Orezone Gold Corporation

Bomboré Phase II Expansion

Burkina Faso, West Africa

Definitive Feasibility Study
National Instrument 43-101 Technical Report

File Number: 5174-GREP-001

Effective Date: 28 March 2023

Signature Date: 24 November 2023

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IMPORTANT NOTICE

This notice is an integral component of the Bomboré Phase II Expansion, Definitive Feasibility Study Technical Report (the Report) and should be read in its entirety and must accompany every copy made of the Report. The Report has been prepared in accordance with the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects published by the Canadian Securities Administrators (NI 43-101). The Report was prepared for Orezone Gold Corporation (the Client) by Lycopodium Minerals Canada Ltd (Lycopodium), P&E Mining Consultants Inc., AMC Mining Consultants (Canada) Ltd., Knight Piésold Pty Limited, and Africa Label Group Inc., (together, the Report Contributors). The Report is based on the expertise and experience of the Report Contributors and information and data supplied to the Report Contributors by the Client and other parties. The quality of information, conclusions, and estimates contained herein are consistent with the level of effort involved in the services of Report Contributors required by NI 43-101, based on: i) information available at the time of preparation of the Report, and ii) the assumptions, conditions, and qualifications set forth in the Report. Each portion of the Report is intended for use by the Client subject to the terms and conditions of its contract with the Report Contributors. Except for the purposes legislated under Canadian provincial and territorial securities law, any other uses of the Report, by any third party, is at that party's sole risk.

1.0 SUMMARY

1.1 Introduction

Orezone Gold Corporation (Orezone) is engaged in mining, developing, and exploring its 90%-owned flagship Bomboré gold mine (Bomboré) in Burkina Faso, West Africa. The Company achieved commercial production at the Bomboré mine on 1 December 2022. The Bomboré oxide process plant was delivered on schedule and under budget and production in Q1 2023 was 41,301 ounces of gold at an all-in sustaining cost of $926/oz sold.

This Technical Report supersedes the 2019 Technical Report and incorporates additional exploration drilling, expanded mineral resources and reserves and a summary of the Feasibility Study for the Bomboré Gold Project Phase II Expansion. The Bomboré mine plan demonstrates a mine life of greater than 11 years with gold production from the existing 5.9 Mtpa oxide plant, plus gold production from a new 4.4 Mtpa hard rock plant.

This Technical Report was compiled by Lycopodium Minerals Canada Ltd (Lycopodium) for Orezone with contributions from Qualified Persons as set out in Table 1.1.1 to support the Company's press release dated 11 October 2023. This Technical Report was prepared in compliance with the disclosure requirements of NI 43-101 and in accordance with the requirements of Form 43-101 F1.

Table 1.1.1 Study Contributors

Contributor	Report Sections
Lycopodium Minerals Canada Ltd. (Lycopodium)	1.1, 1.2, 1.3, 1.6, 1.10, 1.11, 1.13, 1.15, 1.16, 1.17, 2, 3, 4, 5, 6, 13, 17, 18.1 to 18.12, 19, 20, 21.1, 21.2.1, 21.2.3, 21.2.4, 21.3, 22, 24, 25.1, 25.4, 26.1, 26.4, and 27.
P&E Mining Consultants Inc. (P&E)	1.4, 1.5, 1.7, 7, 8, 9, 10, 11, 12, 14, 23, 25.2, 26.2, 26.7, and 27
AMC Mining Consultants (Canada) Ltd. (AMC)	1.8, 1.9, 1.14, 15, 16, 21.2.2, 25.3 and 26.3
Knight Piésold Pty. Ltd. (KP)	18.13, 18.14, 25.5 and 26.5.
Africa Label Group Inc.	1.12, 20, 25.6, 26.6

1.2 Property Description and Ownership

The Bomboré Gold Mine property (Property) is located in Ganzourgou Province, Burkina Faso, approximately 85 km east of the capital city of Ouagadougou (Figure 1.2.1). The Property is easily accessible by the paved road, national highway N4 from Ouagadougou. The Property entrance is accessed from highway N4 via a 5 km lateritic road which is maintained by Orezone.

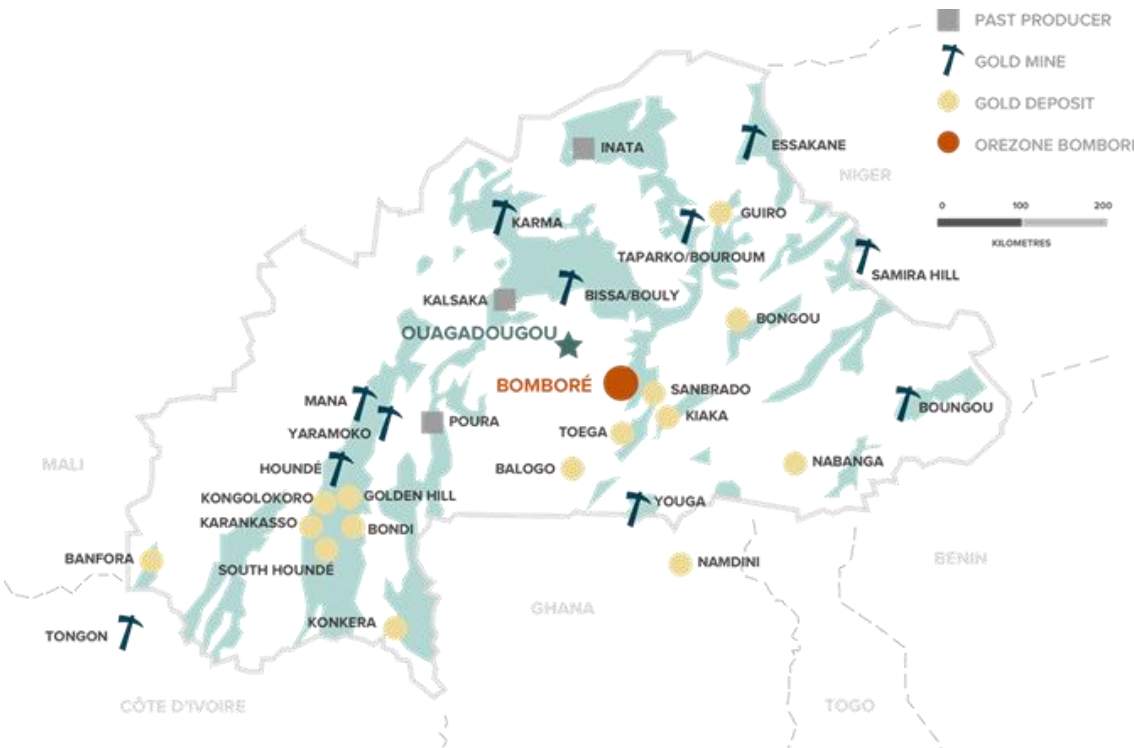
The Property is within 15 km of the regional town of Mogtédo, with a population greater than 15,000. The town is developing rapidly with many substantial multi-storey concrete block buildings established or under construction. Most of the semi-skilled and unskilled labour required for project development is sourced from Mogtédo and surrounding communities.

The local climate consists of dry and wet seasons. It is common for rain to occur from April through October. However, the highest concentration of rainfall events occurs between late June and late September. On average, approximately 800 mm of rainfall occurs annually, typically in short bursts of heavy rain. Construction and mining operations can be scheduled year-round, with short delays during heavy rainfall events expected.

Temperatures range from a low of about 10°C in December and January to highs of about 43°C in March and April with average daily temperatures in the range of 23° to 33°. Between the end of the wet season and March the north-easterly trade winds bring dust down from the Sahara (the Harmattan) resulting in reduced visibility.

The Universal Transverse Mercator (UTM) co-ordinates for the approximate centre of the Property are 1,348,200mN, 664,900mE (Zone 30, ITRF 2008 datum). The geographic co-ordinates for the approximate centroid of the currently defined Bomboré gold deposit are 12°12’N Latitude and 0°12’W Longitude.

Figure 1.2.1 Project Location

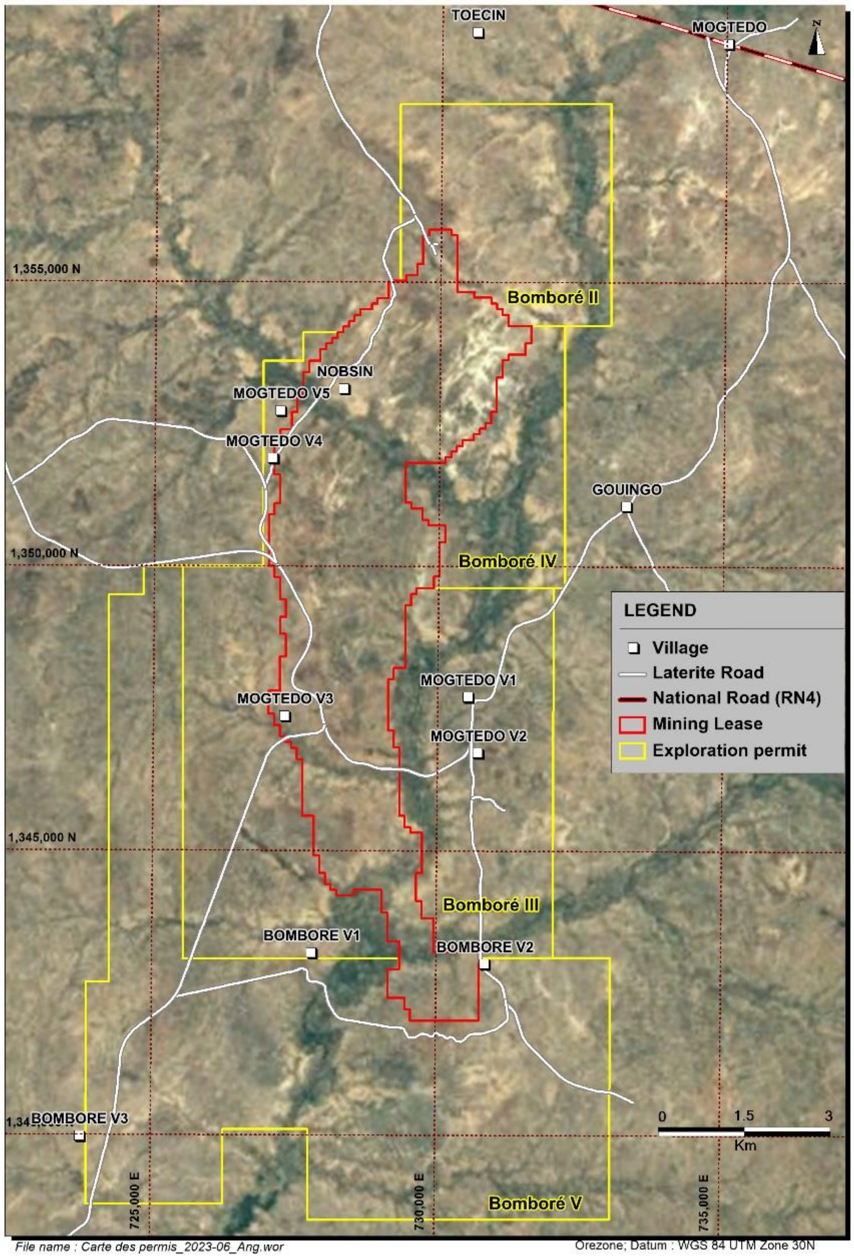


The Property covers an area of 12,963 ha and consists of one Industrial Operating Permit (the Bomboré Mining Permit) of 2,887 ha, surrounded by four Mining Exploration Permits: the Bomboré II Exploration Permit of 1,265 ha, the Bomboré III Exploration Permit of 3,360 ha, the Bomboré IV Exploration Permit of 833 ha and the Bomboré V permit of 4,618 ha. (Figure 1.2.2)

The Bomboré Mining Permit is registered in the name of Orezone Bomboré S.A. (OBSA), a 90%-owned subsidiary of Orezone Inc. S.A.R.L, itself a 100%-owned subsidiary of Orezone Inc., which is 100% owned by Orezone. The Bomboré Mining Permit was granted to OBSA by way of Decree No. 2016-1266/PRES/PM/MEMC/MINEFID/MEEVCC dated 30 December 2016 and is valid for an initial tenure of 10.7 years but can be extended if the mine life is extended beyond what was initially applied for.

All mining ventures in Burkina Faso are subject to a 10% free carried interest and a royalty on gold sold in favour of the Government of Burkina Faso, upon the award of an operating permit from the government.

Figure 1.2.2 Bomboré Tenements



1.3 History

The Property was originally covered by a prospecting authorization covering 605,800 ha, granted to Générale de Mines et de Carrières (GMC) in 1989. In January 1994, following changes in the Mining Act in 1993, a modified exploration permit covering 210,800 ha was issued to GMC.

Channel Mining (Barbados) Company, Ltd. (Channel) entered into an option agreement with GMC in 1994 giving it a 90% working interest in the exploration permit, leaving GMC with a 10% carried interest. In the summer of 1997, GMC converted its 10% interest into Channel shares.

Orezone's rights to the Property arise from an initial option agreement signed in 2002 by Orezone's predecessor Orezone Resources Inc. (ORINC) with Channel and Solomon granting ORINC the right to earn a 50% interest in Bomboré. In 2004, the original Bomboré exploration permit expired and a new Bomboré I exploration permit covering 25,000 ha was granted to Société Orezone, a subsidiary of ORINC, by the MEMC on 17 February 2004.

ORINC earned its 50% interest in Bomboré by issuing 150,000 common shares, making a C$40,000 payment, and spending C$2 million on exploration before 17 January 2007. The Bomboré I exploration permit was renewed on 14 May 2007 and reduced to 10,450 ha. On 3 September 2008, ORINC announced that it had purchased the remaining interest in the Bomboré I exploration permit from Channel and Solomon in consideration of one million common shares of ORINC.

In 2007, ORINC commissioned Met-Chem Canada Inc. (Met-Chem) to prepare an initial mineral resource statement. This mineral resource estimate considered drilling information to March 2007 and is documented in a technical report prepared by Met-Chem and dated 28 February 2008.

On 25 February 2009, ORINC and IAMGOLD Corporation (IAMGOLD) announced that IAMGOLD had acquired ORINC pursuant to a plan of arrangement under the *Canada Business Corporations Act*. As part of the plan of arrangement, a new exploration company, Orezone Gold Corporation, was incorporated and acquired certain assets and liabilities of ORINC, including the Bomboré I exploration permit. There was no further relationship with IAMGOLD after the transaction closed other than IAMGOLD becoming a shareholder of Orezone.

RPA completed an updated resource model in December 2018 incorporating previously excluded seasonal floodplains and all drilling completed to that date on the high-grade P17S deposit. The effective date of this updated resource estimate remained 5 January 2017, as the changes were marginal.

In 2019, Orezone commissioned P&E Mining Consultants Inc. to update the mineral resource model for the Property with assistance of Orezone for the geological and domain modelling. This mineral resource estimate was presented in the Technical Report prepared by Lycopodium Minerals Canada Ltd. with an effective date of 26 June 2019.

In January 2021 the Company announced it had secured a financing package totalling $182M and awarded the engineering, procurement, and construction management (EPCM) contract for the process plant construction and commissioning to Lycopodium Minerals Pty Ltd. (Lycopodium).

The Company declared commercial production at Bomboré on 1 December 2022. Delivery of the Bomboré project construction (excluding the third-party managed power plant) was completed on schedule and under budget. Final project construction costs including pre-production mining but excluding power plant totalled US$168.9M, below the project approved budget of US$173.8M.

1.4 Geology and Mineralization

The Property covers part of a northeast-southwest trending greenstone belt that extends for 50 km from the southwest to the northeast. The Property area is underlain mainly by a metasedimentary flysch-type sequence dominated by meta-sandstones with subordinate carbonaceous meta-pelites and polymictic meta-conglomerates. This metasedimentary sequence is intruded by early meta-gabbro and ultramafic intrusions, and then syntectonic granodiorite intrusions. Late-tectonic quartz-feldspar porphyries occur as dikes and larger bodies within the greenstone belt. Large biotite granite intrusions are present on the Property to the west and to the south of the greenstone belt that is also moulded on a large quartz diorite intrusion located along the eastern limit of the Property. A syenite intrusion referred to as the Petite Suisse outcrops in the west portion of the Property.

The Bomboré Shear Zone (BSZ) is a major, 1- to 3-km thick structure that contains the Bomboré gold mineralization and represents the dominant structural feature in the area. The Bomboré gold mineralization trend is defined by a gold-in-soil anomaly exceeding 0.1 g/t Au, and by the presence of numerous gold showings and orpaillage (artisanal miners) sites. The Bomboré Au anomaly measures 14 km long, several hundred metres across, and occurs within the BSZ.

Surface weathering has affected the rocks to an average depth of 35 m to 50 m, but can be as deep as 100 m on the hanging wall of the P8/P9 and CFU Deposits, and as shallow as 5 m to 10 m in the area of the P17 Deposit.

The gold mineralization on the Property is hosted in the BSZ, the major north-northwest to north-northeast trending structure. This shear zone has an arcuate shape and extends over tens of km beyond the limits of the Property. It is interpreted to be a second-order structure to the Tiébélé-Dori-Markoye Fault, a regional first-order and north-northeast trending sinistral fault that represents a major discontinuity in the Birimian rocks, across which regions of contrasting structural styles are juxtaposed.

Gold occurs generally as fine-grained electrum (< 10 µm), but visible gold has been observed in outcrop. Artisanal mining over the 1990 to 2016 period attests to the existence of coarser gold locally. Gold occurs as free gold and is mainly associated with pyrite, pyrrhotite, chalcopyrite, and arsenopyrite. Most of thesulphides occur as disseminations and fine stringers sub-parallel to the foliation fabric, which suggests development in active shear zone or re-mobilization. Magnetite and graphite are present locally. Although the sulphide content can be as much as 5%, it is on average only 1% to 2% in non-weathered mineralized rocks.

Gold mineralization is most commonly hosted in the biotite schist (meta-gabbro), the surrounding meta-sandstones, and the granodiorite dikes that intrude the gabbros. However, meta-argillites are the main host rocks at the Maga North, P16 and P17N Deposits. The syn-tectonic granodiorite intrusions are also mineralized, although to a smaller extent than the biotite schist and meta-argillites. Conversely, the meta-conglomerate and metaperidotite are unfavourable host rocks. The meta-gabbro might represent the best chemical trap, given its high iron content if gold was transported as a thio-complex, as suggested by its association with pervasive fine-grained pyrite in the sulphide zone. Although much of the gold Mineral Resources defined within the Property area are hosted in the meta-gabbro unit, the deformed granodiorite and its contact zone with the meta-gabbro host is where the higher-grade mineralization is concentrated.

At a cut-off grade of 0.2 g/t Au, the gold mineralization generally exhibits reasonable continuity over a strike length of approximately 10 km. In detail, the gold mineralization forms more restricted corridors 500 m to 1,000 m long and 10 m to 100 m across that define anastomosing patterns parallel and slightly oblique to the general trend of the BSZ. These higher-grade corridors formed the basis for defining geostatistical domains within each litho-domain considered for mineral resource estimation. One of the benefits of the 2010 to 2013 infill drilling programs was the delineation of higher-grade sub-domains based on a cut-off grade of 0.5 g/t Au with the broader low-grade domains based on a lower cut-off grade of 0.2 g/t Au. The higher-grade sub-domains are up to 500 m long and between 5 m and 30 m thick.

1.5 Exploration and Drilling

Exploration in the Property area commenced in 1989. Between 1989 and 2000, mineral exploration programs were completed by La Générale des Mines et des Carrières (GMC), Channel, Solomon, and Placer Dome. A total of 1,271 core, reverse circulation (RC) and rotary air blast (RAB) boreholes were completed, and many geochemical, geophysical, and trenching surveys were carried out.

Between 1994 and 2000, Channel completed 10 diamond boreholes for approximately 1,100 m, 261 RC boreholes for approximately 20,000 m, and 1,000 RAB boreholes for approximately 34,000 m on the Property. However, there are no records describing the procedures used by Channel in their drilling programs.

Since acquisition of the Property in 2003 to April 30, 2023, Orezone has completed systematic mapping, prospecting, sampling, and gold assaying of outcrops and gold workings. Several airborne and ground magnetic and induced polarization / resistivity surveys and core, RC and auger drilling campaigns have also been completed. Between 2003 and 2023, Orezone completed 1,485 core boreholes for approximately 233,000 m, 6,538 RC boreholes for approximately 375,000 m, and 4,221 auger holes for approximately 23,000 m.

