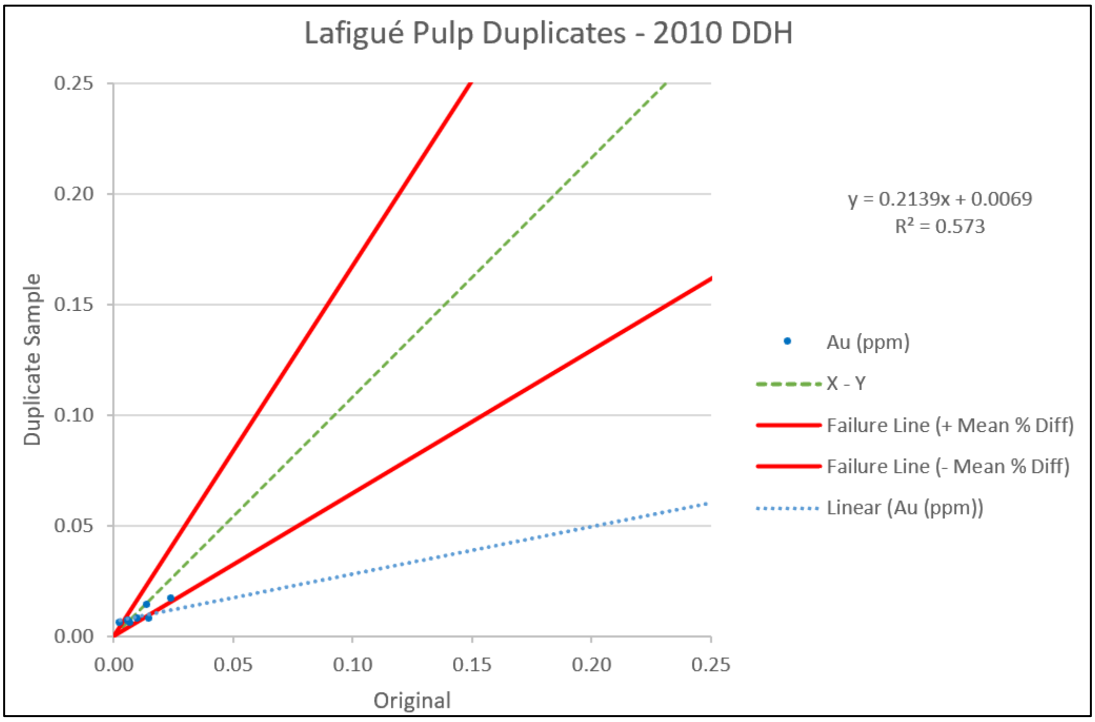
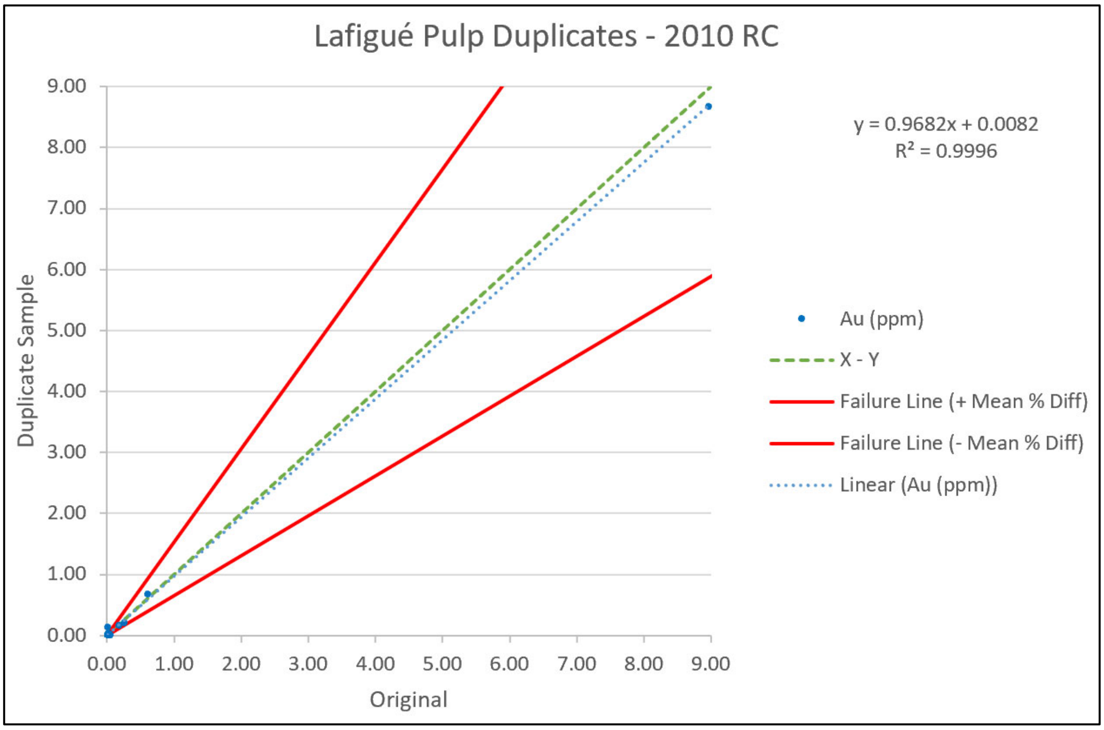
ABJ21002941


Blk-C - 2021 RC-DD
Au_ppm
Au_RangeUpper_ppm
0.060
0.050
0.040
0.030
0.020
0.010
0.000

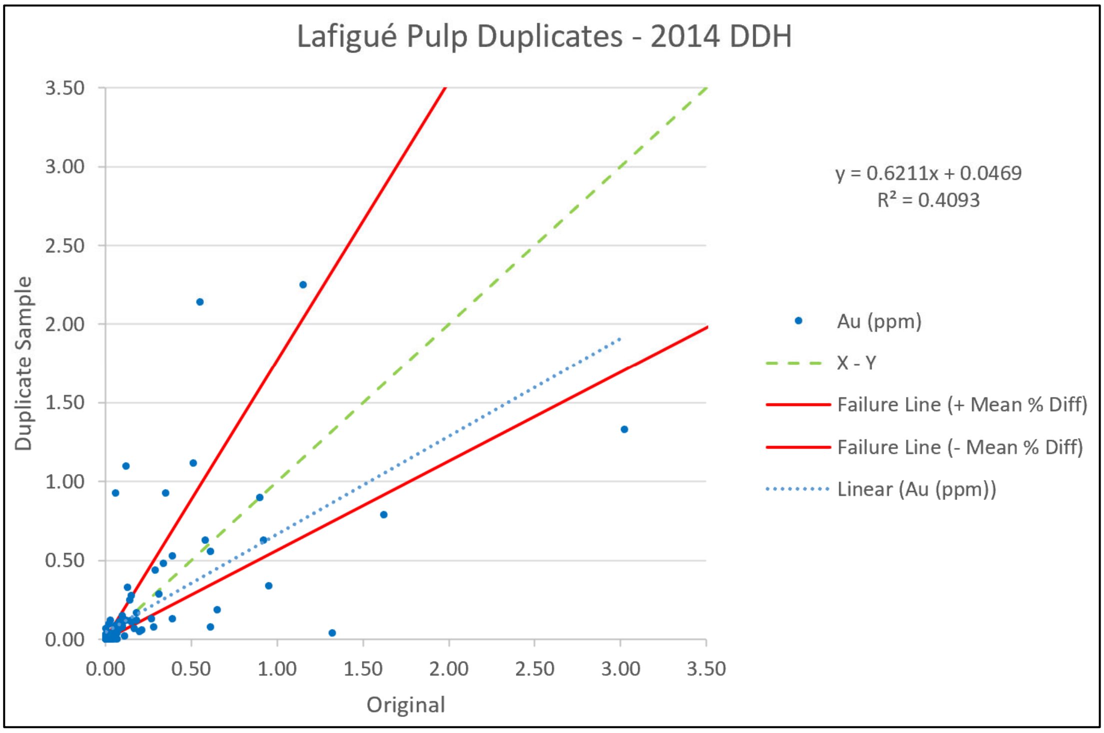
Duplicate Plots:

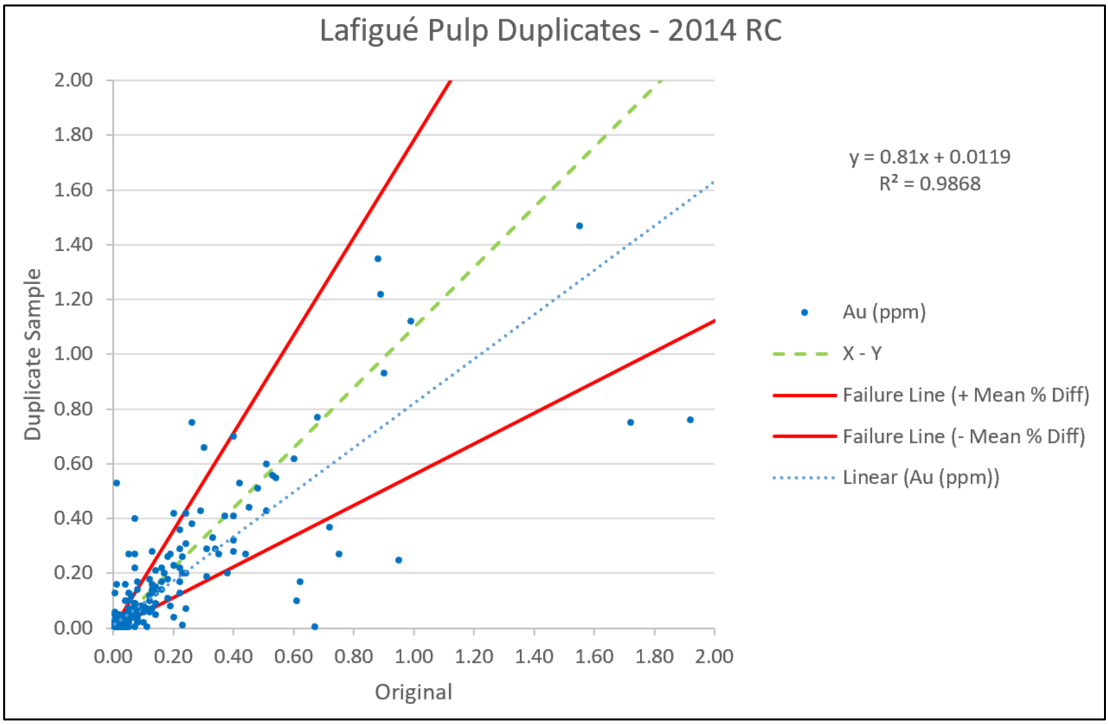
2010 Pulp Duplicate Plots:



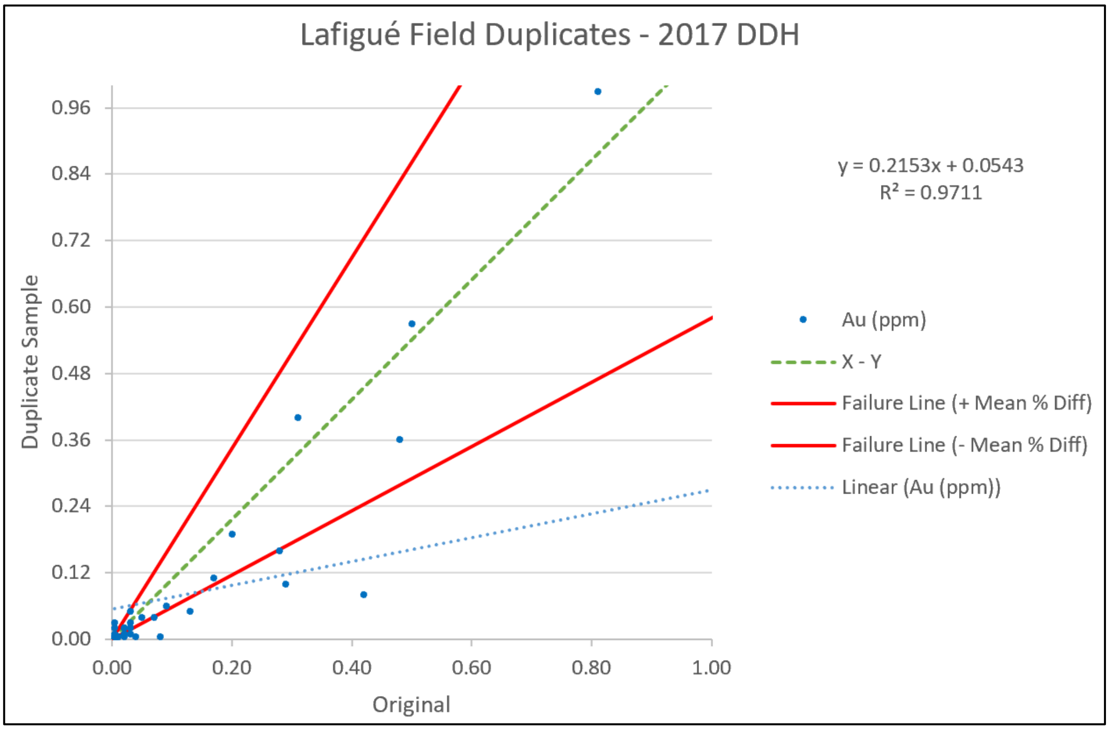


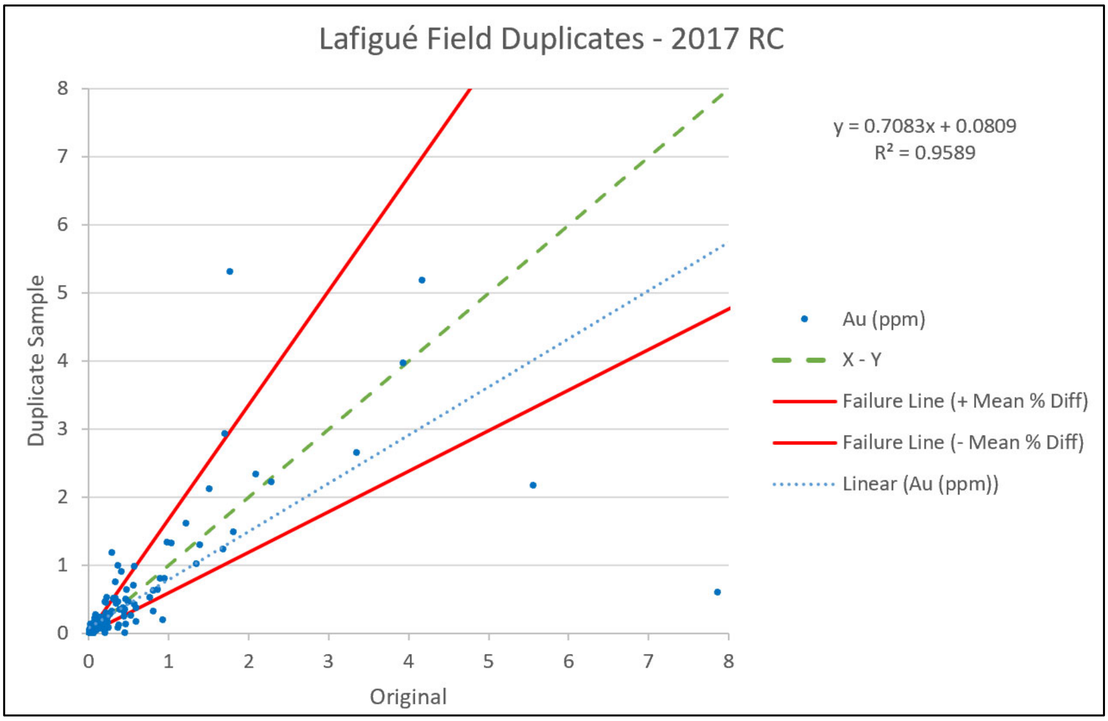
2014 Pulp Duplicate Plots:



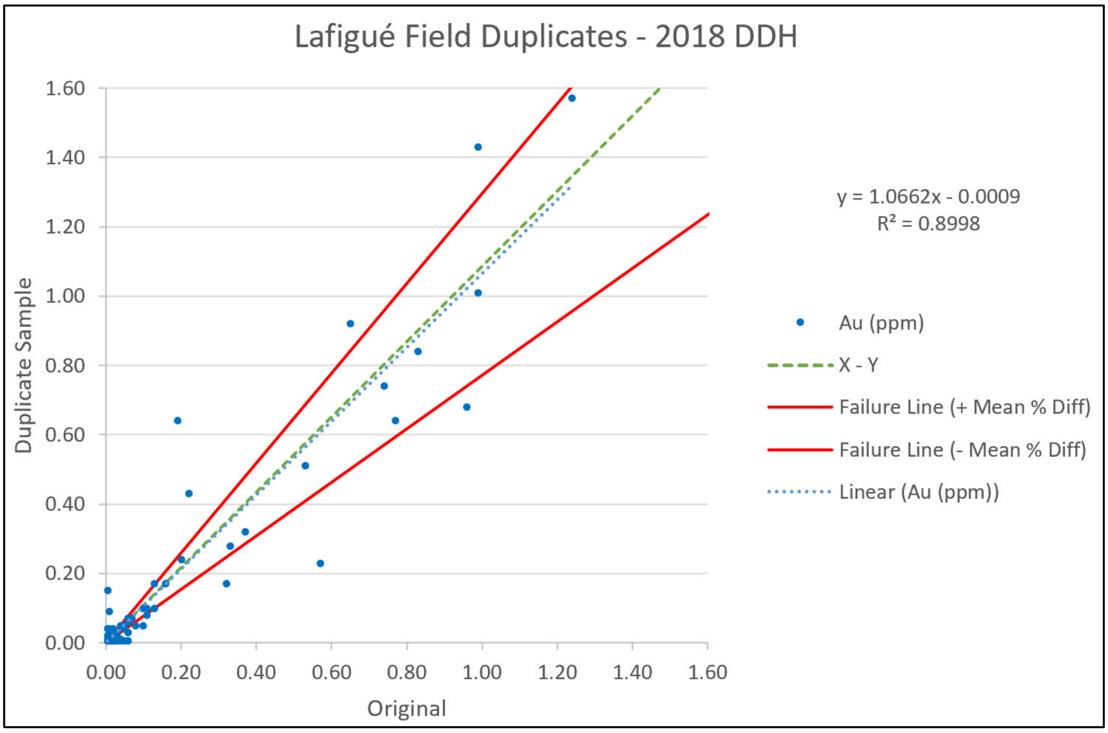


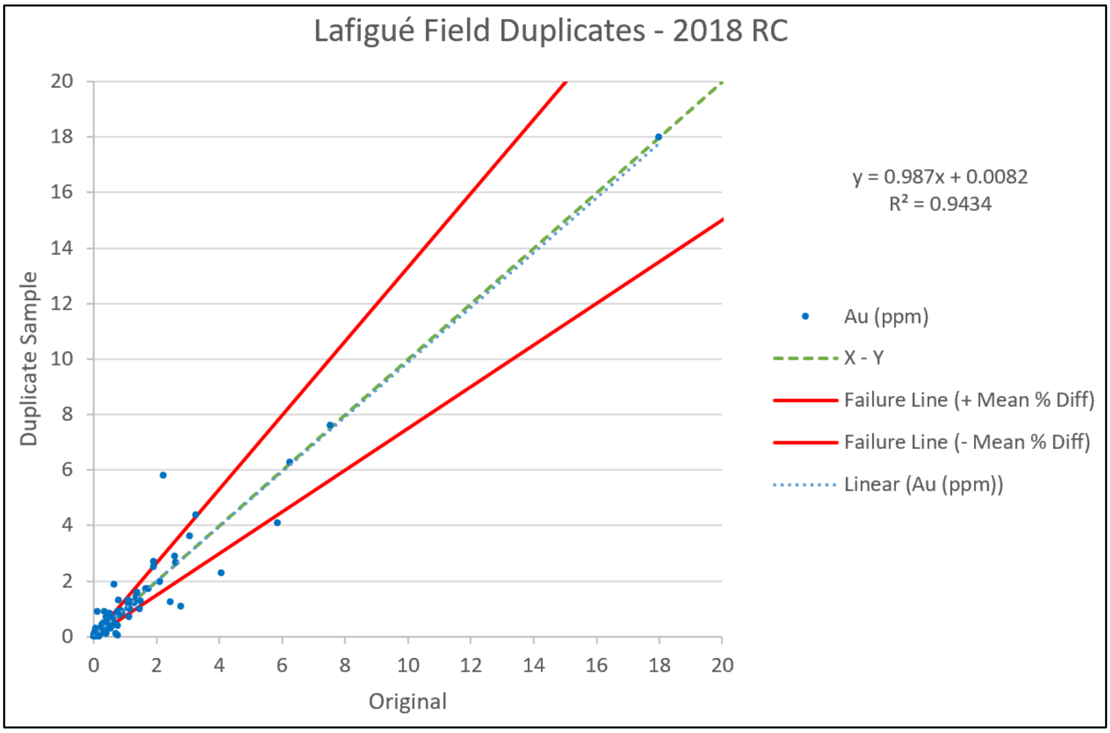
2017 Field Duplicate Plots:

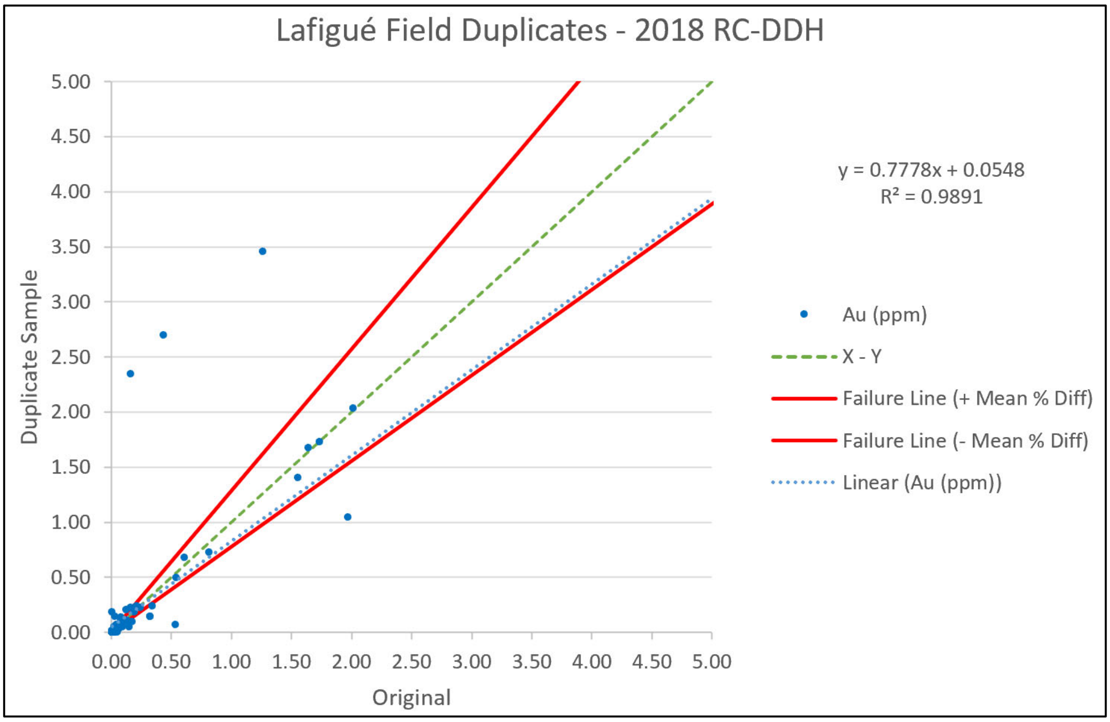




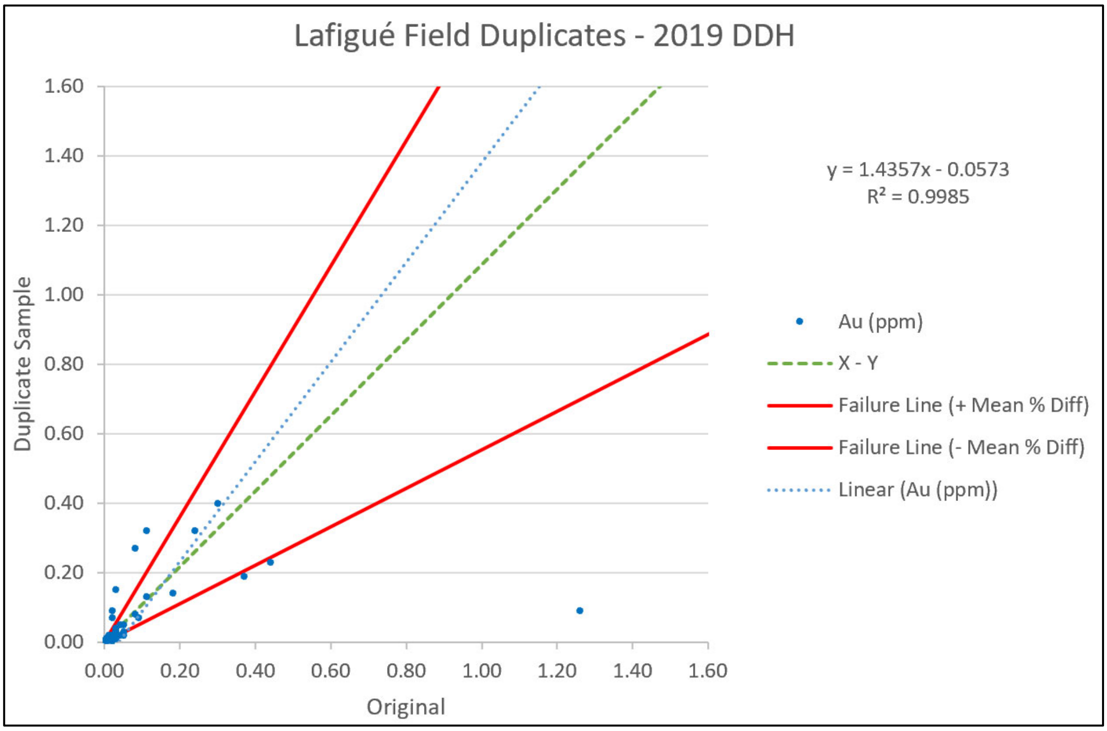
2018 Field Duplicate Plots:

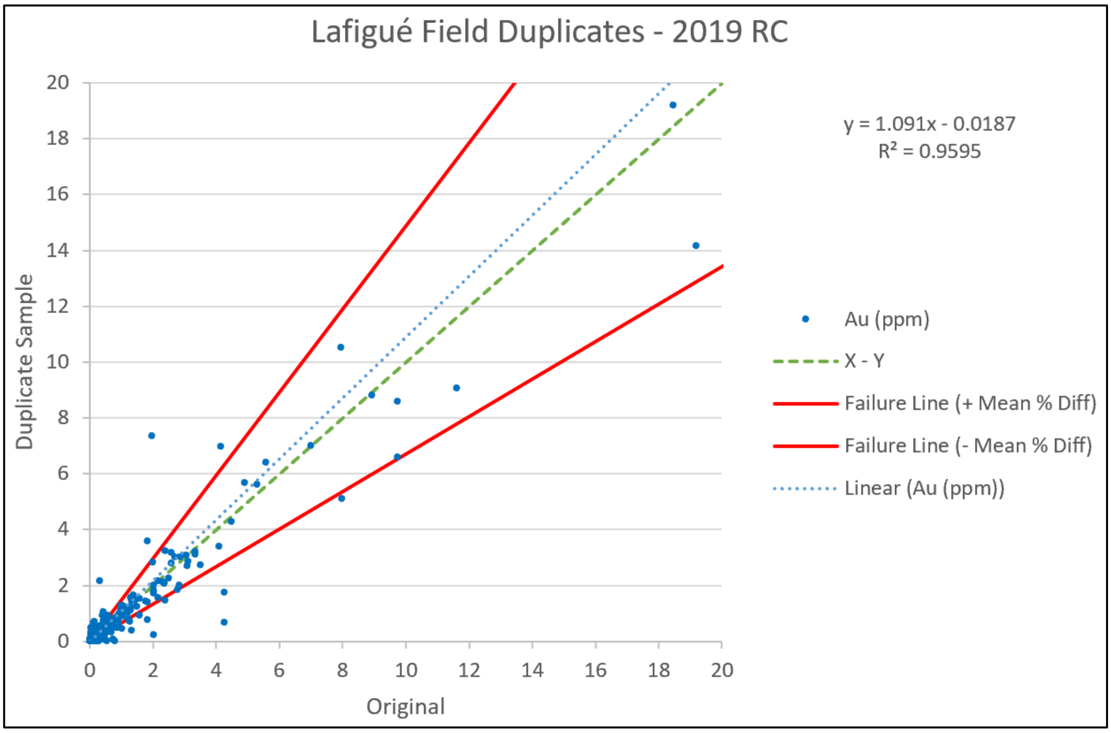


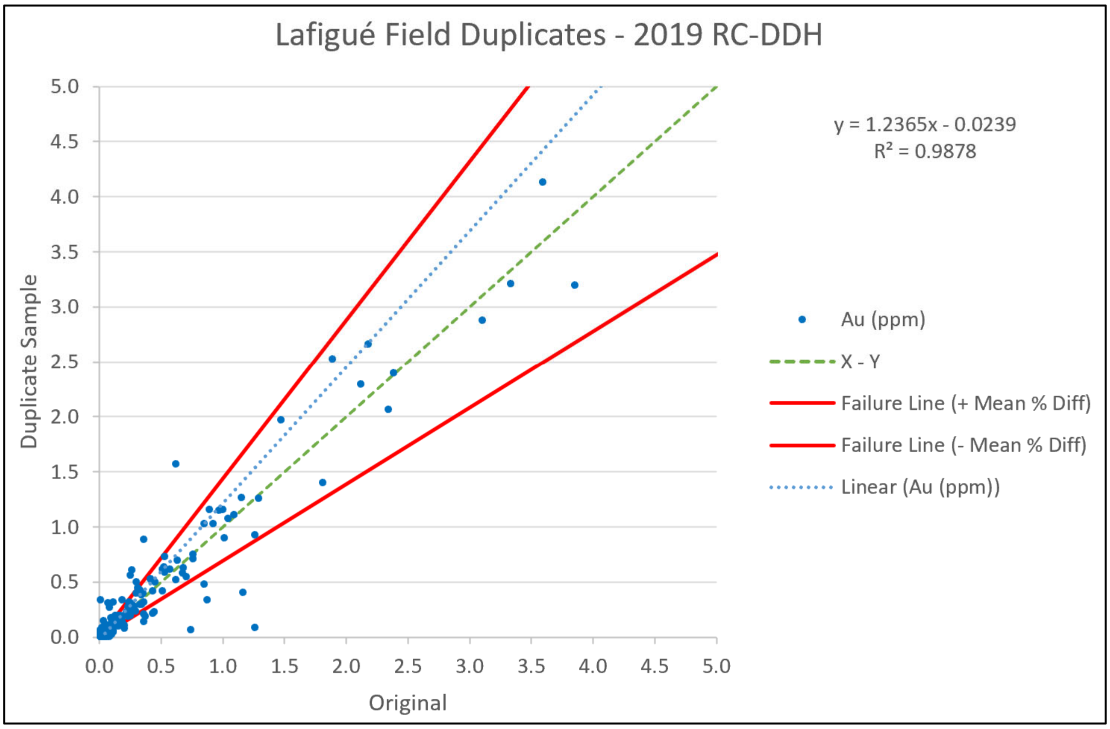



Lafigué Field Duplicates - 2018 RC-DDH
Duplicate Sample
Original
5.00
4.50
4.00
3.50
3.00
2.50
2.00
1.50
1.00
0.50
0.00
0.00 0.50 1.00 1.50 2.00 2.50 3.00 3.50 4.00 4.50 5.00
y = 0.7778x + 0.0548
R² = 0.9891
Au (ppm)
X - Y
Failure Line (+ Mean % Diff)
Failure Line (- Mean % Diff)
Linear (Au (ppm))

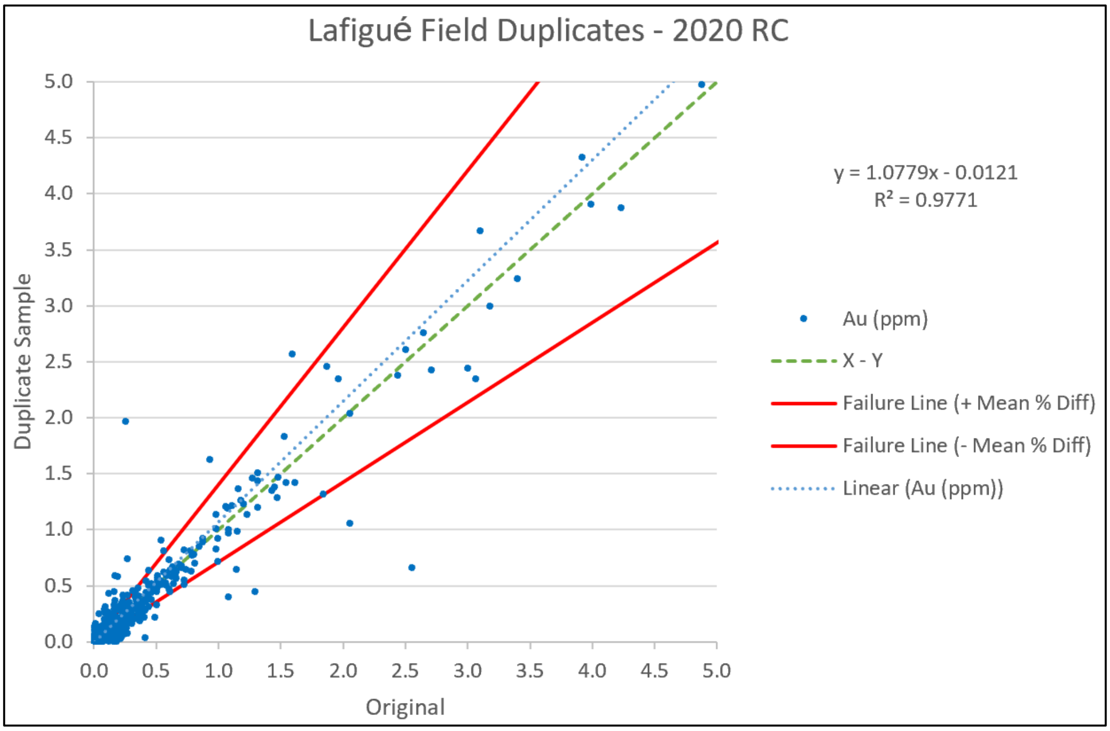
2019 Field Duplicate Plots:

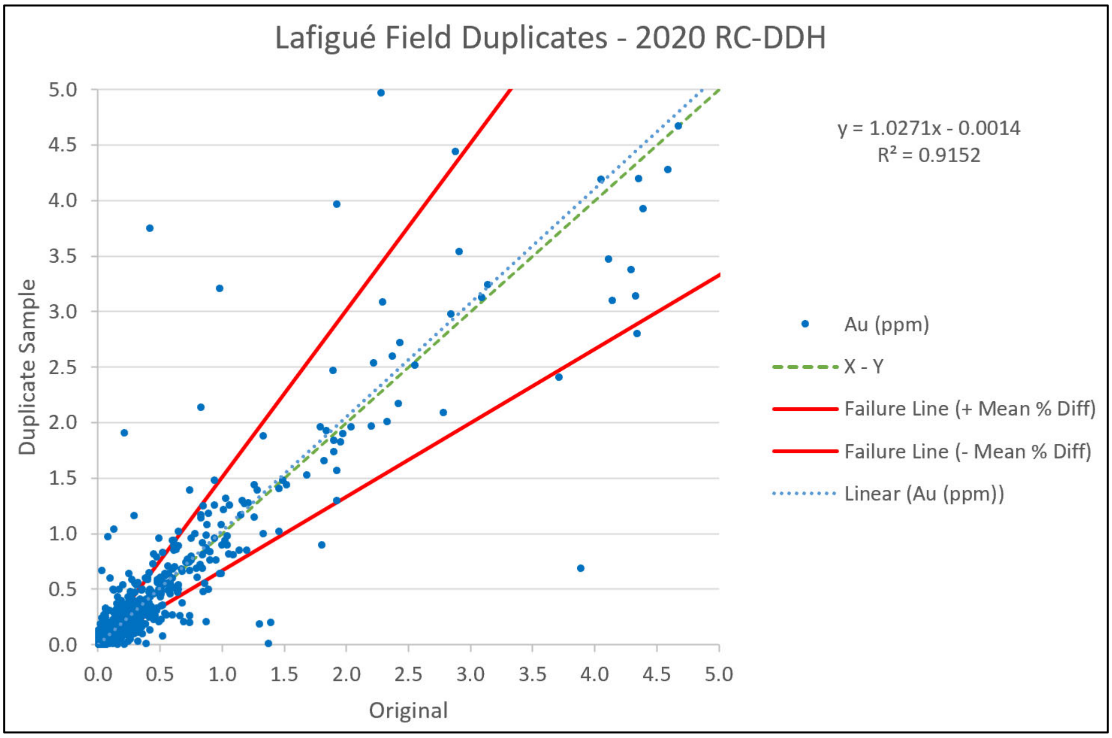




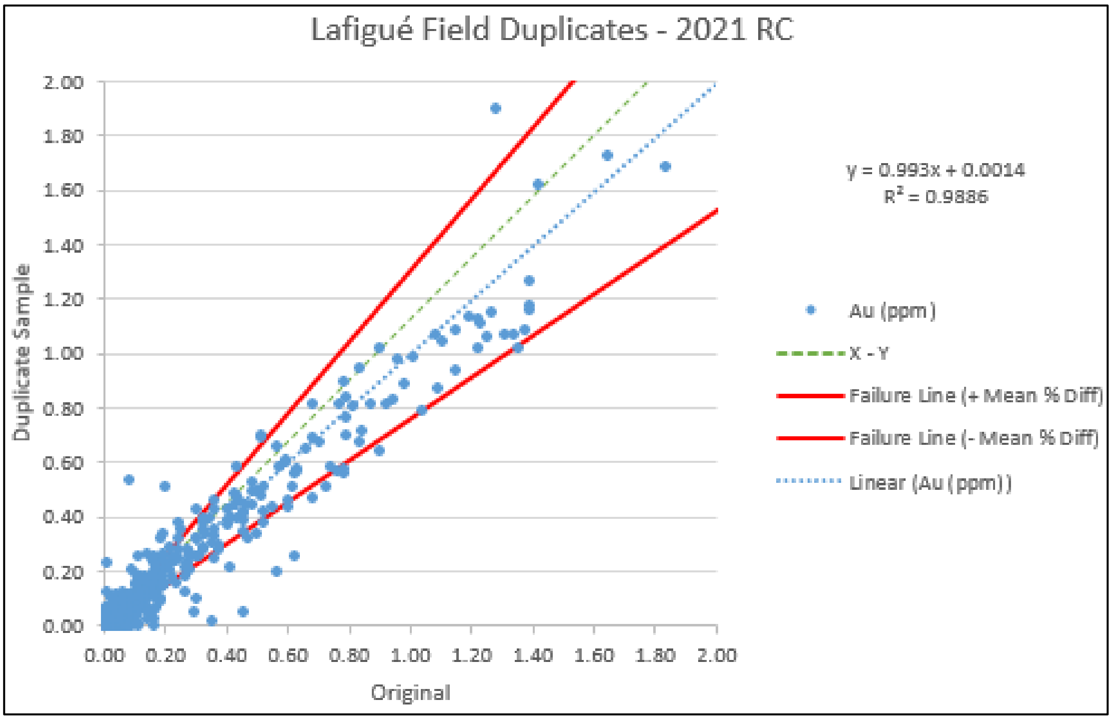

Lafigué Field Duplicates - 2019 RC-DDH
y = 1.2365x - 0.0239
R² = 0.9878
Duplicate Sample
Original
0.0
0.5
1.0
1.5
2.0
2.5
3.0
3.5
4.0
4.5
5.0
Au (ppm)
X - Y
Failure Line (+ Mean % Diff)
Failure Line (- Mean % Diff)
Linear (Au (ppm))

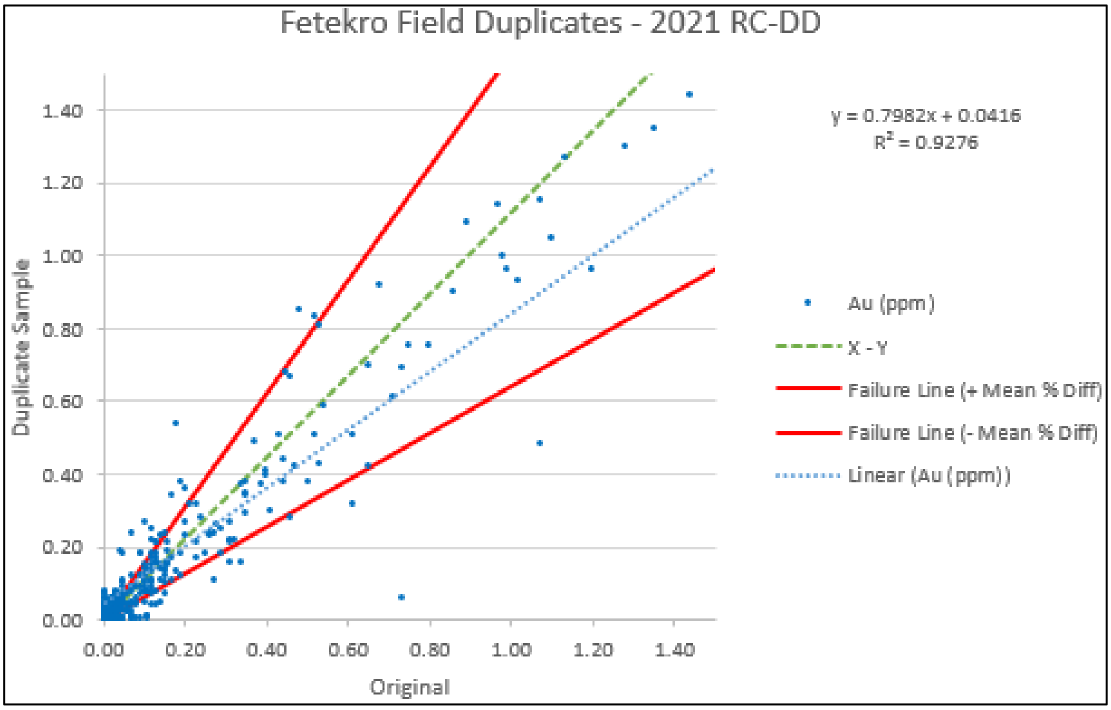
2020 Field Duplicate Plots:



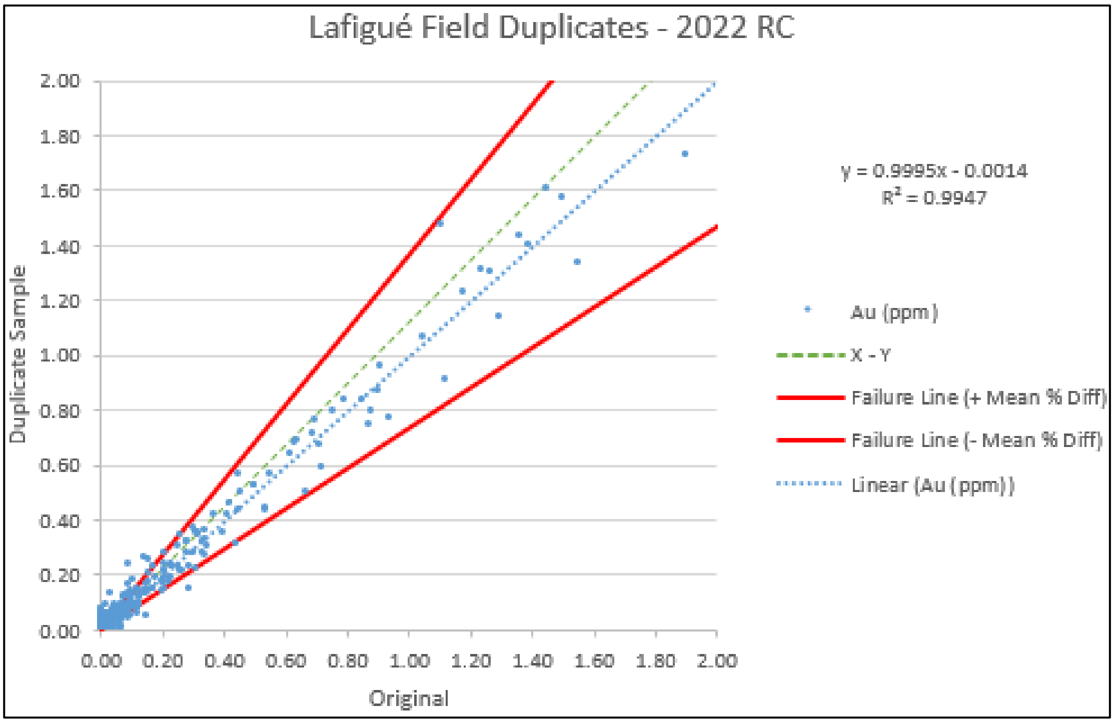


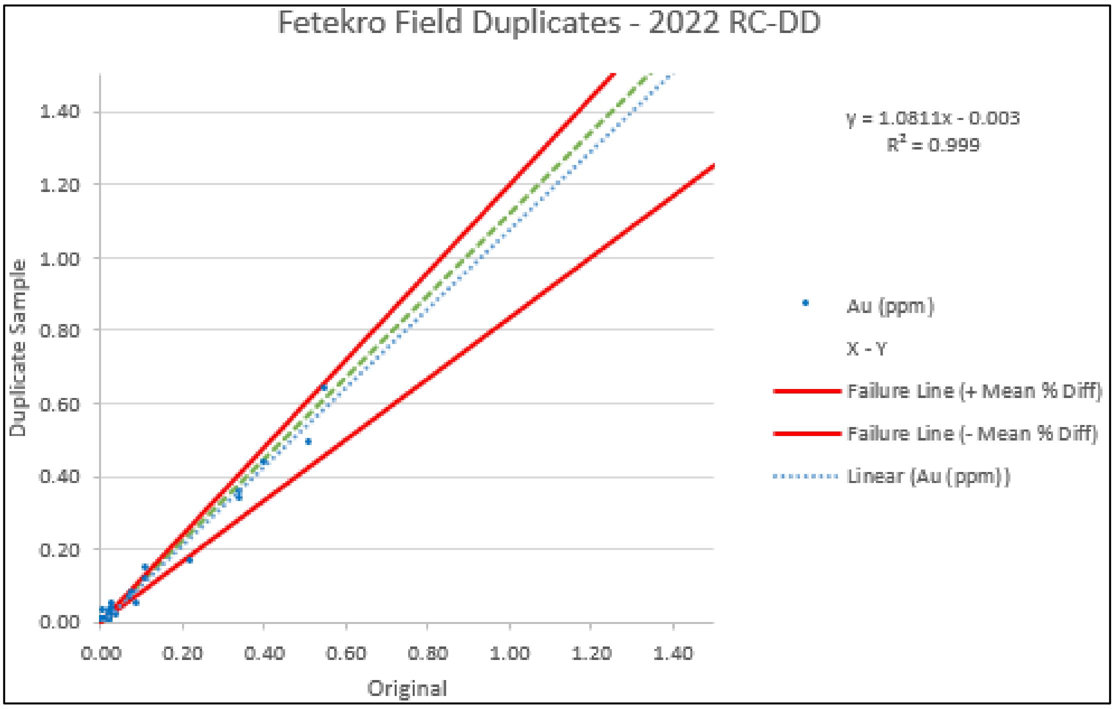
2021 Field Duplicate Plots:





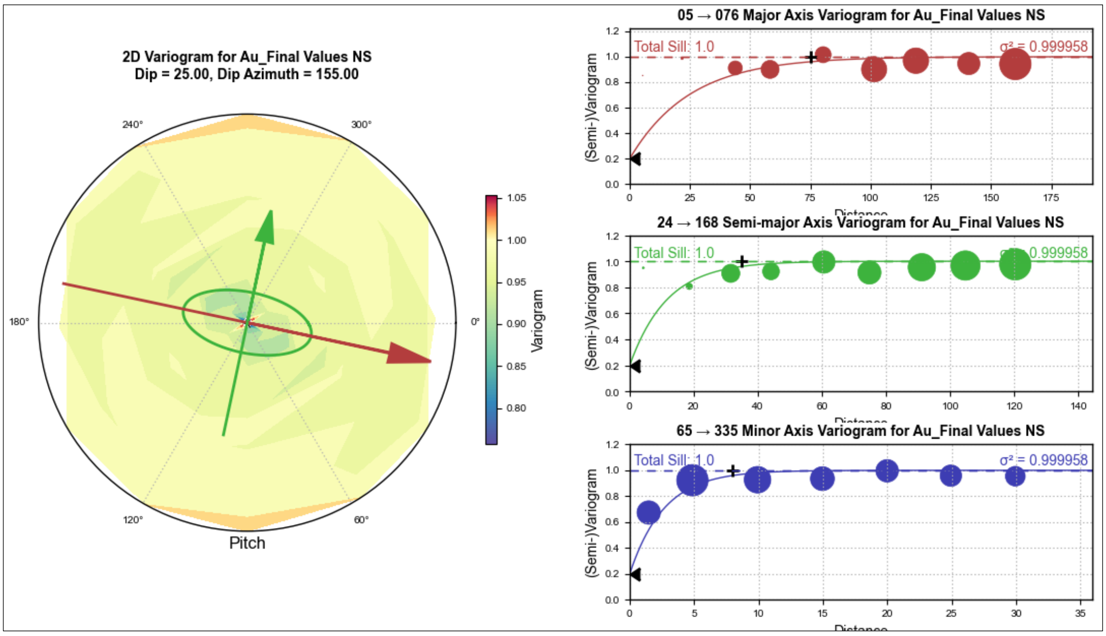
2022 Field Duplicate Plots:



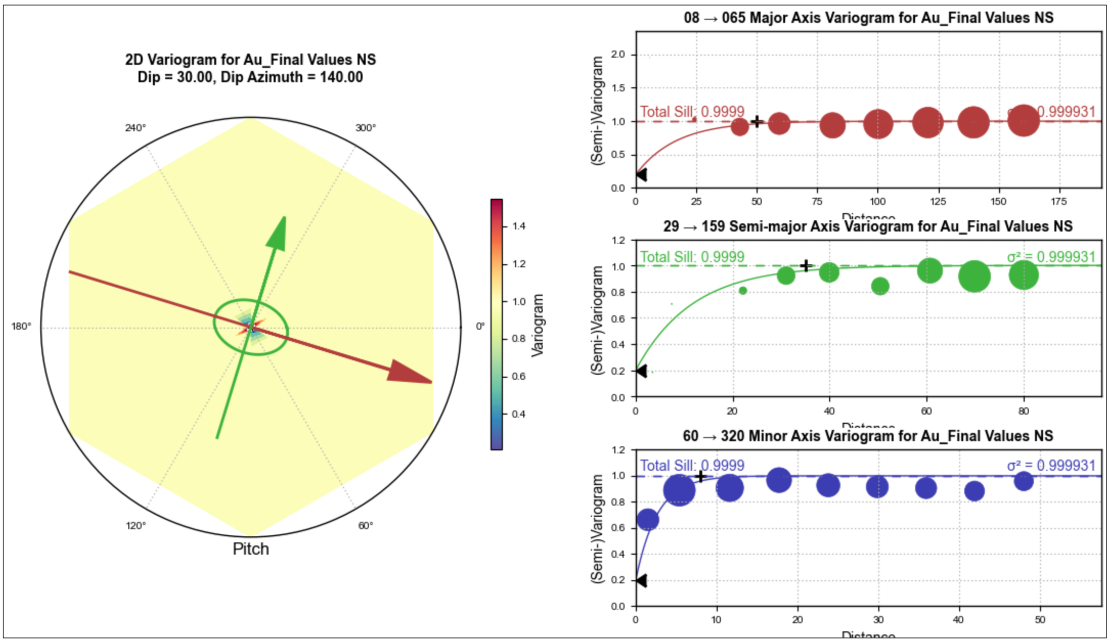


APPENDIX D VARIOGRAMS

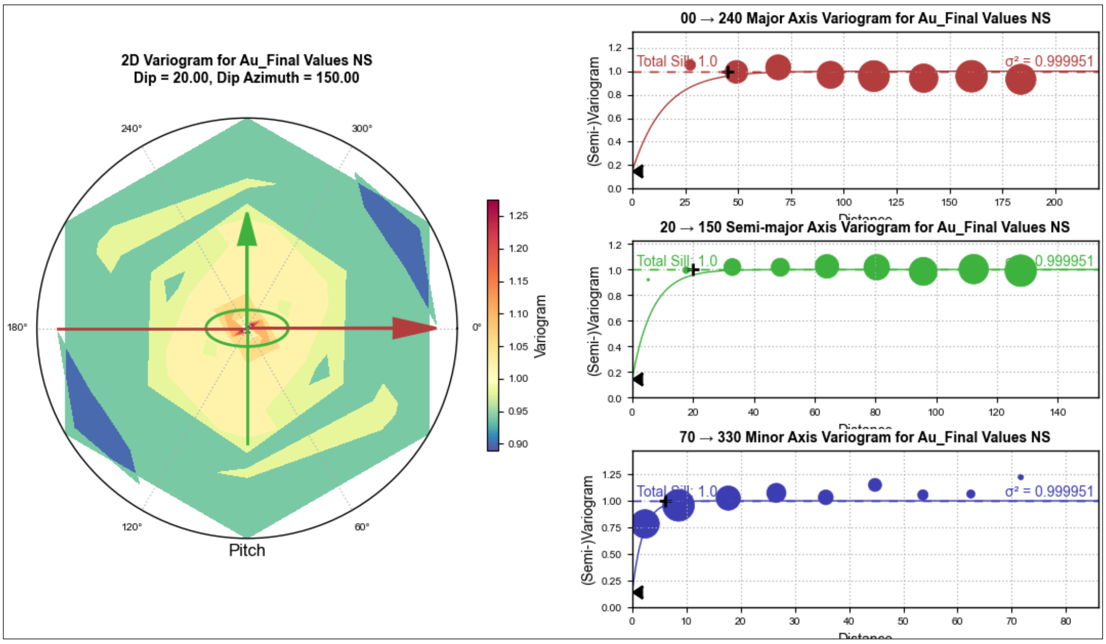
MMZ_Central:



MMZ_East:



MMZ_West:


2D Variogram for Au_Final Values NS
Dip = 20.00, Dip Azimuth = 150.00
Pitch
Variogram
00 → 240 Major Axis Variogram for Au_Final Values NS
20 → 150 Semi-major Axis Variogram for Au_Final Values NS
70 → 330 Minor Axis Variogram for Au_Final Values NS
(Semi-)Variogram
Total Sill: 1.0
σ² = 0.999951

APPENDIX E ARTISANAL MINER EVICTIONS – BACKGROUND INFORMATION

Lafigué ASM Eviction from Main Pits

To:	SRK (UK)
From:	Tony Kneuker
Date:	2 July 2022

Subject: Lafigué MRE – ASM Evictions and Fence Installation

1. BACKGROUND

On 8 June 2022, a document was sent to SRK indicating the progressive eviction of Artisanal or Small-Scale Miners (ASM) from the main Resource pit areas and the relative reduction in activities, after a baseline survey of the ASM workings had been taken in August 2021.

This memo includes the original summary provided by the Société des Mines de Lafigué (SML), which detailed the gendarme special forces (GSLCOI[1]) interventions and relative reduction in ASM activity between August 2021 and May 2022. This original document is appended.

2. ADDITIONAL INFORMATION REQUESTED

SRK (UK) has requested more details on the progress of the perimeter security fence. This is detailed below from the weekly project reports:

- End January 10%
- End February 45%
- End March 71%
- No progress in April (due to issues getting culverts for low-lying areas)
- End May 77%
- 18 June 93%
- 23 June 100%

When the author visited the pit area in April 2022, there was no ASM activity in the pit area.

Attachments:

- Original document detailing intervention of GSLCOI, gendarmes and relative reduction in ASM activity.

Signed:

ENDEAVOUR MINING CO. PROJECT MANAGEMENT OFFICE

Tony Kneuker

Project Director – Société des Mines de Lafigué (SML)

[1] Le Groupement Spécial de Lutte Contre l'Orpaillage Illégal

Statement of artisinal mining activities in the Resource (pit) zones, details of security forces evictions and effectiveness on reducing artisnal activities.
From Deputy Secuity Manager (Andre Rapine) dated 28 May 2022

EVICTIONS AND ILLEGAL GOLD MINING ACTIVITIES

Voluntary evictions

Monday 13/09 Start of the « voluntary eviction » campaign

Friday 17/09 End of « voluntary eviction » of interior sites and a big interior processing site.

Monday 20/09 to 22/09 Intervention with the communities of the Dabakala Gendarmes

Method :

Meeting of the security forces with illegal gold miners and representatives of illegal gold miners, village chiefs, other elected representatives and community representatives

Meeting with the Sureté / SP / Gendarmes of Dabakala with the chiefdom of Lafigue to prohibit the return of illegal gold miners to the permit

Results :

Volume of gold miners evicted from the future fenced area estimated at between 7,000 and 8,000 miners

Numerous gold panners scattered throughout the permit, on smaller, less accessible or more remote gold mining sites

Rodage and incessant attempts by small groups of illegal gold miners to return, followed by a more consistent reoccupation of the main resource area

Before and after photos

GSLCOI clearing

1st GSLCOI intervention from 20 to 23 September 2022

Effectiveness on future pits 90%

Overall final efficiency in the future fenced area 30%.

2nd intervention from 15 to 17 December 2021

Effective efficiency on future pits 99%

Overall final efficiency in the future fenced area 60%.

3rd intervention from 23 to 25 January 2022

Effectiveness on future pits 99%

Overall effectiveness in the future fenced area 98%

Results obtained through a combination of communication with the communities and illegal gold miners, the construction of the fence and the actions of the GSLCOI

Rem :

These data have since stabilised

The main resource area should be considered 100% cleared now.

The very few gold panners caught in the future fenced area should be counted as trespassers

September 2021

January 2022

4th intervention from 27 to 28 April 2022

Intervention on the PE outside the fence and on the PR

5th intervention from 30 April to 03 May

Intervention on the EP outside the fence and on the PR

Anthony Kneuker

ENDEAVOUR MINING CORP. * * PROJECT MANAGEMENT OFFICE * *

Tony Kneuker
Project Director
8 June 2022