1.6 Mineral Processing and Metallurgical Testing

Extensive testwork programs have been carried out at different laboratories for Bomboré with the first test program started in 1997 and the latest completed in 2023. The test programs were conducted on drill core composites, RC cuttings, and RAB drill samples considered representative of the ore deposit at the time of each test program. A summary list of the programs is included in Table 1.6.1.

Table 1.6.1 Summary of Testwork Programs

Program	Leachwell Recoveries	Head Analysis	Variability	Cyanidation	Gravity	Flotation	Carbon-in-Leach (CIL)	Carbon Adsorption & Equilib.	Column Leach (HL)	Comminution	Scrubbing	Gold Deportment	Petrography	Thickening / Rheology	Neutralization	Lime Demand	Acid Mine Drainage
SGS / ITS 1997			✓	✓									✓				
Osborne 2008			✓	✓													
AMMTEC 2009		✓	✓	✓	✓	✓	✓		✓	✓							✓
McClelland 2012		✓	✓	✓	✓	✓	✓			✓			✓	✓	✓		
Phillips 2012										✓							
OREZONE Scrubbing 2012			✓	✓							✓	✓					
Met-Solve 2013											✓	✓					
SGS Lakefield 2013										✓							
COREM 2013				✓						✓			✓				
Met-Solve 2014				✓			✓				✓						
Consolidated Database 2013	✓																
Kappes 2014			✓	✓			✓		✓		✓	✓		✓	✓		
SGS Lakefield 2014										✓							
SGS Lakefield 2016				✓	✓	✓				✓			✓				
SGS Lakefield 2017/2018			✓	✓						✓						✓	
Outotec 2018														✓			
Base Metallurgical Lab 2019		✓	✓	✓						✓			✓	✓			
SGS Lakefield 2019								✓									
Maelgwyn 2023		✓	✓	✓	✓		✓							✓			

A summary of the metallurgical inputs to the oxide plant and hard rock plant process design criteria, derived from the interpretation of the testwork, are presented in Table 1.6.2 and Table 1.6.3 respectively.

Table 1.6.2 Summary of Metallurgical Criteria for Oxide Plant

Criteria	Units	Design	Notes / Source
Plant Throughput	tpa	5,900,000	Orezone
Ore Type	-	Oxide Upper Transition	Mine plan
Design Ore Blend - Oxide - Upper Transition	% %	67 33	Mine plan Mine plan
Head Grade - Gold (Design) - Gold (LOM average)	g/t Au g/t Au	1.0 0.54	Lycopodium/Orezone Mine plan
Gold Recovery Estimation at 1 g Au/t - Upper Oxide - Lower Oxide - Upper Transition - Per Design Ore Blend	 % % % %	 91.8 91.8 89.0 90.9	 Orezone Orezone Orezone Calculated
Ore Specific Density Ore Bulk Density Crushing Work Index (CWi) Rod Mill Work Index (RWi) Bond Ball Mill Work Index (BWi) Bond Abrasion Index (Ai)	t/m^3 t/m^3 kWh/t kWh/t kWh/t g	2.8 1.65 7.7 5.8 4.8 0.031	Testwork Lycopodium/Orezone Testwork Testwork Testwork Testwork
Grind Size P_{80} CIL Circuit Residence Time CIL Slurry Density (for saprolitic ore)	µm hrs % solids	125 24 ~40%	Lycopodium Testwork Lycopodium
Sodium Cyanide Addition	kg/t NaCN	0.28	Current performance
Quicklime Addition	kg/t CaO	1.68	Current performance

Table 1.6.3 Summary of Metallurgical Criteria for Hard Rock Plant

Criteria	Units	Design	Notes/Source
Plant Throughput	tpa	4,400,000	Orezone
Ore Type	-	Lower Transition Fresh Rock	Mine plan
Design Ore Blend - Lower Transition	%	22	Mine plan
- Fresh	%	78	Mine plan
Head Grade - Gold (Design)	g/t Au	1.25	Lycopodium / Orezone
- Gold (LOM average)	g/t Au	1.02	Mine plan
Gold Recovery at 1.25 g/t Au			
- Lower Transition	%	86.0	Orezone
- Fresh Rock	%	81.7	Testwork
- Pit P8P9 and CFU Fresh	%	84.0	Testwork
- Pit P17S	%	95.0	Testwork
- Per Design Ore Blend	%	85.0	Calculated
Ore Specific Density	t/m^3	2.8	Testwork
Ore Bulk Density	t/m^3	1.65	Lycopodium / Orezone
Crushing Work Index (CWi)	kWh/t	19.8	Testwork
Rod Mill work Index (RWi)	kWh/t	17.1	Testwork
Bond Ball Mill Work Index (BWi)	kWh/t	16.9	Testwork
A x b Parameter		27.0	Testwork
Bond Abrasion Index (Ai)	g	0.258	Testwork
Grind Size P_{80}	µm	75	Testwork
CIL Circuit Residence Time	hrs	24	Testwork
CIL Slurry Density	% solids	~50%	Lycopodium
Thickener Solids Loading	m^2/tph	1.00	Testwork
Sodium Cyanide Addition	kg/t NaCN	0.63	Testwork / Calculated
Quicklime Addition	kg/t CaO	0.98	Testwork / Calculated

The following conclusions can be drawn from the metallurgical testwork:

- Oxide, transition, and fresh rock ores at Bomboré are readily amenable to CIL whole ore cyanidation. Gold recovery is expected to be 91.8% for the oxide ore and 89% for the upper transition ore based on current plant data. Metallurgical testwork has shown that gold recovery for the fresh ore is 81.7% apart from the P17S and P8P9 pits where fresh ore gold recoveries are 95% and 84% respectively.

- Optimum grind size for the oxide plant was determined to be a P_{80} of 125 µm based on grind size and recovery relationship. Optimum grind size for the hard rock plant was selected to be a P_{80} of 75 µm based on grind size and recovery relationship. The fresh ore is sensitive to grind size and a grind P_{80} of 106 µm will result in a drop of approximately 5% in gold recovery.

- Leach extraction rates are essentially complete within 24 hours based on the observed leach kinetics for both oxide and hard rock and oxygen addition is beneficial for fresh ore leaching.

- Cyanide consumption rates are expected to be moderate at 0.28 kg/t NaCN for the oxide ore and 0.63 kg/t NaCN for the fresh ore.

- Lime consumption rates are expected to be moderate, averaging 1.68 kg/t CaO for the oxide ore and 0.98 kg/t CaO for the fresh ore.

1.7 Mineral Resource Estimate

The updated Mineral Resource Estimate, shown in Table 1.7.1, was completed by P&E with an effective date of March 28, 2023. Mineral resources are reported within an optimized pit shell at a gold price of $1,700/troy oz. The mineral resource estimate, before taking into account stockpiles of unprocessed material, contains 4.5 million Measured and Indicated gold ounces and 0.6 million Inferred gold ounces.

Table 1.7.1 Bomboré Mineral Resource Estimate Summary

	Measured			Indicated			Measured and Indicated			Inferred		
	Tonnes	Grade	Ounces	Tonnes	Grade	Ounces	Tonnes	Grade	Ounces	Tonnes	Grade	Ounces
	Mt	Au g/t	Au koz	Mt	Au g/t	Au koz	Mt	Au g/t	Au koz	Mt	Au g/t	Au koz
Oxide	16.4	0.59	303	72.9	0.56	1,311	89.3	0.57	1,623	3.3	0.57	60
Hard Rock	11.1	1.09	389	78.8	0.99	2,503	89.9	1.00	2,892	16.7	1.02	549
Total	**27.5**	**0.79**	**701**	**151.7**	**0.78**	**3,814**	**179.3**	**0.78**	**4,515**	**20.0**	**0.95**	**610**

Notes

1. “Oxide” includes Regolith, Oxide and Transitional Upper units reported at a cut-off of 0.25 g/t Au.
2. “Hard Rock” includes Transitional Lower and Fresh units reported at a cut-off of 0.45 g/t Au.

3. Mineral resources, which are not mineral reserves, do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues.
4. Mineral resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.
5. The inferred mineral resource in this estimate has a lower level of confidence than that applied to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of the inferred mineral resource could be upgraded to an indicated mineral resource with continued exploration.
6. Totals may differ due to rounding.
7. Mineral resources are reported within an optimized pit shell at a gold price of $1,700/troy oz.
8. Mineral resources are inclusive of mineral reserves, however, exclude ore stockpiles.
9. The mineral resource estimates include oxide grade reduction factors applied by Orezone based on recent mine to mill reconciliation data.

The Bomboré Property encompasses seven zones: B1, B2, P11, Siga, P16, P17N and P17. A total of 378 individual mineralization domains have been incorporated within the updated Mineral Resource Estimate. Geological wireframe models for each of the seven zones were created by Orezone, and then audited by the Authors.

Most of these models overlap with the neighbouring models, but the Mineral Resources reported from each model in this Technical Report are restricted to reporting limits that are complementary at the Property scale (Figure 1.7.1). Block models for P17, B1 and B2 were developed by Orezone and then audited by the Authors. Block models for the P11, Siga, P16 and P17N Zones were developed by the Authors, and they also generated the USD$1,700/oz gold pit shells constraining the current Mineral Resource Estimates.

Block grades for gold were estimated for each mineralization domain by Inverse Distance Cubed (ID3) weighting of capped composites using a minimum of three, four or five composites and a maximum of twelve composites. Sample selection was restricted to a maximum of four or five composite samples from a single drill hole. The orientation of the search ellipsoids within each individual mineralization domain were defined by Orezone geologists based on the local geology. Ordinary Kriging (OK) estimates were also developed for comparison, and Nearest Neighbour (NN) models were used for validation. The ID3 methodology was selected for Mineral Resource reporting, since many of the variograms developed for the mineralization domains were not of sufficient quality for use with OK. Issues associated with the variogram modelling included high nugget contributions, pronounced drill hole effects associated with different lag distances, and multiple directions of anisotropy.

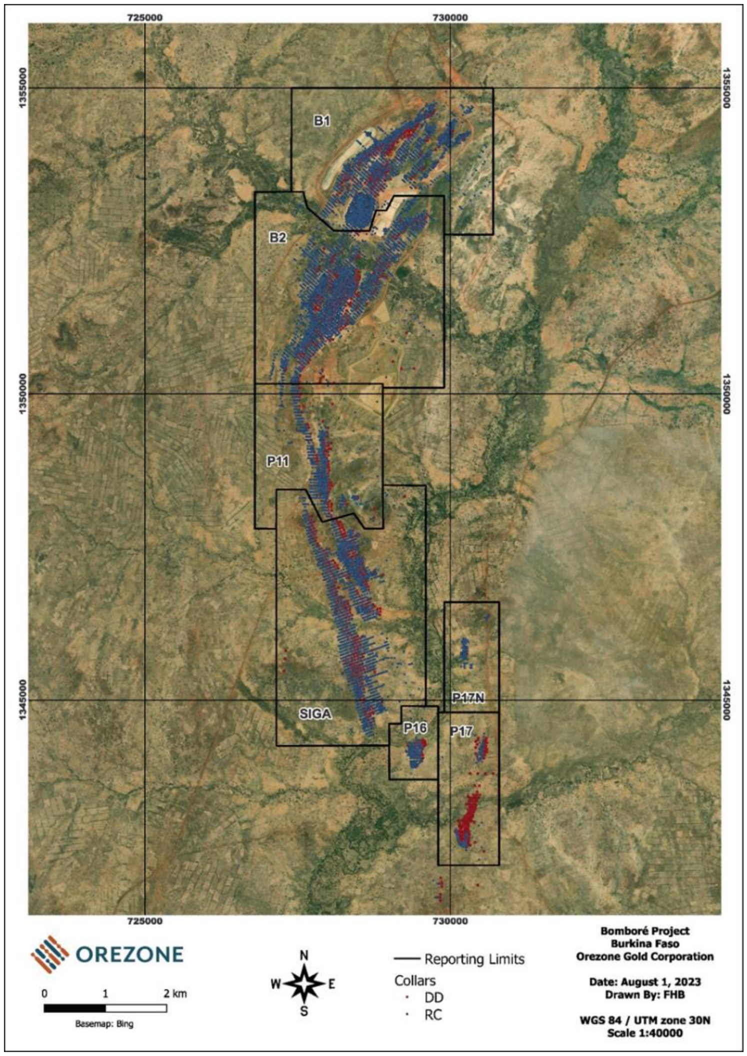
In general, estimated blocks within 25 m of three or more drill holes are classified as Measured, blocks within 50 m of three or more drill holes are classified as Indicated, and additional estimated blocks are classified as Inferred. In some cases, peripheral blocks within the defined veins are classified as Exploration Potential and are not included in the Mineral Resource Estimate.

Mineral Resources are reported within optimized pit shells at the appropriately selected cut-off (Table 1.7.2). Orezone reports whole block volumes using only those blocks where the block centroid lies within the controlling wireframe. A factor has been applied to the oxide Mineral Resource Estimates to discount artisanal mining.

Table 1.7.2 Cut-Off Grades (Au g/t)

Provisional Mineral Resource Estimate							
Unit	**B1**	**B2**	**P11**	**Siga**	**P16**	**P17N**	**P17**
Regolith	0.25	0.25	0.25	0.25	0.25	0.25	0.25
Oxide	0.25	0.25	0.25	0.25	0.25	0.25	0.25
Upper Transition	0.25	0.25	0.25	0.25	0.25	0.25	0.25
Lower Transition	0.45	0.45	0.45	0.45	0.45	0.45	0.45
Fresh	0.45	0.45	0.45	0.45	0.45	0.45	0.45

Figure 1.7.1 Bomboré Mineral Resource Estimate Zones



1.8 Mineral Reserve Estimate

Mineral Resources and Mineral Reserves are reported in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101). CIM (CIM, 2014) definitions were followed for Mineral Reserves.

The Bomboré Mineral Reserves are estimated to contain 103.5 Mt at a grade of 0.72 g/t Au containing 2,403 koz Au. Mineral Reserves are composed of open pit Mineral Reserves of 95.7 Mt at an average grade of 0.75 g/t Au containing 2,301 koz Au and oxide stockpiles of 7.9 Mt at an average grade of 0.40 g/t Au containing 102 koz Au. The Mineral Reserves are summarized in Table 1.8.1.

Table 1.8.1 Bomboré Mineral Reserve Estimate as of 28 March 2023

Classification	Proven			Probable			Proven & Probable		
	Tonnes Mt	Gold grade g/t Au	Contained gold koz Au	Tonnes Mt	Gold grade g/t Au	Contained gold koz Au	Tonnes Mt	Gold grade g/t Au	Contained gold koz Au
Material type									
Oxide	6.2	0.62	124	50.5	0.55	897	56.7	0.56	1,020
Hard Rock	3.3	1.29	137	35.6	1.00	1,144	38.9	1.02	1,281
Total open pit	9.5	0.86	261	86.2	0.74	2,041	95.7	0.75	2,301
Oxide stockpiles				7.9	0.40	102	7.9	0.40	102
Total	**9.5**	**0.86**	**261**	**94.0**	**0.71**	**2,143**	**103.5**	**0.72**	**2,403**

Notes:

1. CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) were used for reporting of Mineral Reserves.
2. Mineral Reserves are estimated using a long-term gold price of $1,500 per troy oz for all mining areas.
3. Mineral Reserves are stated in terms of delivered tonnes and grade before process recovery.
4. "Oxide" includes Regolith, Oxide, and Upper Transition material. Hard Rock includes Lower Transition and Fresh material.
5. Mineral Reserves are based on modified re-blocked mine models with variable internal dilution and mining recoveries.
6. Mineral Reserves for Block 1 (Maga), Block 2 (CFU and P8P9), Block 3 (P11), and Block 4 (Siga) are based on marginal cut-off grades that range from 0.252 to 0.270 g/t Au for Oxides, and 0.464 to 0.516 g/t Au for Hard Rock.
7. Mineral Reserves for mining blocks Block 5 (P16) and Block 6 (P17) are based on polygons developed by Orezone delimiting oxide material averaging above 0.30 g/t Au and fresh rock above 0.50 g/t Au.
8. The Mineral Reserve estimates include oxide grade reduction factors applied by Orezone based on recent mine to mill reconciliation data.
9. Tonnage and grade measurements are in metric units. Contained Au is reported as troy ounces.
10. Processing recovery varies by weathering unit and location.
11. Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability.
12. Mineral Reserves are reported effective 28 March 2023.
13. Rounding of some figures might lead to minor discrepancies in totals.

The process through which the Mineral Reserves were estimated is as follows:

1) The resource block models were re-blocked by Orezone to account for dilution and losses generating diluted mine block models.

2) Geotechnical slope regions and pit optimization inputs, including mining and processing operating costs were added to the diluted block models by AMC.

3) Pit optimization was undertaken by AMC using Whittle and pit shells were selected from the results to form the basis of pit design.

4) Pit designs were created by AMC in Datamine Studio OP and Deswik based on the selected pit shells and geotechnical and operational design criteria. Ultimate pit designs were split into practical mining phases, including intermediate phase designs where appropriate.

5) Pit phase inventories were defined and imported into Minemax by AMC to generate the life-of-mine (LOM) schedule.

6) Appropriate modifying factors for conversion of Mineral Resources to Mineral Reserves were applied.

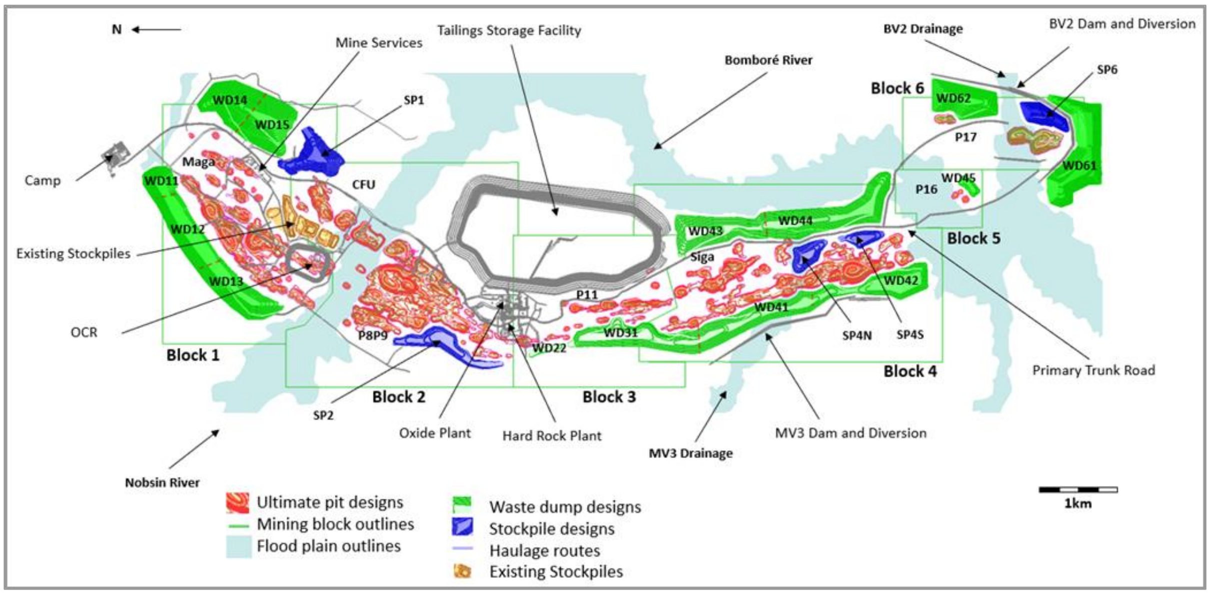
Pit optimization was undertaken by AMC in Whittle. Six pit optimizations were completed for each block model area:

- Block 1 – The Maga deposit north of the Nobsin River.
- Block 2 – The CFU deposit north and P8P9 deposit south of the Nobsin River.
- Block 3 – The P11 deposit.
- Block 4 – The Siga deposit.
- Block 5 – The P16 deposit.
- Block 6 – The P17 deposit.

A layout showing the location and extent of the six optimization areas is shown in Figure 1.8.1.

AMC generated ultimate pit designs based on the selected revenue factor 1 Whittle shells. The design incorporates 76 individually designed pits varying in depth from 18 to 180 m along a 14 km strike. 10 current starter pit phases are included in the mine plan along with a new starter phase pit in Siga and P17S. Certain larger pits were split into smaller parcels to add granularity to the mining schedule. These smaller parcels have individual pit names and are treated as pit pushback phases. Waste dumps and long-term stockpiles were designed, and a haul road network was prepared. A site layout plan view of the design is shown in Figure 1.8.1.

Figure 1.8.1 Ultimate pit designs and site layout



1.9 Mining

The Bomboré mine has been in commercial production since December 2022 and will be further developed as an open pit operation mining oxide and hard rock material using conventional truck and shovel mining methods. The 'oxide' includes the regolith, oxide, and upper transition weathering units. The regolith and oxide material are primarily free-digging material. The 'hard rock' includes lower transition and fresh rock. Upper transition and hard rock will require drill and blast prior to being loaded onto trucks. The hard rock ore will be treated in the planned new processing plant located adjacent to the existing oxide plant.

Mining of ore and waste is conducted by contractors with an owner's team responsible for site management, grade control, and mine planning activities. Mining of oxides is currently undertaken with 50 to 80 t diesel hydraulic excavators equipped with 3 to 5 m^3 buckets. Similar shovels are planned for mining the hard rock. The haulage requirements for oxide and hard rock material have been estimated based on rigid frame highway trucks with 26 t payload as currently deployed in the mining operations. Orezone is considering the application of trucks with higher payload of 30-60 t for all material types as part of the hard rock expansion.

ROM ore will be hauled to the process plants and low-grade and medium-grade material hauled to the ore stockpiles. Waste will be hauled to waste dumps with approximately 25% used for site and TSF construction.

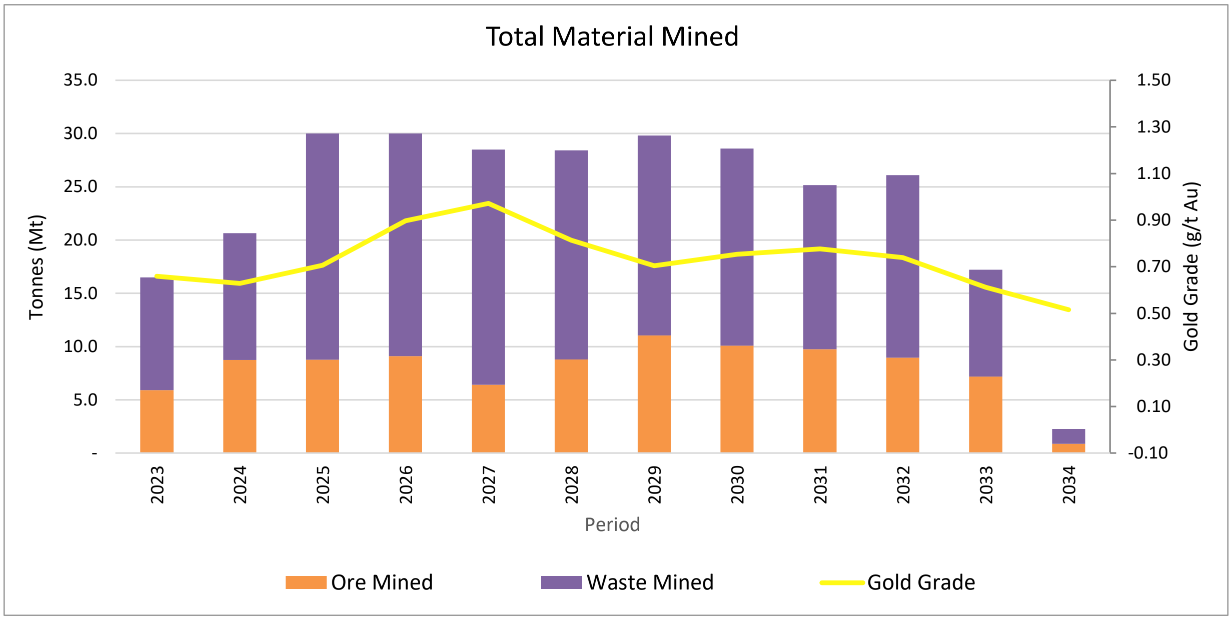
The mining schedule was optimized using Minemax Scheduler software. Pits were sequenced in order of value within assorted constraints such as wet seasons, access, TSF construction and plant ramp up. The target feed throughputs of both oxide and hard rock ore were achieved.

The key project LOM highlights are:

- 283.2 Mt total material mined.
- 103.5 Mt of ore:
  - 95.7 Mt of ore at 0.75 g/t Au mined and processed
  - 7.9 Mt of existing oxide stockpiles at 0.40 g/t Au reclaimed.
- 187.6 Mt waste.
- 2.1 Moz of Au produced.
- 2.0 strip ratio.
- 11.3-year mine life.

The total annual ore and waste movements are presented in Figure 1.9.1.

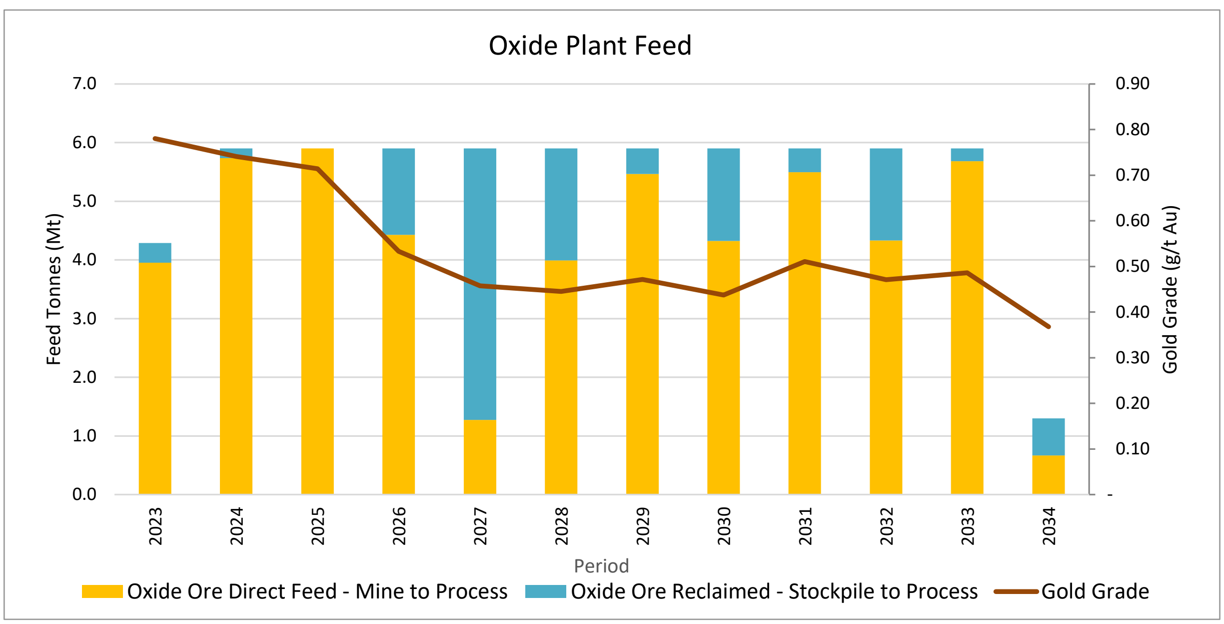
Figure 1.9.1 Annual Material Mined



Source: AMC, 2023.

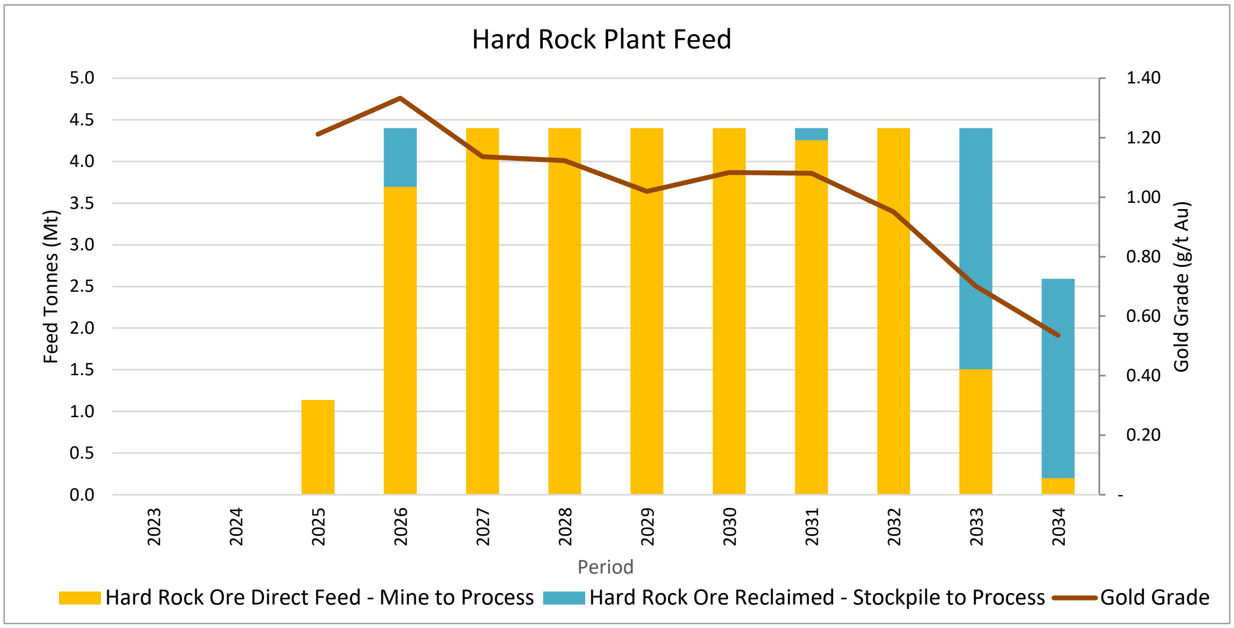
The oxide and hard rock process feed are presented in Figure 1.9.2 and Figure 1.9.3 respectively.

Figure 1.9.2 Oxide plant feed schedule



Source: AMC, 2023.

Figure 1.9.3 Hard rock plant feed schedule



Source: AMC, 2023.

Equipment numbers peak at 18 excavators and 132 trucks employing approximately 1,100 contractors and 250 owner's team personnel at the mine (excluding plant personnel).

1.10 Process Plant

Gold will be recovered from the Bomboré ore based on conventional unit operations including crushing, milling, Carbon-in-Leach (CIL) leaching, Zadra elution, gold electrowinning and carbon regeneration. The process plant design is based on a robust metallurgical flowsheet, developed for optimum recovery while minimizing initial capital expenditure and life of mine operating costs. The existing oxide plant will continue to process the oxide and upper transition ores at a nominal rate of 5.9 Mtpa. A new 4.4 Mtpa hard rock plant will process the lower transition and fresh ores.

Trucks transporting ore to the existing oxide plant rear-dump the ore onto a static grizzly and into the receiving bin. The grizzly is kept clear, as necessary, by a front-end loader. The saprolitic ore is introduced into a MMD sizer via an inclined apron feeder and then fed by conveyor into a single stage 3.2 MW ball mill, in closed circuit with hydrocyclones, to produce a P_{80} grind size of 125 µm. The ball mill discharge slurry is screened with a trommel and oversize pebbles are dropped into a bunker for manual removal. Cyclone overflow is screened to remove trash and is pumped to the leach circuit at a slurry density of 40% w/w solids. Lime is added onto the conveyor belt to maintain the pH, and liquid cyanide is pumped to the leach circuit to leach the gold. Activated carbon is used to adsorb the gold out of the slurry and loaded carbon is acid washed and pressure stripped in a Zadra elution circuit. A carbon regeneration kiln removes organic foulants from the carbon and reactivates the carbon. Gold is precipitated in electrowinning cells and is smelted in an electric furnace. The final product is doré bullion bars.

Trucks transporting ore to the 4.4Mtpa hard rock plant will rear-dump the ore onto a static grizzly and into the receiving bin. There will be two dump pockets to facilitate simultaneous dumping. The grizzly will be kept clear, as necessary, by a front-end loader. The hard rock will be delivered to a jaw crusher via an inclined apron feeder where it will be crushed to minus 314mm and conveyed to a 24h live capacity crushed ore stockpile. Ore will be reclaimed via two apron feeders and then fed by conveyor into a single stage 18 MW SAG mill, in closed circuit with hydrocyclones, to produce a P80 grind size of 75 µm. The SAG mill discharge slurry will be screened over a vibrating horizontal screen and the oversize pebbles will be conveyed back to the SAG mill feed.

Cyclone overflow at a slurry density of 30% w/w solids, will flow by gravity to trash removal screens and then to a 29 m diameter thickener. The slurry will be thickened to a density of 45% and pumped to a pre-oxidation tank, followed by seven CIL tanks providing the required 24 hours of residence time for optimum gold recovery. Oxygen is injected in the pre-oxidation and CIL tanks to improve leaching kinetics. Lime is added onto the conveyor belt to maintain the pH, and liquid cyanide is pumped to the CIL circuit to leach the gold. Activated carbon is used to adsorb the gold out of the slurry and loaded carbon is acid washed and pressure stripped in a Zadra elution circuit. The existing carbon regeneration kiln will reactivate the carbon and the existing gold room will precipitate gold in new electrowinning cells and produce doré bullion bars from the existing electric furnace.

The hard rock plant process flow sheet, plan view and isometric view are shown respectively in Figure 1.10.1, Figure 1.10.2 and Figure 1.10.3.

Figure 1.10.1 Bomboré Hard Rock Process Plant Flow Diagram

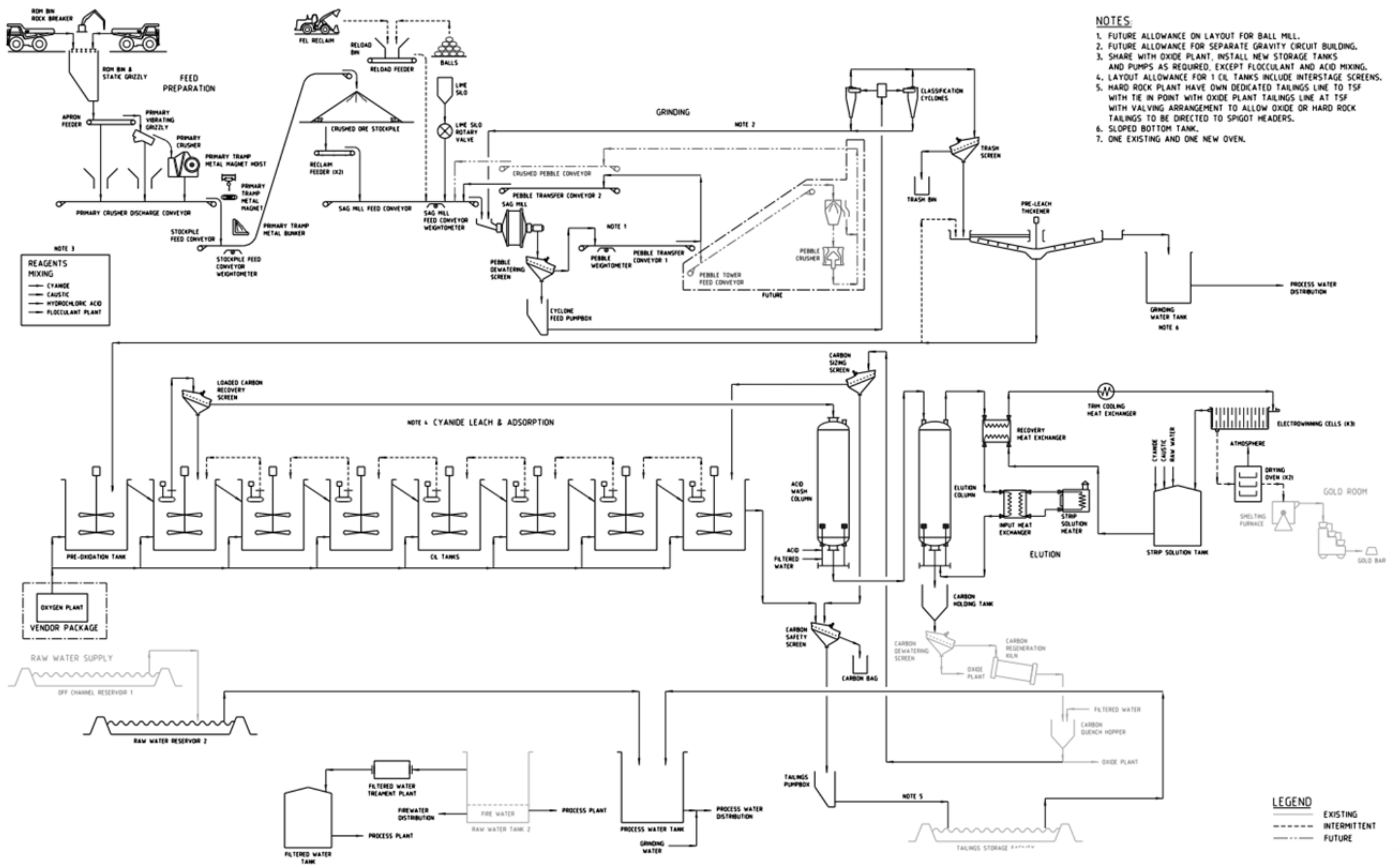


Figure 1.10.2 Hard Rock Plant Plan View

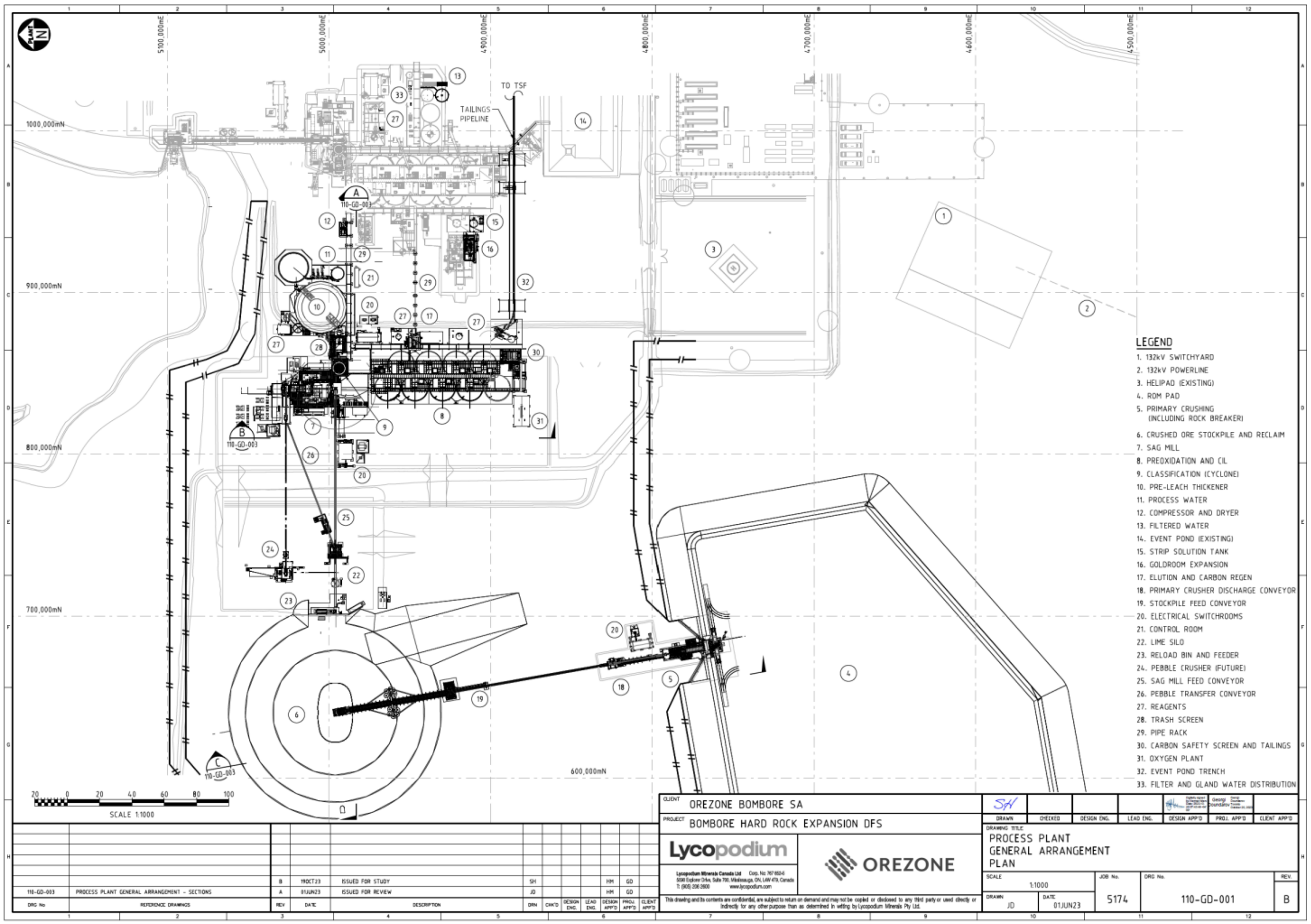
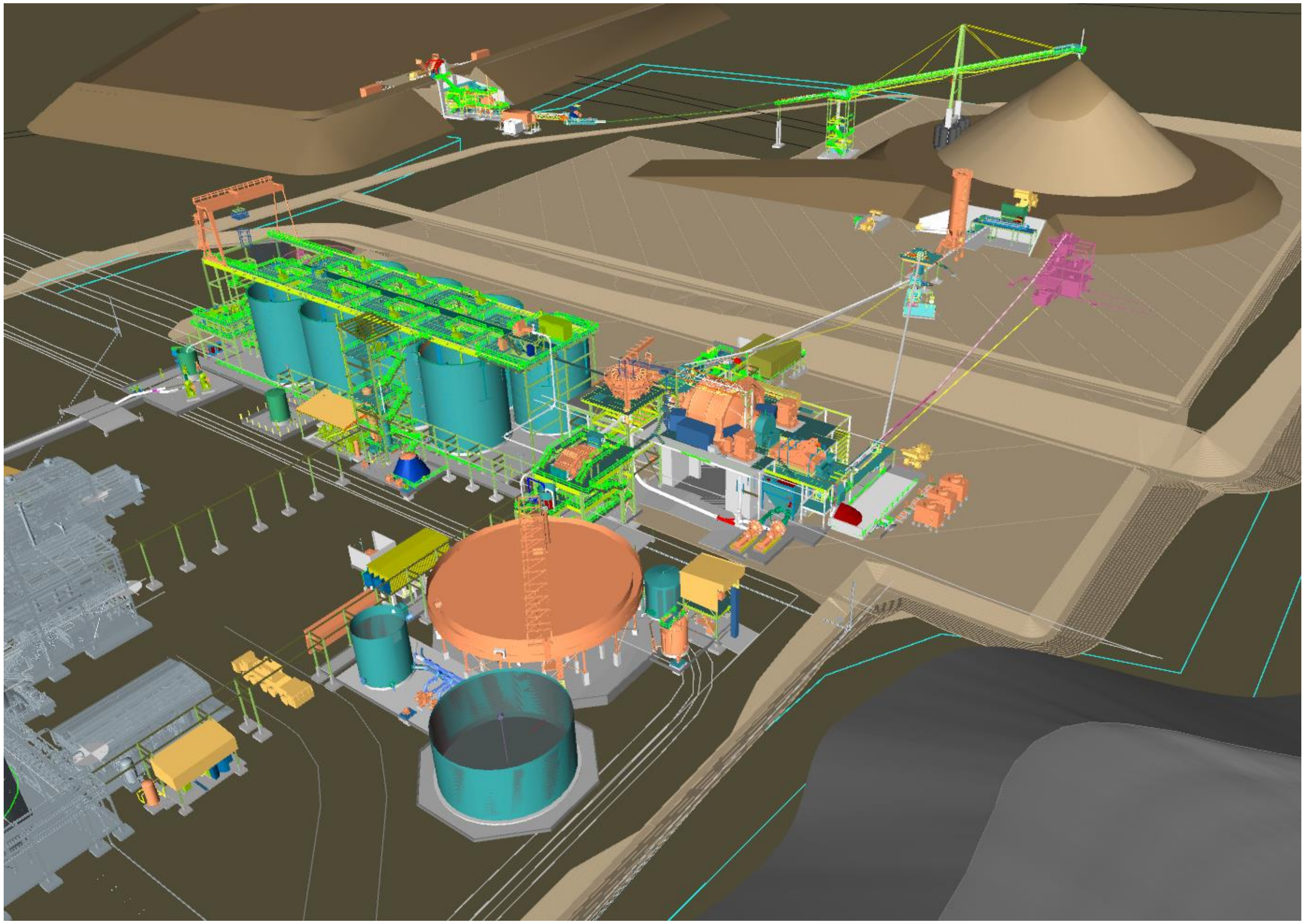


Figure 1.10.3 Hard Rock Plant Isometric View



The hard rock plant will comprise the following circuits:

1) Primary jaw crusher designed for a throughput of 670 (dry) tph and availability of 6,570 hours per annum.

2) Crushed coarse ore stockpile, which provides a live capacity of 15,000 tonnes.

3) A SAG mill and classification circuit with throughput capacity of 4.4 Mtpa of ore at design grind of P_{80} 75 µm.

4) A leach feed thickener to increase milling circuit classification efficiency and reduce the volume required in the leaching circuit.

5) A pre-oxidation tank with four hours retention time of thickener underflow, supported by an oxygen plant and oxygen sparging.

6) A conventional CIL circuit consisting of seven leach-adsorption tanks to achieve the 24 hours target residence time; in fact, the design allows for 27.8 hours at 45% solids.

- Gold recovery and refining consisting of an elution circuit, electrowinning cells and smelting.

7) An expanded existing tailings storage facility for pumped tailings with a decant return system to reclaim water for use as process water.

Process plant infrastructure within the plant boundary fence will comprise typical items such as the reagent storage and make-up circuits and reticulation, utilities and services supply and distribution, offices, control rooms, workshops, stores, ablution blocks, roads, and security. The existing oxide plant infrastructure will be utilized as far as practical with expansion and additions to sections as required.

1.11 Infrastructure

The Bomboré Mine benefits from a strong mining culture and excellent local infrastructure. Burkina Faso has an expanding pool of available mining contractors, suppliers, and skilled labour. The mine is favourably situated only 85km from the capital city of Ouagadougou, accessed by a 5km all-weather road connecting to the main sealed highway (RN4). In addition, current construction of the neighbouring Kiaka mine (West Africa Resources Ltd.) is underway and Orezone’s Phase II Expansion is expected to benefit from synergies including the use of common contractors.

1.11.1 Tailings Storage Facility

The existing tailings storage facility is fully lined with a pump out decant system. The facility is designed to be raised in stages over the mine life with downstream embankment construction techniques using run-of-mine waste rock. The capacity of the tailings storage facility will be expanded from 70 Mt to 128 Mt, which is sufficient for the current mineral reserves plus additional space for potential future expansions.

1.11.2 Power Supply

The Company's grid power project to connect Bomboré to Burkina Faso's national grid is scheduled for completion before the end of 2023. ECG Engineering Pty Ltd. (ECG) is managing the design, construction, and commissioning of the new high voltage transmission line and dedicated substations, and has been working closely with SONABEL, Burkina Faso's state-owned electricity company, to ensure timely deliverables and adherence to schedule. ECG is a specialized engineering firm that has successfully delivered on similar projects in West Africa, including Burkina Faso. All major equipment and materials have shipped, and installations are progressing on schedule.

1.11.3 Water Supply

Raw water is currently sourced from the seasonal Nobsin River and diverted by a weir into an existing 5.2Mm^3 off-channel reservoir (OCR). A pit in the P8P9 orebody has been selected for early excavation to serve as a second 1.8 Mm^3 reservoir which will store sufficient water for the expanded plant throughput.

1.12 Environmental, Permitting and Resettlement Action Plan

1.12.1 Environmental, Social and Permitting

The approach developed by Orezone throughout the various environmental and social studies that have been conducted since 2009, especially in the context of the Environmental and Social Impact Assessment, emphasized stakeholder concerns and integrated the environmental and social aspects into the initial stages of the Bomboré Mine design and continues into the Phase II Expansion design. This approach has ensured the integration of environmental and social issues in the design for the Bomboré Mine.

Various permits and authorizations are required for the Bomboré Mine. Orezone holds all permits that are required for its current operations and those envisioned in the 2019 technical report. Orezone has been successful in obtaining such permits and authorizations in the past and is confident that it will be able to obtain the required permits and authorizations for the Phase II Expansion.

1.12.2 Resettlement Action Plan

RAP Phases II and III follow the successful completion of Phase I RAP and involves the construction of three new resettlement communities (MV3, MV2, and BV2). Phase II is well-advanced with the construction of MV3 sequenced as the first community to construct in order to gain access to mining areas that are currently contemplated in the 2024 mine plan. MV3 is the largest of the resettlement communities.

A RAP Phase IV is planned to accommodate an increased footprint to the mining lease. This resettlement will be performed progressively over 2024 through to 2027.

1.13 Capital and Operating Costs

1.13.1 Phase II Expansion Project Capital Cost

The capital cost of the Phase II Expansion is estimated at $167.5 M as shown in Table 1.13.1. The capital cost estimate was compiled by Lycopodium and is based on Q3-2023 pricing. The estimate is deemed to have an accuracy of ±15%.

Table 1.13.1 Phase II Expansion Capital

Description	Total Costs $M
Process Plant	81.0
Infrastructure	13.2
Construction Indirects	14.5
Owner's Cost (including EPCM)	47.7
Subtotal	156.5
Contingency	11.0
Total Expansion Capital Costs	167.5

1.13.2 Sustaining Capital & Closure Costs

Growth capital includes the grid power connection project that will be completed in Q4-2023, RAP Phases II & III, that are currently underway and will be completed in 2024, and RAP Phase IV that will be performed progressively over 2024 through to 2027. Sustaining capital costs include ongoing tailings storage facility raises, haul road extensions, grade control drills, and mine dewatering and surface water management equipment. Closure cost includes the remediation work required to return the site to meet all conditions of the Environmental and Social Impact Assessment. LOM Growth Capital, Sustaining Capital and Closure Costs are summarized in Table 1.13.2.

Table 1.13.2 LOM Growth Capital, Sustaining Capital, and Closure Costs ($M)

Description	Total Costs
Growth Capital	
Grid Power	16.3
RAP Phase 2 & 3	23.0
RAP Phase 4	18.4
Growth Capital total	**57.7**
Sustaining Capital	
Plant	2.1
Infrastructure	87.0
Mining	8.4
G&A	3.6
Sustaining Capital total	**101.0**
Closure Costs	
Reclamation and Closure	19.1
Salvage Value	(9.9)
Total Closure Cost	**9.3**
Total Growth, Sustaining & Closure Cost	**168.0**

1.13.3 Operating Cost Estimate

The life of mine AISC is estimated at $1,122/oz using a base case gold price of $1,750/oz and a USD to XOF exchange rate of 600. Electrical grid power is projected to reduce energy costs to $0.21/kWh from the current $0.62/kWh which is based on diesel generation. Contract mining has been selected as the basis for open pit mining activities, to be managed by the Bomboré operation team, and costs are based on contractor proposals. Processing cost estimates are life of mine averages and include various annual blends of oxide, transition, and fresh ores as mill feed, incorporating the associated reagent consumptions, work indices, abrasion indices, and power requirements. Operating costs are summarized in Table 1.13.3.

Table 1.13.3 Operating Cost Estimate Summary (Oxide & Hard Rock)

Description	Total Costs	$/tonne milled	$/ounce
Mining	840.2	8.12	398
Processing	945.6	9.13	448
Site G&A	242.9	2.35	115
Refining and transport	5.8	0.06	3
Government royalties	222.3	2.15	105
Total Cash Costs	**2,256.7**	**21.80**	**1,070**
Sustaining capital	101.0	0.98	48
Rehabilitation and closure	19.1	0.18	9
Salvage Value	(9.9)	(0.10)	(5)
All-in Sustaining Cost	**2,367.0**	**22.87**	**1,122**

1.14 Annual and Life-of-Mine Production

Life of mine gold production is 2.1 million ounces from 103.5 million tonnes of ore processed, as presented in Table 1.14.1.

Table 1.14.1 LOM and Annual Production

	Units	LOM	Annual											
			9M 2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Oxide Production														
Mill Feed	Mt	64.6	4.3	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	1.3
Grade, Au	g/t	0.54	0.78	0.74	0.71	0.53	0.46	0.45	0.47	0.44	0.51	0.47	0.49	0.37
Recovery	%	90.9	90.8	91.1	90.8	91.0	91.4	90.8	90.3	91.0	90.9	91.0	90.8	91.1
Oxide Gold Production	**koz**	**1,020**	**98**	**128**	**123**	**92**	**79**	**77**	**81**	**75**	**88**	**81**	**84**	**14**
Hard Rock Production														
Mill Feed	Mt	38.9			1.1	4.4	4.4	4.4	4.4	4.4	4.4	4.4	4.4	2.6
Grade, Au	g/t	1.02			1.21	1.33	1.14	1.12	1.02	1.08	1.08	0.95	0.70	0.54
Recovery	%	85.0			85.7	87.6	86.8	87.5	83.6	82.3	83.0	84.0	84.1	84.8
Hard Rock Gold Production	**koz**	**1,089**			**38**	**165**	**140**	**139**	**121**	**126**	**127**	**113**	**83**	**38**
Combined Production														
Mill Feed	Mt	103.5	4.3	5.9	7.0	10.3	10.3	10.3	10.3	10.3	10.3	10.3	10.3	3.9
Grade, Au	g/t	0.72	0.78	0.74	0.79	0.87	0.75	0.73	0.71	0.71	0.75	0.68	0.58	0.48
Recovery	%	87.8	90.8	91.1	89.5	88.8	88.4	88.7	86.1	85.4	86.1	86.8	87.3	86.4
Combined Gold Production	**koz**	**2,109**	**98**	**128**	**161**	**257**	**219**	**216**	**201**	**202**	**215**	**194**	**167**	**52**

1.15 Economic Analysis

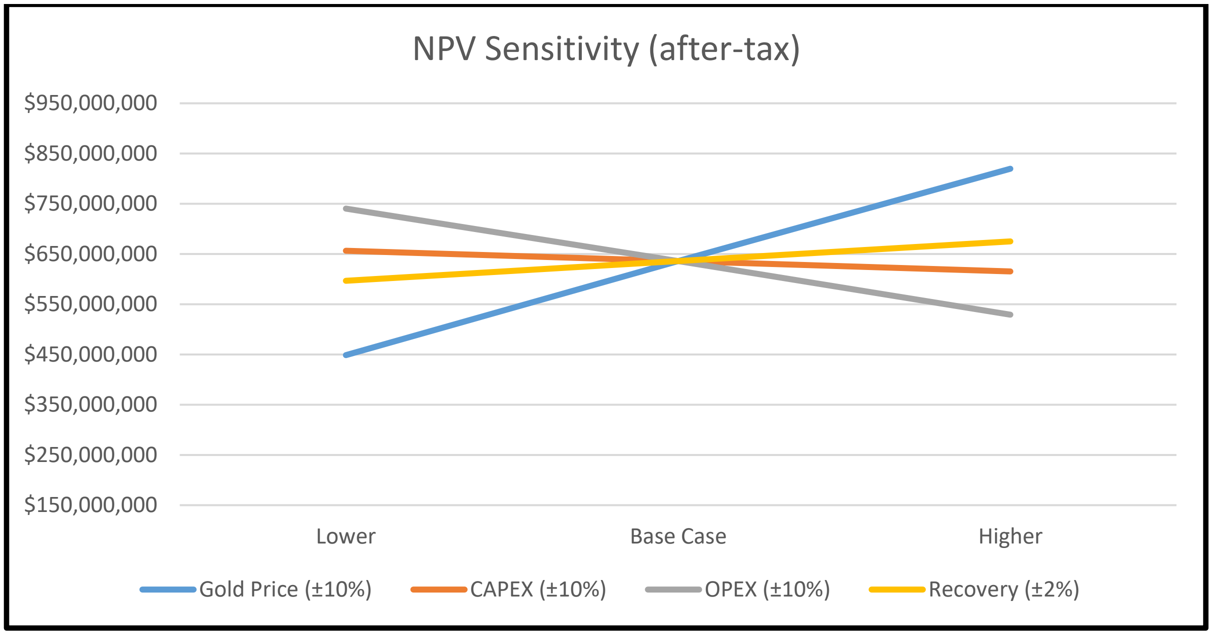
An economic assessment of the Phase II Expansion has been conducted using a pre and after-tax cash flow model prepared by Lycopodium on behalf of Orezone. The project demonstrates positive economics as summarized in Table 1.15.1.

Table 1.15.1 Financial Summary

	Value
LOM gold production	2.11 Moz
Revenue from Gold (99.95% Payable)	\$3,704M
Operating Costs	\$2,257M
Phase II Capital	\$167.5M
Growth Capital	\$57.7M
Sustaining Capital and Closure Costs	\$110.3M
Pre-Tax Cash Flow	\$1,143M
After-Tax Cash Flow	\$885M
After-Tax economics:	
NPV (5%)	\$636M

The Net Present Value of the Project including the Phase II Expansion value was assessed by undertaking sensitivity analyses on the gold price, gold recoveries, operating costs, and capital costs. The Project's NPV is most sensitive to changes in the gold price and then operating costs. The results of after-tax sensitivity analyses are presented in Figure 1.15.1.

Figure 1.15.1 NPV Sensitivity (After-tax)



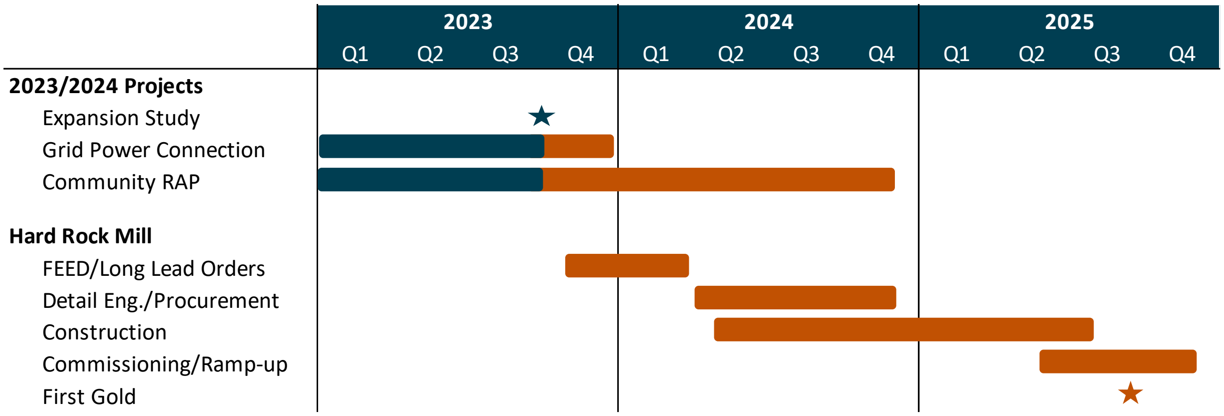
On 27 October 2023, the President of Burkina Faso signed a decree to increase royalty rates on gold sales. The decree increases the royalty from the previous 5.0% on all gold sales at or above $1,500 per ounce to a new rate of 6.0% on gold sales at or above $1,500 and under $1,700 per ounce, 6.5% on sales at or above $1,700 and under $2,000 per ounce and has been capped at 7.0% for gold sales at or above $2,000 per ounce. Certain legislative procedural matters are required before the new royalty rates become law and as of the date of filing this Technical Report, these had not yet occurred. Although these new rates will not have a material impact on the cash flow model, readers are cautioned that the new royalty rates have not been included in the economic analysis.

Assuming the royalty increase is officially adopted into law at the beginning of 2024, the after-tax NPV of the Project would be reduced from $636M to $607M.

1.16 Project Implementation and Schedule

The Phase II Expansion is being managed by the same team who successfully delivered the Phase I plant on time and under budget. The overall schedule, summarized in Figure 1.16.1, is 24 months with the critical path being the delivery and installation of the SAG mill. The Company expects to place the order for the SAG mill in Q4-2023 with early works on site expected to commence in Q1-2024.

Figure 1.16.1 Phase II Expansion Schedule



1.17 Conclusions and Recommendations

Based on the work undertaken and the conclusions listed in Section 25, Bomboré Phase II Expansion is a viable development opportunity, centred around the construction of a 4.4 Mtpa Hard Rock plant to process lower transition and fresh material.

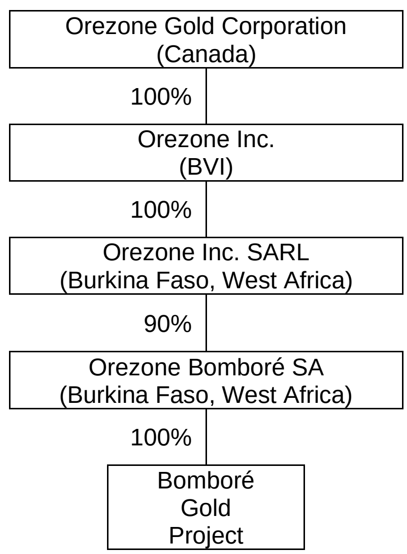
Refer to Sections 25 and 26 for specific conclusions and recommendations.

2.0 INTRODUCTION

2.1 Introduction

Orezone Gold Corporation (Orezone) is engaged in mining, developing, and exploring its 90%-owned flagship Bomboré gold mine (Bomboré) in Burkina Faso, West Africa. The Company achieved commercial production at the Bomboré mine on 1 December 2022. The Bomboré oxide process plant was delivered on schedule and under budget and production in Q1 2023 was 41,301 ounces of gold at an all-in sustaining cost of $926/oz sold.

Bomboré is owned 90% by Orezone and 10% by the Burkina Faso government. The following chart illustrates the material subsidiaries of Orezone and, for the purposes of this Report, unless otherwise stated, the term Orezone includes its material subsidiaries and any predecessor companies.



The Property comprises a block of contiguous permits totalling 12,963 ha located in Ganzourgou Province, Burkina Faso, approximately 85 km east of the capital city of Ouagadougou. The Property is easily accessible from Ouagadougou via the paved national highway N4.

Lycopodium Minerals Canada Ltd (Lycopodium) was commissioned by Orezone Gold Corporation (Orezone) to prepare an NI 43-101 Technical Report (Report) for the Bomboré Mine. The firms and consultants who are responsible for the content of this Report are Lycopodium Minerals Canada Ltd (Lycopodium), Knight Piesold Pty Limited (Knight Piesold), P&E Mining Consultants Inc. (P&E), AMC Mining Consultants Ltd. (AMC), and Africa Label Group Inc.

The previous Bomboré Technical Report (Feasibility Study of the Bomboré Gold Project, effective date 26 June 2019) contemplated the construction of a 5.2 Mtpa oxide processing installation, followed by an expansion to process 2.2 Mtpa of hard rock ore while continuing processing oxide at a reduced rate of 3.0 Mtpa.

This Technical Report supersedes the 2019 Technical Report and incorporates additional exploration drilling, expanded mineral resources and reserves and a summary of the Feasibility Study for the Bomboré Phase II Expansion Project. The Bomboré mine plan demonstrates a mine life of greater than 11 years with gold production from the existing oxide plant operating at 5.9 Mtpa, plus gold production from a new 4.4 Mtpa hard rock plant to process the lower transition and fresh ore.

2.2 Terms of Reference and Purpose of this Report

This Report is prepared in support of Orezone's press release dated 11 October 2023 entitled "Orezone Announces Bomboré Phase II Expansion Study Results".

This Report provides updated information on the current operation of Bomboré, including updated Resource and Reserve estimates, descriptions of the oxide mining and processing operations, and feasibility reporting of a planned expansion to mine and process fresh ore.

2.3 Qualified Persons

The individuals presented in Table 2.3.1, by virtue of their education, experience and professional association are considered Qualified Persons (QPs) as defined in NI 43-101 for this Report. The QPs meet the requirement of independence as defined in NI 43-101.

Table 2.3.1 Persons Who Prepared this Report

Qualified Persons Responsible for the Sections of this Report						
Qualified person	**Position**	**Employer**	**Independent of Orezone**	**Date of last site visit**	**Professional Designation**	**Report Sections**
Georgi Doundarov	Senior Project Manager	Lycopodium Minerals Canada Ltd (LMCL)	Yes	N/A	P.Eng, PMP, CCP	1.1, 1.2, 1.3, 1.11, 1.13, 1.15, 1.16, 1.17, 2, 3, 4, 5, 6, 18.1 to 18. 12, 19, 20, 21.1, 21.2.1, 21.2.4, 21.3, 22, 24, 25.1, 26.1, and 27.
Olav Mejia	Manager of Process	Lycopodium Minerals Canada Ltd (LMCL)	Yes	N/A	P.Eng	1.6, 1.10, 13, 17, 21.2.3, 25.4, 26.4
Antoine Yassa	Sr. Associate Geologist	P&E Mining Consultants Inc. (P&E)	Yes	9-14 October, 2017	P.Geo	12,14, and 27
William Stone	Sr. Associate Geologist	P&E Mining Consultants Inc. (P&E)	Yes	N/A	P.Geo	1.4, 1.5, 7, 8, 9, 10, 23, 25.2, 26.2, 26.7, and 27
Jarita Barry	Associate Geologist	P&E Mining Consultants Inc. (P&E)	Yes	N/A	P.Geo	11, 12, and 27
Fred Brown	Associate Geologist	P&E Mining Consultants Inc. (P&E)	Yes	N/A	P.Geo	14 and 27
Eugene Puritch	President & Principal Mining Engineer	P&E Mining Consultants Inc. (P&E)	Yes	N/A	P.Eng	1.7, 14, 25.2, 26.2, 26.7, and 27
David Warren	Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	N/A	P.Eng.	1.8, 1.9, 1.14, 15, 16, 21.2.2, 25.3 and 26.3
David Morgan	Managing Director	Knight Piésold Pty Ltd (KP)	Yes	10-13 July 2023		18.13, 18.14, 25.5 and 26.5.
Bright Oppong Afum	Environmental Consultant	Africa Label Group Inc.	Yes	19 November, 2023	MAusIMM (CP)	1.12, 20, 25.6, and 26.6

2.4 Site Visits

The Bomboré Project was visited by Mr. Antoine Yassa, P.Geo., between 9 to 14 October 2017 for the purposes of completing a site visit and due diligence sampling. General data acquisition procedures, core logging procedures and QA/QC were discussed during the visit.

David Morgan visited the site between 10 to 13 July 2023. While at site, Mr. Morgan conducted walkover inspections of the Tailings Storage Facility, tailings and decant pipelines.

Bright Oppong Afum visited the site on 19 November 2023. While at site, Dr. Afum visited the communities and resettlement areas.

2.5 Abbreviations

Abbreviation	Meaning
%	Percent/percentage
μm	Micrometre (micron)
AAVV	l'Autorité pour l'Aménagement des Vallées de la Volta
AFD	Agence Française de Développement
AMD	Acid mine drainage
ARD	Acid rock drainage
AVV	Aménagement des Vallées des Volta
BFT	Bomboré First Target
BLEG	Bulk Leach Extractable Gold
BLK	Blank (blind to the preparation laboratory)
BOO	Build-Own-Operate
BSZ	Bomboré Shear Zone
BUMIGEB	Bureau des Mines et de la Géologie du Burkina
BUNEE	Bureau National des Évaluations Environnementales
BWi	Bond ball mill work index
BV	Bed volumes
CaO	Calcium oxide (lime or quicklime)
CDA	Canadian Dam Safety
CIL	Carbon in Leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	Carbon in Pulp
cm	Centimetre
CM	Certified Reference Material
CMB	Chambre des Mines du Burkina Faso
CNM	Commission Nationale des Mines
CNT	Commission Nationale de la Transition
COTEVE	Comité technique sur les Évaluations Environnementales
CoV	Coefficient of variation
CRM	Certified Reference Material
Cu	Copper
CWi	Crusher Work index
d	day
DFS	Definitive Feasibility Study
DGMG	Direction Générale des Mines et de la Géologie
DGPE	Direction Générale de Préservation de l'Environnement

Abbreviation	Meaning
DGPS	Differential Global Positioning System
DNEF	Direction Nationale des Eaux et Forêts
EIA	Environmental Impact Assessment
EPCM	Engineering Procurement Construction Management
ERP	Emergency Response Plan
ESIA	Environmental and Social Impact Assessment
ESMP	Environmental Social Management Plan
FA	Fire Assay
FD	Field Duplicate (blind to preparation laboratory)
Fe	Iron
FEED	Front End Engineering Design
FEL	Front end loader
FS	Feasibility Study
g	Gram
g Ag/t	Grams of silver per tonne
g Au/t	Grams of gold per tonne
G&A	General and Administration
g/t	Grams per tonne
GARD	Global Acid Rock Drainage
GMC	Générale de Mines et de Carrières
GWh	Gigawatt hours
H_2SO_4	Sulphuric acid
ha	Hectare
HCl	Hydrochloric acid
HCN	Hydrogen cyanide
HDPE	High density poly-ethylene
HFO	Heavy fuel oil
Hg	Mercury
HGO	High Grade Oxide Ores
HGS	High Grade Hard Rock Ores
HL	Heap leach
HQ	63.5 mm diameter core drill tube
Hr or h	Hours
ICP	Inductively couple plasma
ID2	Inverse distance squared
ID3	Inverse distance cubed
IHC	In-house referenced material

Abbreviation	Meaning
IP	Induced polarization
IPAQ	Hewlett-Packard brand name for data logger device
IPP	Independent power provider
IRR	Internal rate of return
kg	Kilogram
kg/t	Kilograms per tonne
km	Kilometre
kPag	Kilopascal Gauge
kW	Kilowatt
L	Litre
LAPD	Lab-Aware Pulp Duplicate (known to the analytical laboratory)
LAQE	Laboratoire d'Analyse de la Qualité de l'Environnement
LOM	Life of mine
LW	LeachWELL
m	Metre
M&I	Measured and Indicated
m^2	Metres squared/square metre
m^3	Metres cubed/cubic metre
m^3/h	Cubic metres per hour
MAC	Mining Association of Canada
MC	Master Composite
MCC	Motor control centre
ME	Ministère de l'énergie
MEDD	Ministry of the Environment and Durable Development
MEEVCC	Ministère de l'Environnement de l'Économie Verte et du Changement Climatique
MEMC	Ministère de l'Énergie des Mines et des Carrières
MGO	Medium Grade Oxide Ores
MGS	Medium Grade Hard Rock Ores
Mins	minutes
ML	Leachable metals
MLCM	Million Loose Cubic Metres
mm	Millimetre
MMC	Ministère des Mines et des carrières
MRE	Mineral Resource estimate
mS	Micro-Siemen
Mt	Million tonnes
Mtpa	Million tonnes per annum

Abbreviation	Meaning
MPa	Megapascal
MW	Megawatt
NaCN	Sodium cyanide
NaOH	Sodium hydroxide/Caustic soda
NC	Non certified reference material
Ni	Nickel
NN	Nearest neighbour
NSR	Net smelter royalty
OCR	Off-Channel Reservoir
OIT	Operator interface terminals
OK	Ordinary kriging
oz	Ounce
PCS	Process control system
PD	Pulp Duplication (blind to analytical laboratory)
PEA	Preliminary Economic Assessment
PFR-G	Plan foncier rural du Ganzourgou
PFS	Pre-feasibility Study
pH	Measure of acidity/basicity
PLC	Programmable logic controller
pmp	Probable maximum precipitation
pmf	Possible maximum flood
ppb	Parts per billion
ppm	Parts per million
PQ	85 mm diameter core drill tube
QA/QC	Quality Assurance/Quality Control
QEM-RMS	Method of determining mineralogy of a sample using an electron microscope
RAB	Rotary air blast
RAP	Resettlement Action Plan
RC	Reverse Circulation
rpm	Revolutions per minute
ROM	Run of mine
RQD	Rock Quality Designation
RSD	Rotary sample divider
RWi	Bond rod mill work index
SCADA	Supervisory control and data acquisition
sec	Seconds
SMBS	Sodium metabisulphite

Abbreviation	Meaning
SMU	Selective mining unit
SO_2	Sulphur dioxide
STD	Standard (blind to the preparation laboratory)
t	Tonne
Te	Tellurium
ToR	Terms of Reference
TSF	Tailings storage facility
TVA	Taxe sur la Valeur Ajoutée (Value Added Tax)
UCS	Unconfined compressive strength
U/F	Underflow
UPS	Uninterrupted power supply
US$	United States Dollar
UTM	Universal Transverse Mercator
V	Volt
VLF-EM	Very low frequency – electromagnetic
VMP	Vibrating wire piezometers
WAD	Weak acid dissociable
WAF	West African Resources Ltd.
WRD	Waste rock dumps
w/o	Without
w/w	Weight/weight
yr	Year
XFR	X-Ray fluorescence

3.0 RELIANCE ON OTHER EXPERTS

3.1 Legal Standing of Tenements

Regarding the legal standing of the Project's tenements, the authors of this Report relied on a legal opinion provided by Bobson Coulibaly of Yanogo Bobson Avocats Lawyers, Rue 30.81, ZAD, Ouagadougou, Burkina Faso in November 2023 indicating that fees and taxes have been paid, permits are in good standing and all required authorizations have been obtained to run mining production activities at Bomboré.

3.2 Tax

Lycopodium has relied on Mr. Peter Tam, Chief Financial Officer, Orezone Gold Corporation, for Burkina Faso specific tax advice relating to the Project when developing the cash flow model. Mr. Tam assisted with the development of the model and provided advice relating to tax rates, depreciation, and other model inputs.

3.3 Production Capacity

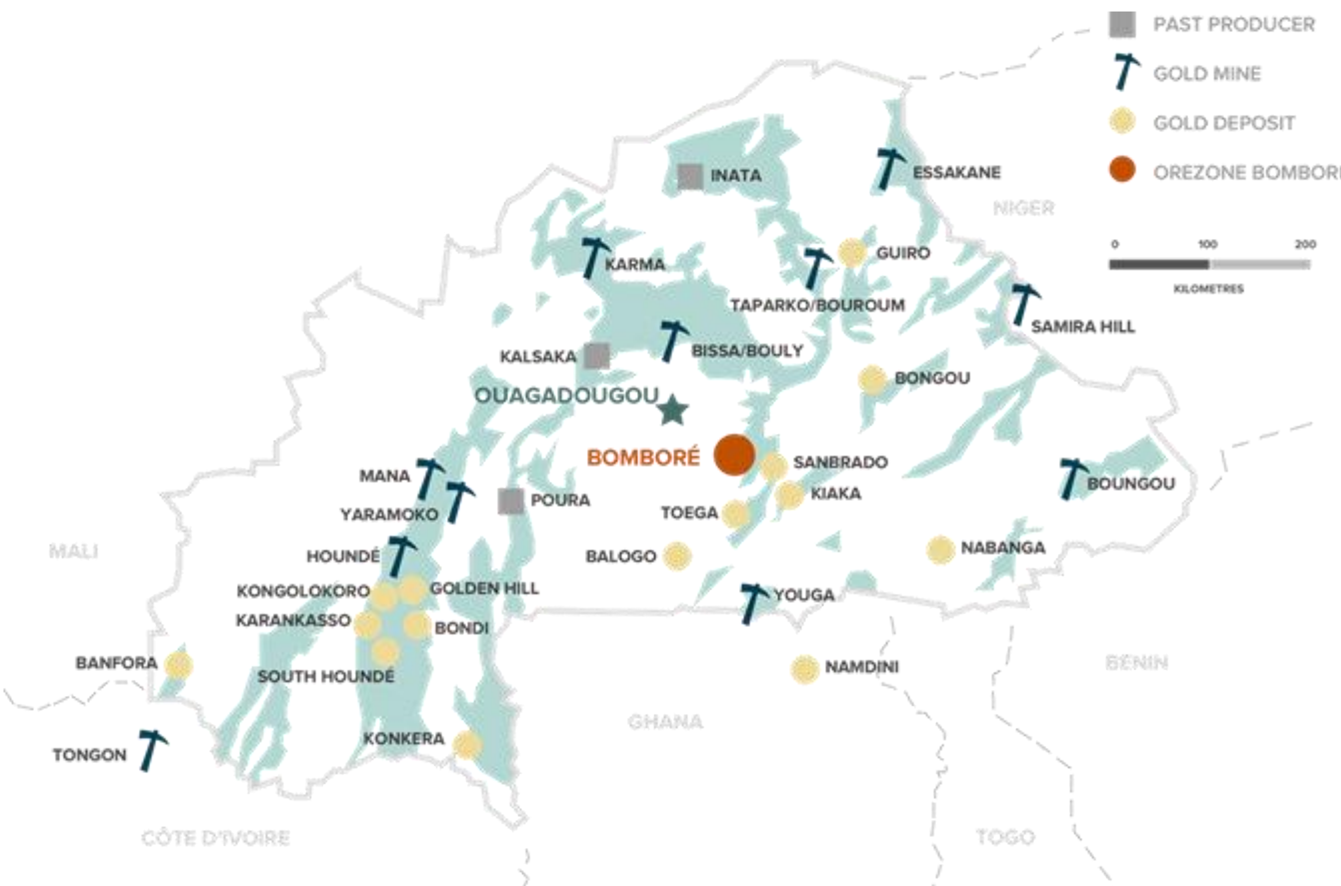
Lycopodium has relied on Mr. Rob Henderson, Vice President Technical Services, Orezone Gold Corporation, for 2023 operational oxide plant production results and expected production on that plant until year 2034 as presented in Section 22 of this Report. Mr. Henderson assisted with the development of the production model and provided input on actual and expected operational results and plant performance parameters.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The Property is located in Ganzourgou Province, Burkina Faso, approximately 85 km east of the capital city of Ouagadougou (Figure 4.1.1). The Property is easily accessible from Ouagadougou via the paved national highway N4.

Figure 4.1.1 Property Location



4.2 Land Tenure

The Property covers an area of 12,963 ha and consists of one Industrial Operating Permit (the Bomboré Mining Permit) of 2,887 ha, surrounded by four Mining Exploration Permits: the Bomboré II Exploration Permit of 1,265 ha, the Bomboré III Exploration Permit of 3,360 ha, the Bomboré IV Exploration Permit of 833 ha and the Bomboré V permit of 4,618 ha.

Mining permits are granted by Decree of the Council of Ministers and exploration permits are granted by order of the MEMC of Burkina Faso. The Government of Burkina Faso retains a 10% free carried interest in a mining company holding a mining permit. The government's free carried interest cannot be diluted.

Exploration permits are issued for an initial three-year term as of the date of issuance and may be renewed for a maximum of two consecutive three-year terms according to the Mining Act.

Exceptional extensions of up to three additional years have been granted for several permits in recent years.

The land tenure information presented herein is derived from copies of the Decree and Orders by which the Property permits were granted.

The boundaries of each permit are defined by corner posts positioned according to geographic coordinates (ITRF 2008 BFTM datum, UTM Zone 30) as indicated in Tables 4.2.1 to Table 4.2.5. The boundaries of mining permits must be physically marked on the ground and legally surveyed within six months of its issuance.

The boundaries of exploration permits are not subject to this requirement.

Table 4.2.1 Bomboré Mining Permit Boundaries

Apex	X_ITRF2008	Y_ITRF2008	Apex	X_ITRF2008	Y_ITRF2008
P001	666600	1355100	P091	666100	1342600
P002	667000	1355100	P092	665900	1342600
P003	667000	1355000	P093	665900	1343100
P004	667100	1355000	P094	665800	1343100
P005	667100	1354000	P095	665800	1343500
P006	667400	1354000	P096	665300	1343500
P007	667400	1353900	P097	665300	1343400
P008	667700	1353900	P098	665000	1343400
P009	667700	1353800	P099	665000	1343500
P010	667800	1353800	P100	664900	1343500
P011	667800	1353700	P101	664900	1343600
P012	667900	1353700	P102	664800	1343600
P013	667900	1353600	P103	664800	1343800
P014	668000	1353600	P104	664700	1343800
P015	668000	1353400	P105	664700	1344100
P016	668400	1353400	P106	664600	1344100
P017	668400	1353100	P107	664600	1344800
P018	668300	1353100	P108	664400	1344800
P019	668300	1352900	P109	664400	1345800
P020	668000	1352900	P110	664300	1345800
P021	668000	1352800	P111	664300	1346000
P022	667900	1352800	P112	664200	1346000
P023	667900	1352500	P113	664200	1346100
P024	667800	1352500	P114	664100	1346100
P025	667800	1351900	P115	664100	1346200
P026	667700	1351900	P116	664000	1346200
P027	667700	1351700	P117	664000	1346400
P028	667600	1351700	P118	663900	1346400
P029	667600	1351600	P119	663900	1346600
P030	667500	1351600	P120	663800	1346600
P031	667500	1351500	P121	663800	1347000
P032	667400	1351500	P122	664000	1347000
P033	667400	1351400	P123	664000	1347600
P034	667200	1351400	P124	664100	1347600

Apex	X_ITRF2008	Y_ITRF2008	Apex	X_ITRF2008	Y_ITRF2008
P035	667200	1351300	P125	664100	1348000
P036	667100	1351300	P126	664000	1348000
P037	667100	1351200	P127	664000	1348300
P038	667000	1351200	P128	664100	1348300
P039	667000	1351100	P129	664100	1348600
P040	666900	1351100	P130	664000	1348600
P041	666900	1351000	P131	664000	1349000
P042	666200	1351000	P132	663900	1349000
P043	666200	1350300	P133	663900	1349200
P044	666300	1350300	P134	663800	1349200
P045	666300	1350200	P135	663800	1350100
P046	666500	1350200	P136	663900	1350100
P047	666500	1350000	P137	663900	1350300
P048	666600	1350000	P138	664000	1350300
P049	666600	1349900	P139	664000	1350800
P050	666900	1349900	P140	663900	1350800
P051	666900	1349600	P141	663900	1351300
P052	666800	1349600	P142	664000	1351300
P053	666800	1349000	P143	664000	1351600
P054	666700	1349000	P144	664200	1351600
P055	666700	1348800	P145	664200	1351900
P056	666400	1348800	P146	664300	1351900
P057	666400	1348700	P147	664300	1352200
P058	666300	1348700	P148	664400	1352200
P059	666300	1348500	P149	664400	1352600
P060	666200	1348500	P150	664500	1352600
P061	666200	1347400	P151	664500	1352800
P062	666000	1347400	P152	664700	1352800
P063	666000	1347200	P153	664700	1353000
P064	665900	1347200	P154	664800	1353000
P065	665900	1346300	P155	664800	1353100
P066	666000	1346300	P156	664900	1353100
P067	666000	1346000	P157	664900	1353200
P068	666100	1346000	P158	665000	1353200
P069	666100	1344800	P159	665000	1353300
P070	666200	1344800	P160	665100	1353300

Apex	X_ITRF2008	Y_ITRF2008	Apex	X_ITRF2008	Y_ITRF2008
P071	666200	1344600	P161	665100	1353400
P072	666300	1344600	P162	665200	1353400
P073	666300	1344500	P163	665200	1353500
P074	666500	1344500	P164	665300	1353500
P075	666500	1343800	P165	665300	1353600
P076	666400	1343800	P166	665400	1353600
P077	666400	1343300	P167	665400	1353700
P078	666500	1343300	P168	665600	1353700
P079	666500	1343000	P169	665600	1353800
P080	666700	1343000	P170	665700	1353800
P081	666700	1342400	P171	665700	1353900
P082	667500	1342400	P172	665900	1353900
P083	667500	1341200	P173	665900	1354200
P084	666300	1341200	P174	666200	1354200
P085	666300	1341400	P175	666200	1354300
P086	666200	1341400	P176	666400	1354300
P087	666200	1341600	P177	666400	1354500
P088	665900	1341600	P178	666500	1354500
P089	665900	1342100	P179	666500	1355000
P090	666100	1342100	P180	666600	1355000

Note: ITRF 2008 BFTM Datum, UTM 30 N

Source: Orezone

Table 4.2.2 Bomboré II Permit Boundaries

Apex	X_TF	Y_TF
A	666100	1357300
B	669800	1357300
C	669800	1353400
D	668000	1353400
E	668000	1353600
F	667900	1353600
G	667900	1353700
H	667800	1353700
I	667800	1353800
J	667700	1353800
K	667700	1353900

Apex	X_TF	Y_TF
L	667400	1353900
M	667400	1354000
N	667100	1354000
O	667100	1355000
P	667000	1355000
Q	667000	1355100
R	666600	1355100
S	666600	1355000
T	666500	1355000
U	666500	1354500
V	666400	1354500
W	666400	1354300
X	666200	1354300
Y	666200	1354200
Z	666100	1354200

Note: ITRF 2008 BFTM Datum, UTM 30 N

Source: Orezone

Table 4.2.3 Bomboré III Permit Boundaries

Apex	X_TF	Y_TF
0	663700	1352800
1	664400	1352800
2	664400	1353300
3	665000	1353300
4	665000	1353200
5	664900	1353200
6	664900	1353100
7	664800	1353100
8	664800	1353000
9	664700	1353000
10	664700	1352800
11	664500	1352800
12	664500	1352600
13	664400	1352600
14	664400	1352200
15	664300	1352200

Apex	X_TF	Y_TF
16	664300	1351900
17	664200	1351900
18	664200	1351600
19	664000	1351600
20	664000	1351300
21	663900	1351300
22	663900	1350800
23	664000	1350800
24	664000	1350300
25	663900	1350300
26	663900	1350100
27	663800	1350100
28	663800	1349200
29	663900	1349200
30	663900	1349000
31	664000	1349000
32	664000	1348600
33	664100	1348600
34	664100	1348300
35	664000	1348300
36	664000	1348000
37	664100	1348000
38	664100	1347600
39	664000	1347600
40	664000	1347000
41	663800	1347000
42	663800	1346600
43	663900	1346600
44	663900	1346400
45	664000	1346400
46	664000	1346200
47	664100	1346200
48	664100	1346100
49	664200	1346100
50	664200	1346000
51	664300	1346000

Apex	X_TF	Y_TF
52	664300	1345800
53	664400	1345800
54	664400	1344800
55	664600	1344800
56	664600	1344100
57	664700	1344100
58	664700	1343800
59	664800	1343800
60	664800	1343600
61	664900	1343600
62	664900	1343500
63	665000	1343500
64	665000	1343400
65	665300	1343400
66	665300	1343500
67	665800	1343500
68	665800	1343100
69	665900	1343100
70	665900	1342600
71	666700	1342600
72	666700	1343000
73	666500	1343000
74	666500	1343300
75	666400	1343300
76	666400	1343800
77	666500	1343800
78	666500	1344500
79	666300	1344500
80	666300	1344600
81	666200	1344600
82	666200	1344800
83	666100	1344800
84	666100	1346000
85	666000	1346000
86	666000	1346300
87	665900	1346300

Apex	X_TF	Y_TF
88	665900	1347400
89	666200	1347400
90	666200	1348500
91	666300	1348500
92	666300	1348700
93	666400	1348700
94	666400	1348800
95	668800	1348800
96	668800	1342300
97	662300	1342300
98	662300	1349200
99	663700	1349200

Note: ITRF 2008 BFTM Datum, UTM 30 N

Source: Orezone

Table 4.2.4 Bomboré IV Permit Boundaries

Apex	X_TF	Y_TF
A	668400	1353400
B	669000	1353400
C	669000	1348800
D	666700	1348800
E	666700	1349000
F	666800	1349000
G	666800	1349600
H	666900	1349600
I	666900	1349900
J	666600	1349900
K	666600	1350000
L	666500	1350000
M	666500	1350200
N	666300	1350200
O	666300	1350300
P	666200	1350300
Q	666200	1351000
R	666900	1351000
S	666900	1351100

T	667000	1351100
U	667000	1351200
V	667100	1351200
W	667100	1351300
X	667200	1351300
Y	667200	1351400
Z	667400	1351400
AA	667400	1351500
AB	667500	1351500
AC	667500	1351600
AD	667600	1351600
AE	667600	1351700
AF	667700	1351700
AG	667700	1351900
AH	667800	1351900
AI	667800	1352500
AJ	667900	1352500
AK	667900	1352800
AL	668000	1352800
AM	668000	1352900
AN	668300	1352900
AO	668300	1353100
AP	668400	1353100

Table 4.2.5 Bomboré V Permit Boundaries

Apex	X_ITRF	Y_ITRF
A	661000	1348700
B	661600	1348700
C	661600	1349200
D	662300	1349200
E	662300	1342300
F	666100	1342300
G	666100	1342100
H	665900	1342100
I	665900	1341600
J	666200	1341600

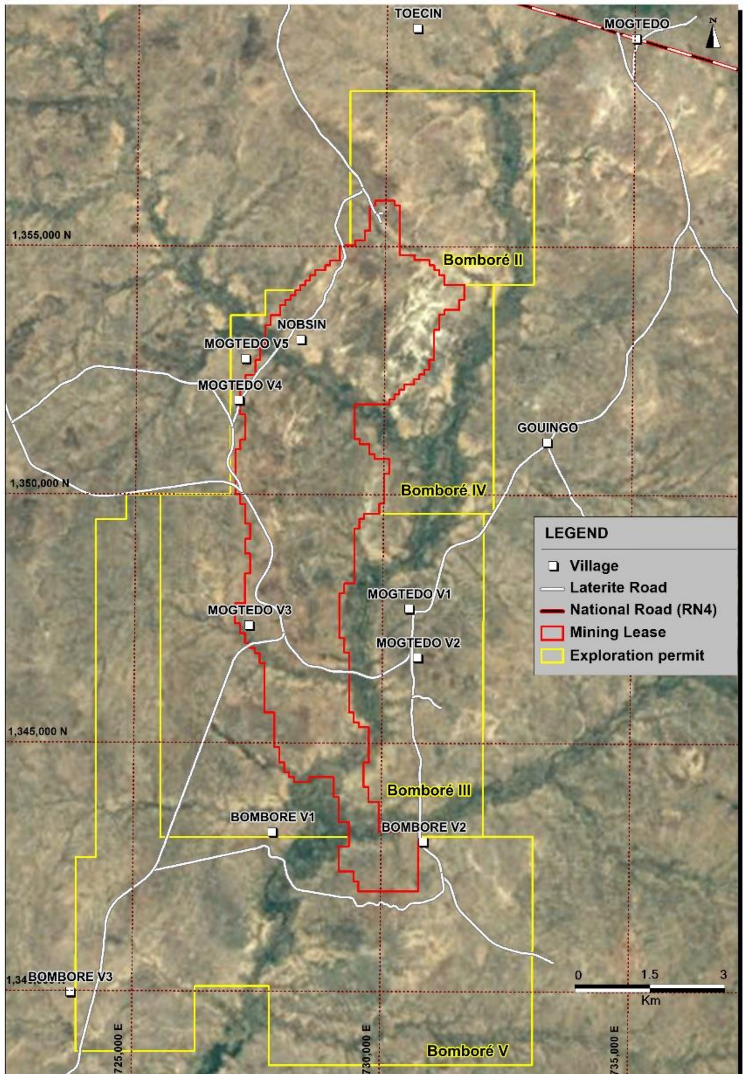
Apex	X_ITRF	Y_ITRF
K	666200	1341400
L	666300	1341400
M	666300	1341200
N	667500	1341200
O	667500	1342300
P	669800	1342300
Q	669800	1337700
R	664500	1337700
S	664500	1339300
T	663000	1339300
U	663000	1338000
V	660600	1338000
W	660600	1341900
X	661000	1341900

The Bomboré Mining Permit is registered in the name of Orezone Bomboré SA, a 90%-owned subsidiary of Orezone. The Bomboré Mining Permit was granted by way of Decree No. 2016-1266/PRES/PM/MEMC/MINEFID/MEEVCC dated 30 December 2016 and is valid for an initial tenure of 10.7 years but can be extended if the mine life is extended beyond what was initially applied for. The Bomboré Mining Permit was extended to cover a total area of 28,870 ha by way of Decree No. 2021-0144/PRES/PM/MINEFID/MEMC/MEEVCC dated 23 March 2021; this extension has not changed the initial tenure of 10.7 years.

The Bomboré II, Bomboré III, Bomboré IV and Bomboré V Exploration Permits are registered in the name of Orezone Inc. SARL, a 100%-owned subsidiary of Orezone. The Bomboré II, III and IV permits were granted on 17 January 2017, were renewed for a third three-year period on 29 March 2023, 23 May 2023 and 22 March 2023, respectively. They are valid until 16 January 2026 when they can be renewed on an exceptional basis. The Bomboré V permit was granted on 24 November 2020 and is valid until 23 November 2023 when it will be renewable for the second of three possible three-year terms.

The Mineral Resources reported in this Report are essentially located within the Bomboré Mining Permit (Figure 4.2.1), with two small deposits on the Bomboré III Exploration Permit (P17N and the northeast extension of the P17S deposit along the P17 mineralized trend).

Figure 4.2.1 Bomboré Tenements



4.3 Underlying Agreements

The current Property was originally covered by a prospecting authorization covering 605,800 ha, granted to Générale de Mines et de Carrières (GMC) in 1989. In January 1994, following changes in the Mining Act in 1993, a modified exploration permit covering 210,800 ha was issued to GMC.

Channel Mining (Barbados) Company, Ltd. (Channel) entered into an option agreement with GMC in 1994 giving it a 90% working interest in the exploration permit, leaving GMC with a 10% carried interest. In the summer of 1997, GMC converted its 10% interest into Channel shares.

A sub-option agreement between Channel and Solomon Resources Limited (Solomon) allowed Channel to secure financing for further exploration. By the end of 1997, Solomon had earned a 45% interest, leaving Channel with a 45% interest in the permit, assuming a 10% free-carried interest owned by the Government if the project were to be developed. The exploration permit was renewed in early 1998.

In 1999, Placer Dome (Africa) Inc. (Placer) reached an agreement to earn a 20% interest in the exploration permit but never fulfilled the conditions to earn in. In July 2001, the exploration permit was reduced to 150,000 ha upon renewal.

Orezone's rights to the Property arise from an initial option agreement signed in 2002 by Orezone's predecessor Orezone Resources Inc. (ORINC) with Channel and Solomon granting ORINC the right to earn a 50% interest in the Property. In 2004, the original Bomboré exploration permit expired and a new Bomboré I exploration permit covering 25,000 ha was granted to Société Orezone, a subsidiary of ORINC, by the MEMC on 17 February 2004. ORINC earned its 50% interest in the Property by issuing 150,000 common shares, making a C$40,000 payment, and spending C$2 million on exploration before 17 January 2007. The Bomboré I exploration permit was renewed on 14 May 2007 and reduced to 10,450 ha.

On 3 September 2008, ORINC announced that it had purchased the remaining interest in the Bomboré I exploration permit from Channel and Solomon in consideration of one million common shares of ORINC (ORINC press release dated 3 September 2008).

On 25 February 2009, ORINC and IAMGOLD Corporation (IAMGOLD) announced that IAMGOLD had acquired ORINC pursuant to a plan of arrangement under the Canada Business Corporations Act. As part of the plan of arrangement, a new exploration company, Orezone Gold Corporation, was incorporated and acquired certain assets and liabilities of ORINC, including the Bomboré I exploration permit. There was no further relationship with IAMGOLD after the transaction closed other than IAMGOLD becoming a shareholder of Orezone.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Bomboré Gold Mine is located in Ganzourgou Province, Burkina Faso, approximately 85 km east of the capital city of Ouagadougou. The Property is easily accessible from Ouagadougou via paved national highway N4. The Property entrance is accessed from highway N4 via a 5 km lateritic road which is maintained by Orezone.

Although Burkina Faso is landlocked, road and rail links to the West African coast are well established and currently support mining and other activities in Burkina Faso and its regional neighbours. Good road links exist to the ports of Lome in Togo, Tema and Takoradi in Ghana and road and rail links to Abidjan in the Ivory Coast. These routes have been used extensively through the construction and operations phase for delivery of equipment and materials.

The International Airport in Ouagadougou is serviced on a regular basis by several international and regional carriers including Air France, Brussels Airlines, Royal Air Maroc, Ethiopian Airlines and Turkish Airlines. Regular air cargo services are available, and the airport also accepts charter cargo flights. A new international airport is planned for and in the early stages of development to the northeast of Ouagadougou.

5.2 Climate

The local climate consists of dry and wet seasons. It is common for rain to occur from April through October; however, the highest concentration of rainfall events occurs between late June and late September. On average, approximately 800 mm of rainfall occurs annually, typically in short bursts of heavy rain. Construction and mining operations can be scheduled year-round, with short delays during heavy rainfall events expected.

Temperatures range from a low of about 10°C in December and January to highs of about 43°C in March and April with average daily temperatures in the range of 23° to 33°. Between the end of the wet season and March the north-easterly trade winds (the Harmattan) bring dust down from the Sahara (the Harmattan) resulting in reduced visibility.

5.3 Local Infrastructure and Resources

Ouagadougou is a typical inland West African city with limited heavy industry but with an established network of companies and suppliers servicing the regional mining industry. Several large regional contracting companies maintain a presence in the country and are equipped with, or can readily mobilize, the resources necessary to construct and support mine development.

The Property is within 15 km of the regional town of Mogtédo, with a population greater than 15,000. The town is developing rapidly with many substantial multi-storey concrete block buildings established or under construction. Most of the semi-skilled and unskilled labour required for project development and operations is sourced from Mogtédo and surrounding communities.

Mine infrastructure is discussed in Section 18.

5.4 Physiography

The topography of the property is generally flat with low hills, in the order of 30 m to 50 m in elevation. The land surface consists of outcrop, sub-crop, and hard ferruginous lateritic cap rock that form a gently southwesterly-sloping plateau.

The seasonal Bomboré River crosses the Project area along a north-northeast south-southwest course and its tributaries follow northeast and northwest directions. The Bomboré River is a tributary of the Nakanbé River. The drainage pattern is rectangular–dendritic, reflecting late fracture systems trending north-south, east-west, and northwest-southeast and the predominantly north-northeast trend of the stratigraphic units.

Vegetation in uncultivated areas comprises mostly savannah woodlands, with dense bush growing only near streams and rivers. Farmers cultivate staple crops such as millet, rice, sorghum, maize corn, and cash crops, such as cotton, sesame, and groundnuts. Deforestation is widespread over the permit area. Wildlife is mostly restricted to small game and birds, but snakes are common, and a few monkey sightings have been reported. The south-west corner of the property lies approximately 11 km away from the classified forest of the Nakanbé River (Volta Blanche or White Volta River). As it flows southwards toward Ghana, the Nakanbé marks the border of this protected area.

Figure 5.4.1 shows typical landscapes during the dry and wet seasons.

Figure 5.4.1 Wet and Dry Season Landscapes

6.0 HISTORY

6.1 Exploration History

The Property was originally covered by a prospecting authorization covering 605,800 ha, granted to Générale de Mines et de Carrières (GMC) in 1989. In January 1994, following changes in the Mining Act in 1993, a modified exploration permit covering 210,800 ha was issued to GMC.

Channel Mining (Barbados) Company, Ltd. (Channel) entered into an option agreement with GMC in 1994 giving it a 90% working interest in the exploration permit, leaving GMC with a 10% carried interest. In the summer of 1997, GMC converted its 10% interest into Channel shares.

Orezone's rights to the Property arise from an initial option agreement signed in 2002 by Orezone's predecessor Orezone Resources Inc. (ORINC) with Channel and Solomon granting ORINC the right to earn a 50% interest in the Project. In 2004, the original Bomboré exploration permit expired and a new Bomboré I exploration permit covering 25,000 ha was granted to Société Orezone, a subsidiary of ORINC, by the MEMC on 17 February 2004.

ORINC earned its 50% interest in the Project by issuing 150,000 common shares, making a C$40,000 payment, and spending C$2 million on exploration before 17 January 2007. The Bomboré I exploration permit was renewed on 14 May 2007 and reduced to 10,450 ha. On 3 September 2008, ORINC announced that it had purchased the remaining interest in the Bomboré I exploration permit from Channel and Solomon in consideration of one million common shares of ORINC (ORINC press release dated 3 September 2008).

On 25 February 2009, ORINC and IAMGOLD Corporation (IAMGOLD) announced that IAMGOLD had acquired ORINC pursuant to a plan of arrangement under the Canada Business Corporations Act. As part of the plan of arrangement, a new exploration company, Orezone Gold Corporation, was incorporated and acquired certain assets and liabilities of ORINC, including the Bomboré I exploration permit. There was no further relationship with IAMGOLD after the transaction closed other than IAMGOLD becoming a shareholder of Orezone Gold Corporation.

In 2007, ORINC commissioned Met-Chem Canada Inc. (Met-Chem) to prepare an initial mineral resource statement for the Property. This mineral resource estimate considered drilling information to March 2007 and is documented in a Technical Report prepared by Met-Chem and dated 28 February 2008. The most recent Technical report for the Property was prepared by Lycopodium Minerals Canada Ltd with an effective date of 26 June 2019.

In January 2021 the Company announced it had secured a financing package totalling $182M and awarded the engineering, procurement, and construction management (EPCM) contract for the process plant construction and commissioning to Lycopodium Minerals Pty Ltd. (Lycopodium).

The Company declared commercial production at Bomboré on 1 December 2022. Delivery of the Bomboré project construction (excluding the third-party managed power plant) was completed on schedule and under budget. Final project construction costs including pre-production mining but excluding power plant totalled US$168.9M, below the project approved budget of US$173.8M.

6.2 Previous Mineral Resource Estimates

In 2007, Orezone commissioned Met-Chem Canada Inc. (Met-Chem) to prepare an initial mineral resource statement for the Property. This mineral resource estimate considered drilling information to March 2007 and is documented in a Technical Report prepared by Met-Chem and dated 28 February 2008.

In June 2008, Orezone commissioned SRK to audit an updated mineral resource model prepared by Orezone. This mineral resource statement included drilling information to May 2008 and was documented in a Technical Report prepared by SRK and dated 26 November 2008.

In July 2010, Orezone commissioned SRK to audit an updated mineral resource model prepared by Orezone. This mineral resource estimate included drilling information to July 2010 and was documented in a Technical Report prepared by SRK and dated 29 November 2010.

In March 2012, Orezone commissioned SRK to update the mineral resource model with assistance of Orezone for the geological and domain modelling. This mineral resource estimate included drilling information to June 2012 and was documented in a Technical Report prepared by SRK and dated 11 October 2012. This report also included technical information and economic parameters used in an earlier PEA completed by GMSI in August 2011.

SRK completed a further resource model in 2013 which was presented in the KCA Technical Report dated 28 April 2015.

In consideration of additional drilling data and a geological re-interpretation of the mineralized domains, coupled with restrictions on the grade modelling of the low-grade domains, RPA completed a new mineral resource estimate for the Project as reported in the Technical Report dated 31 October 2016. This was further refined to produce the 5 January 2017 mineral resource estimate presented in the 23 August 2018 Technical Report.

RPA completed an updated resource model in December 2018 incorporating previously excluded seasonal floodplains and all drilling completed to that date on the high-grade P17S deposit. The effective date of this updated resource estimate remained 5 January 2017, as the changes were marginal.

In 2019, Orezone commissioned P&E Mining Consultants Inc. to update the mineral resource model for the Property with assistance of Orezone for the geological and domain modelling. This mineral resource estimate, which included drilling information to July 2021, was presented in the Technical Report prepared by Lycopodium Minerals Canada Ltd with an effective date of 26 June 2019.

6.3 Production

Orezone declared commercial production on 1 December 2022. Production to 28 March 2023 is summarized in Table 6.3.1.

Table 6.3.1 Production History

Period	Mill Feed Mt	Feed Grade g/t Au	Mill Recovery % Au	Gold Production oz Au
2022	1,019,465	0.92	91.9	27,830
Q1-2023	1,445,693	0.96	92.2	41,301

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

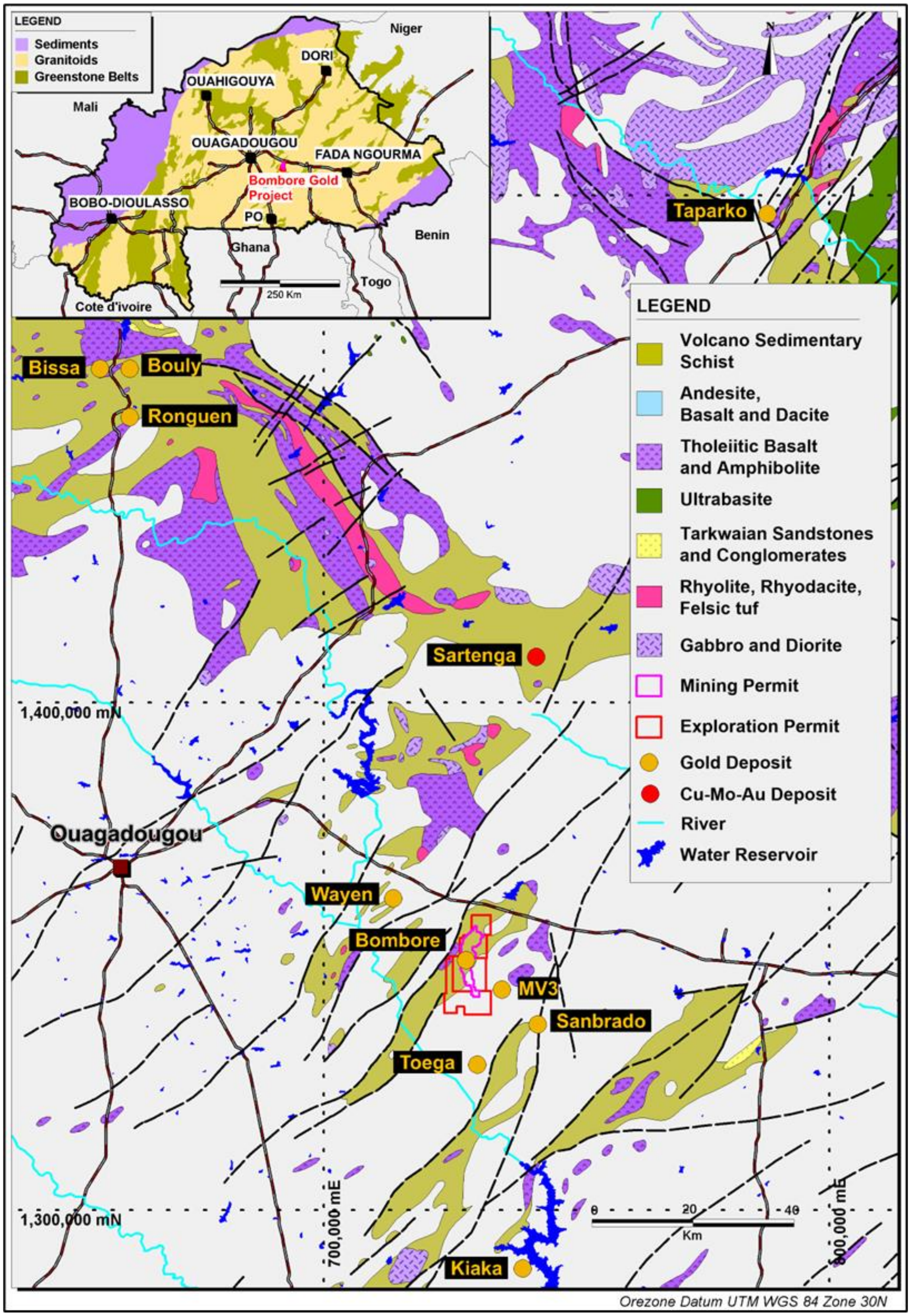
The geology of Burkina Faso is dominated by the Proterozoic Baoulé-Mossi Domain, which corresponds to the eastern portion of the West African Craton. Neoproterozoic to Paleozoic sedimentary rocks cover the west and southeast of the country (inset of Figure 7.1.1).

The Baoulé-Mossi Eburnean orogenic domain contains Birimian (Lower Proterozoic) volcano-sedimentary units arranged in elongated belts and relics of the Archean basement. The belts generally trend north-northeast but form arcuate belts to the north of Ouagadougou. They are bounded by older granite gneiss terrains and have been intruded by syn- to late tectonic granite bodies.

The Birimian Supergroup has been divided into a Lower sequence comprised of wacke, argillite and volcaniclastic rock, and an Upper sequence of basalt with interflow sedimentary rock. Post-Eburnean marine and continental sedimentary rocks unconformably overlie the Lower Proterozoic sequences. The Birimian formations have been affected by three tectono-metamorphic phases with up to greenschist facies metamorphism.

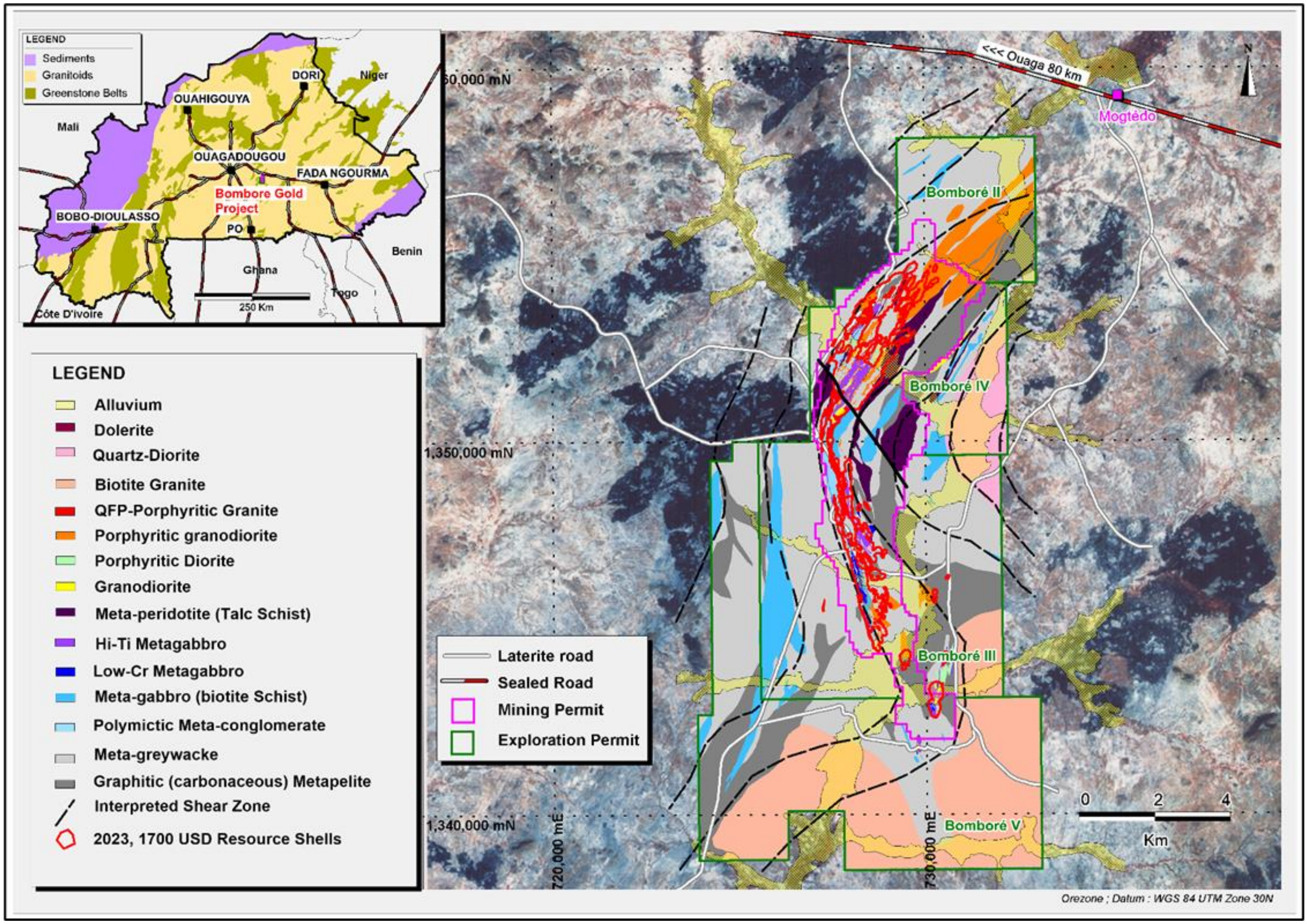
The Project lies in a small northeast-trending belt located to the west of the major Tiébélé-Dori-Markoye Shear Zone that sub-divides the country into domains characterized by different structural patterns. Lithological and structural elements of the Project area are illustrated in Figures 7.1.1 and 7.1.2.

Figure 7.1.1 Regional Geology and Gold Deposits



Source: Orezone (September 2023)

Figure 7.1.2 Local Geology



Source: Orezone (September 2023)

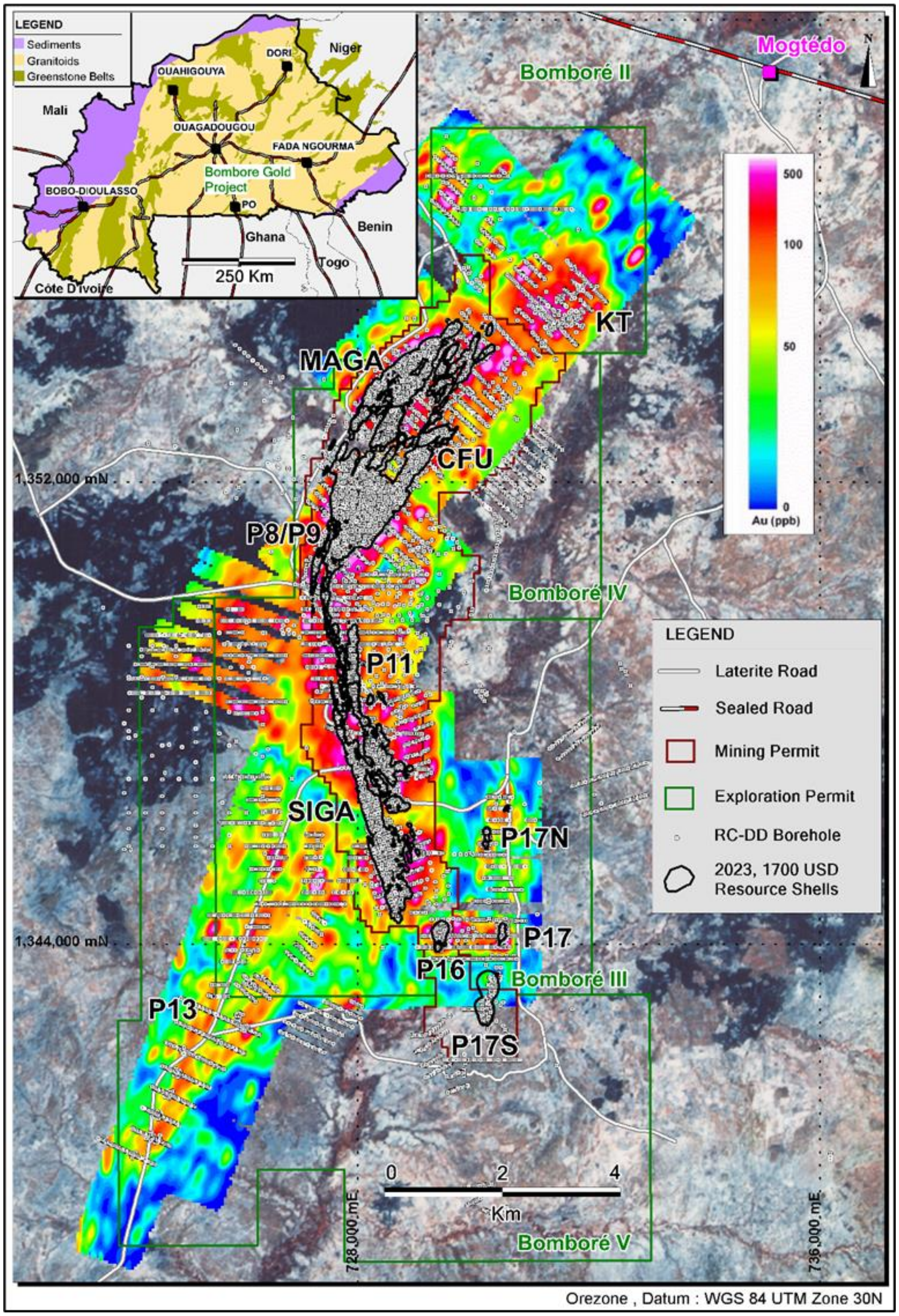
7.2 Property Geology

The Project covers part of a northeast-southwest trending greenstone belt extending for 50 km from the southwest corner to the village of Meguet in the northeast. The permit area is underlain mainly by a meta-sedimentary flysch-type sequence dominated by meta-sandstones with subordinate carbonaceous meta-pelites and polymictic meta-conglomerates. This metasedimentary sequence is intruded by early meta-gabbroic and ultramafic intrusions, and subsequently by syntectonic granodiorite intrusions. Late-tectonic quartz-feldspar porphyries occur as dikes and larger bodies within the greenstone belt. Large biotite granite intrusives are present on the Property to the west and to the south of the greenstone belt that is also moulded on a large quartz diorite intrusive located along the eastern limit of the Project. A syenite intrusion, referred to as the Petite Suisse, is exposed in the west portion of the Property.

The Bomboré Shear Zone (BSZ) is a major, one to three-kilometre thick structure that contains the Bomboré gold mineralization and represents the dominant structural feature of the area. The Bomboré gold mineralization trend is defined by a gold-in-soil anomaly exceeding 0.1 g Au/t (Figures 7.2.1 and 7.2.2), and by the presence of numerous gold showings and orpaillage (artisanal miners) sites. The Bomboré anomaly measures 14 km in length, is several hundred metres in width, and occurs within the BSZ. The main mineralized areas are shown in Figures 7.2.1 and 7..2.2.

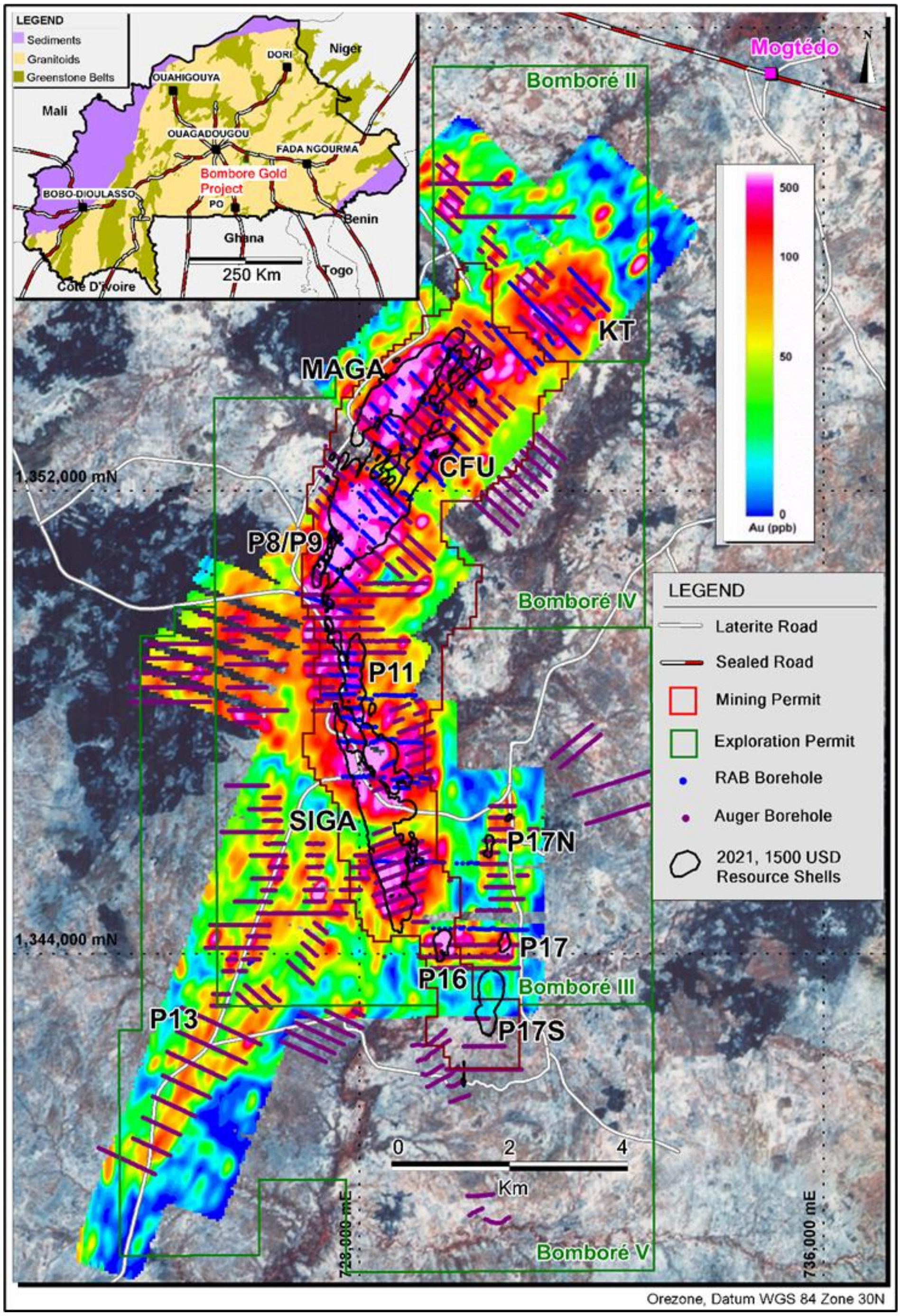
Surface weathering has affected the rocks to an average depth of 35 m to 50 m but can be as deep as 100 m on the P8/P9 and CFU hanging wall, and as shallow as 5 m to 10 m in the P17 area.

Figure 7.2.1 Location of Drill Hole Collars, Gold in Soil Anomolies and Outline of Conceptual Pit Shells



Source: Orezone (September 2023)

Figure 7.2.2 Location of Auger and RAB Drill Hole Collars, Gold in Soils Results and Outline of Conceptual Pit Shells



Source: Orezone (September 2023)

7.2.2 Lithologies

Several lithological units have been recognized by Orezone on the Property, in surface outcrops, drill core, and reverse circulation (RC) chips. The current geological model integrates new information derived from additional drilling, petrographic examinations, and the systematic field X-ray fluorescence (XRF) analyses of all samples.

The main lithological units are described below, from the oldest to youngest based on the current understanding of the Property litho-stratigraphic history. Representative major litho-types, as observed in the drill core, are shown in Figures 7.2.3 to 7.2.6. Plan maps showing the distributions of the lithologies of the Maga to P8/P9 and Siga to P11 areas are shown in Figures 7.2.7 and 7.2.8, respectively.

S4: Meta-Pelite

This unit consists of a sequence of laminated to finely bedded dark grey graphitic or carbonaceous meta-argillite and grey to grey-greenish meta-siltstone and fine meta-sandstone. This sequence can be up to 500 m thick, with a lateral extent >10 km. It is the second most common unit within the greenstone sequence and is interpreted to be the oldest volcano-sedimentary unit recognized in the area. The map pattern suggests that this unit forms regional tight folds with major closures located to the southwest on the Bomboré V Exploration Permit, to the north in the Maga area and to the south in the P16 area. Although primary sedimentary structures and textures can be locally preserved, they are generally overprinted by the regional deformation and metamorphism.

S3: Meta-Sandstone

This unit is the most important within the portion of the greenstone belt underlying the Property and is interpreted to be overlying the meta-argillite unit. It occurs as a sequence up to one kilometre thick dominated by greyish meta-sandstones interbedded with carbonaceous dark grey lamina. Although primary sedimentary structures and textures can be locally preserved, they are generally overprinted by the regional deformation and metamorphism. In thin section, the meta-sandstone beds consist of a quartz-sericite-biotite±graphite (carbonaceous matter) schist with a lepidoblastic to granoblastic texture. The main fabric is a pressure solution cleavage on which primary lithological contacts and early veinlets are typically transposed.

S1: Polymictic Meta-Conglomerate

This unit occurs as elongated lenses adjacent to the meta-sandstone unit. The lenses are typically <100 m thick but can display a kilometric lateral extent. They consist of poorly sorted polymictic meta-conglomerate and conglomeratic lithic meta-sandstone. The lithic clasts consist of meta-sandstone, chert, carbonaceous-graphitic meta-argillite, granite, and quartz, predominantly as granules, pebbles, and cobbles set in a matrix of meta-sandstone. In thin section, the sandy matrix consists mostly of chlorite, sericite, quartz, and calcite with a lepidoblastic to granoblastic texture. The abundant chlorite and carbonate in this unit seems to represent a retrogressive assemblage overprinting the regional metamorphic assemblage; it is responsible for the greenish colour of this unit below the weathering profile.

I3: Mafic Intrusions

This unit intrudes the metasedimentary sequence where it is generally para-conformable to the regional pressure-solution cleavage. The meta-gabbro is characterized by heterogeneous strain, locally with a massive or brecciated texture but in most instances strongly deformed (MI3 sub-unit) with a mylonitic foliation that can be crenulated or micro-folded. Where least-altered and least-deformed, it is greenish and fine to medium grained. It is composed of idiomorphic plagioclases and interstitial pyroxenes, with subordinate hornblende and biotite. In the P11 and Siga areas, gabbro intrusions may contain millimetric blue quartz phenocrysts. This unit is commonly metasomatized and strongly overprinted by a ductile deformation event that has transformed the meta-gabbro into a quartz-biotite-actinolite-albite-calcite-ankerite±pyrite±pyrrhotite schist that is a major host of the Bomboré gold mineralization; this unit displays a characteristic brownish colour below the weathering profile.

I4: Ultramafic Intrusions

Ultramafic intrusion units are present essentially in the northern portion of the Property. Least deformed and least altered units outside of the BSZ consist of massive meta-peridotite, where primary olivine and pyroxene largely retrograde altered to mineral assemblage of talc, asbestos, chlorite, and carbonate pseudomorphs.

Talc schists host gold mineralization in the CFU Deposit area.

IIC: Pre- to Syn-Tectonic Micro-Porphyritic Granodiorite Intrusions

Within the BSZ, a fine grained micro-porphyritic granodiorite occurs as narrow dikes and larger elongated intrusions, mostly on the hanging wall of the main P8/P9, P11, and Siga East Deposits, but also as the main mineralized unit of the P17 and P17S Deposits. This unit seems to be mostly intruding the meta-gabbro unit, and forms with the metasediments and the meta-gabbro a sequence that has been folded prior to or synchronously with the main gold deformation and mineralization event. Most of the rare occurrences of visible gold within the BSZ are associated with this unit.

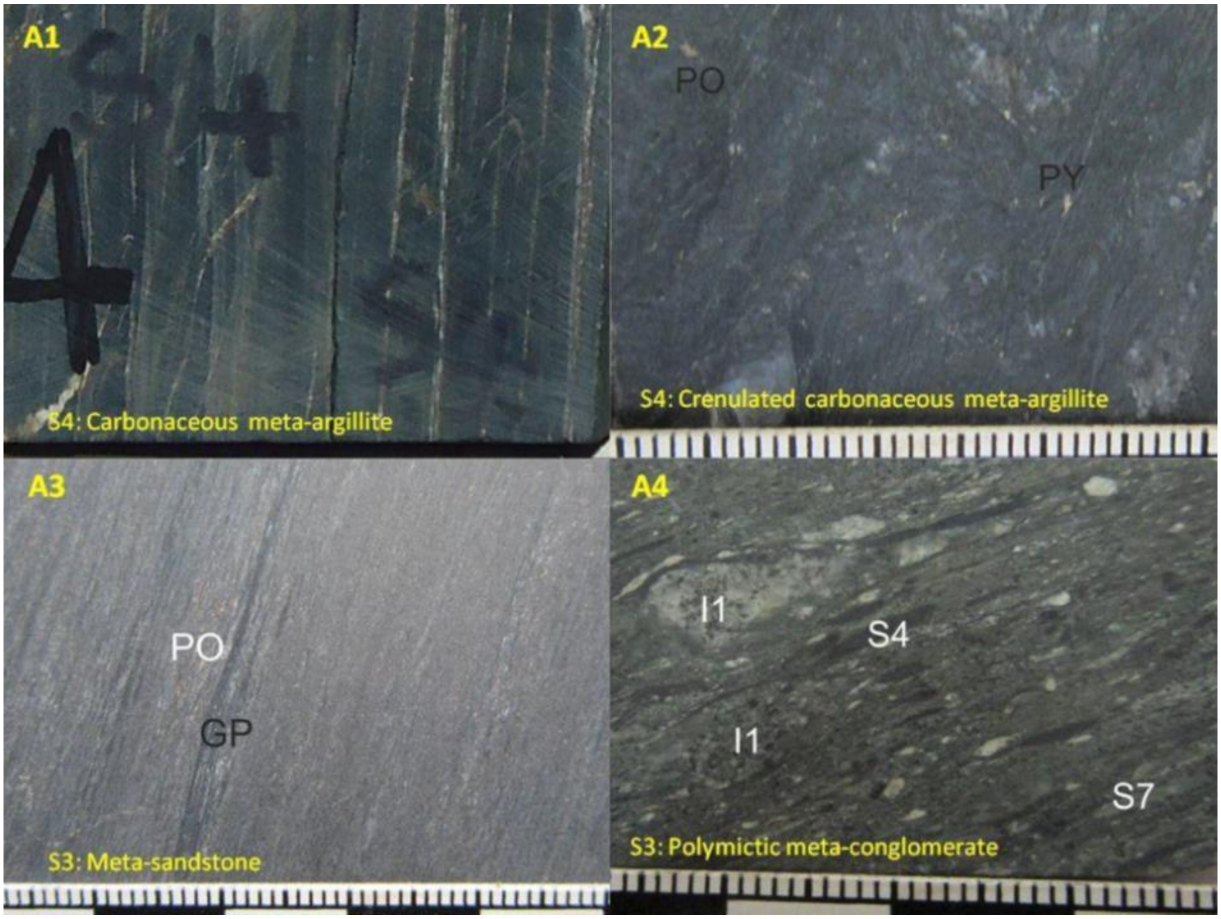
I2: Syn-Tectonic Porphyritic Granodiorite Intrusions

Within the BSZ, a porphyritic granodiorite intrusion characterized by abundant zoned plagioclase phenocrysts up to 12 mm set in a groundmass of fine-grained quartz-biotite-sericite occurs as 1 m to 100 m thick dikes typically at a low counter-clockwise angle to the pre-existing lithological units and fabrics, but also as larger elongated intrusions in the Maga and KT Deposits areas. They are syn-tectonic and pre- to syn-gold mineralization but are less deformed and less well mineralized than the older units that they intrude. The sheared and mineralized porphyritic granodiorite is difficult to distinguish from the sheared and deformed meta-sandstone even in core boreholes.

I1: Late Quartz Feldspar Porphyry Granite Dikes

Within the BSZ, late pale grey fine-grained granitic dikes characterized by abundant corroded quartz and plagioclase-albite phenocrysts set in a microlitic and sericitic ground mass occur as narrow metric (typically one to three metres wide) dikes, mostly in the Maga and P8/P9 area. They are post-tectonic and post-gold mineralization.

Figure 7.2.3 Photographs of the Meta-Sedimentary Units



Source: Orezone (September 2023)

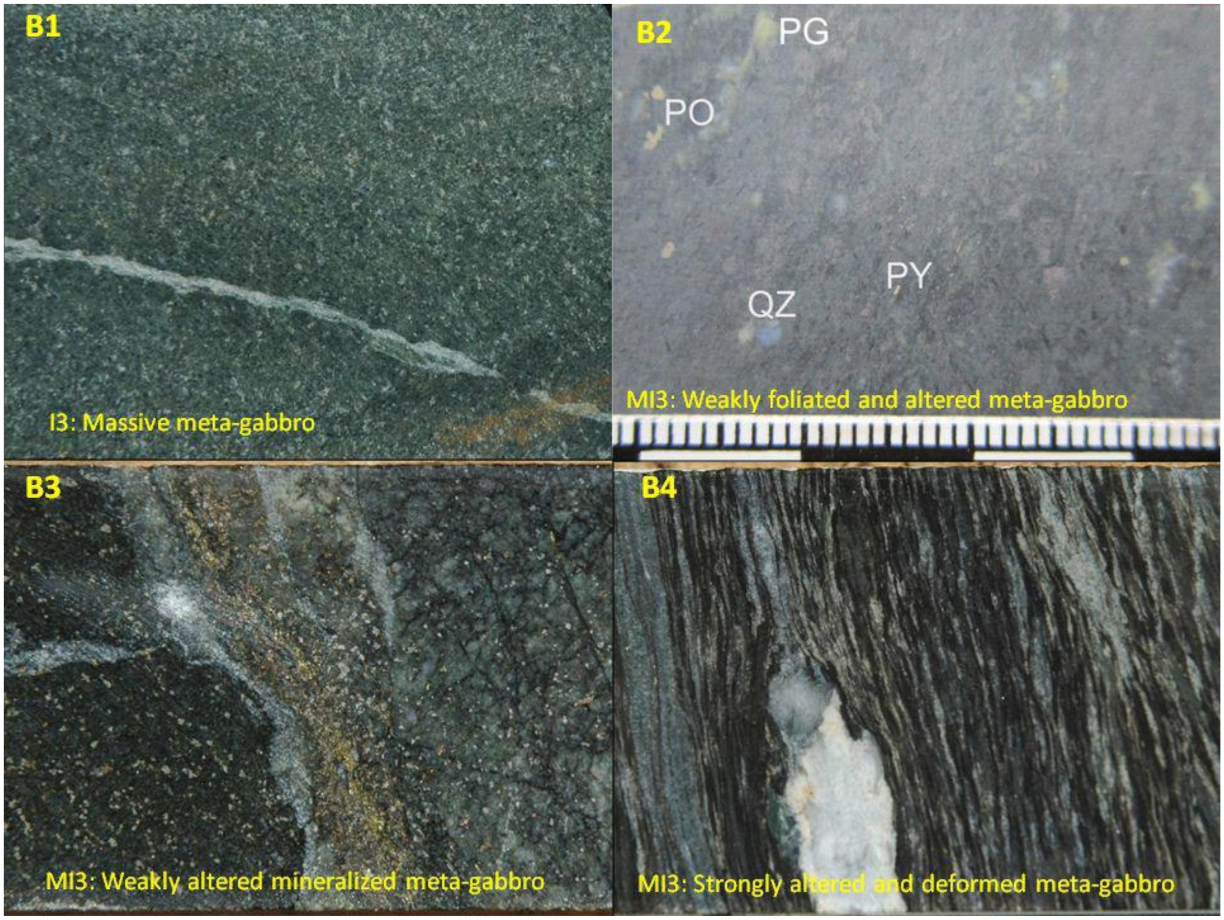
A1: Carbonaceous meta-argillite.

A2: Crenulated carbonaceous meta-argillite. P16, BBD0012, 40.79 m.

A3: Meta-sandstone. Disseminated pyrrhotite (PO) and graphite (GP) laminae. P16, BBD0214, 146.0 m.

A4: Polymictic meta-conglomerate. Lithic clasts of meta-argillite (S4), chert (S7) and felsic intrusive (I1). P8/P9, BBD0069, 75.65 m.

Figure 7.2.4 Photographs of the Meta-Gabbro Units



Source: Orezone (September 2023)

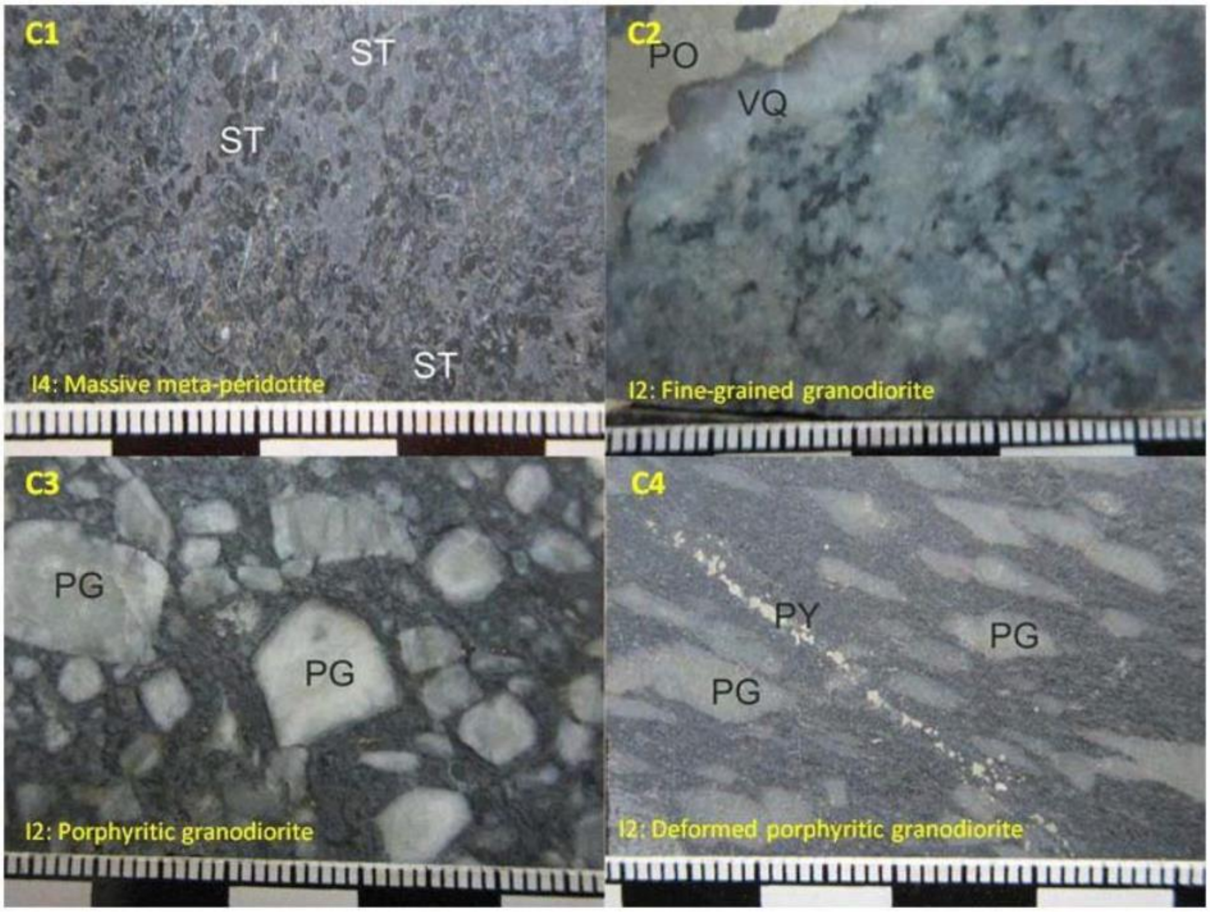
B1: Massive meta-gabbro. P11, BDD0029.

B2: Weakly foliated and altered meta-gabbro with blue quartz (QZ) and altered plagioclase (PG) phenocrysts. Disseminated pyrite (PY) and pyrrhotite (PO). Siga W, BBD0013, 120.29 m.

B3: Weakly altered mineralized meta-gabbro. P11, BBD0029.

B4: Strongly altered biotitic meta-gabbro. P11, BBD0029.

Figure 7.2.5 Photographs of the Peridotite and Granodiorite Units



Source: Orezone (September 2023)

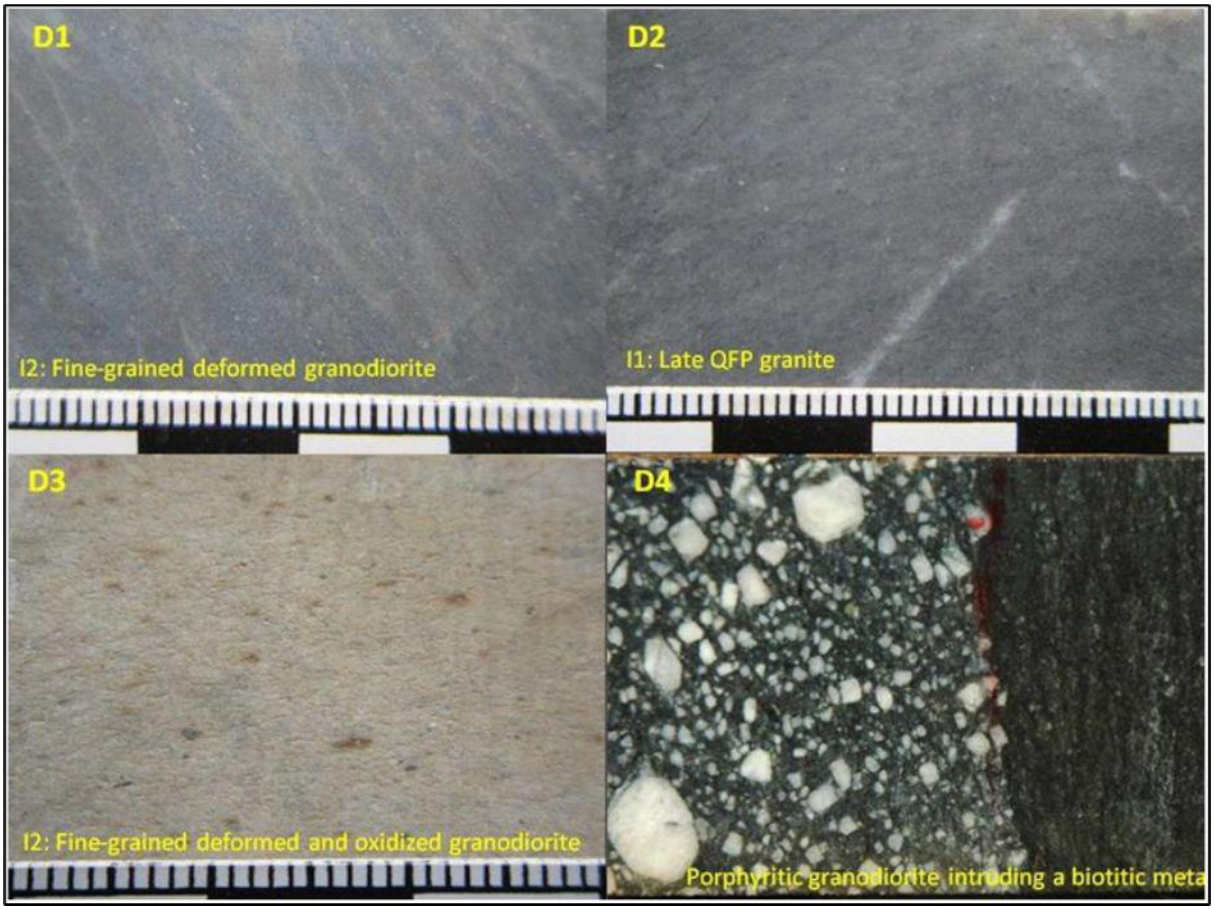
C1: Massive meta-peridotite with serpentine (ST) olivine pseudomorphs. Outcrop AE14, P11 east area.

C2: Fine-grained granodiorite cut by a quartz (VQ) and pyrrhotite (PO) vein. P8/P9, BBD0491, 120.18 m.

C3: Porphyritic granodiorite with large zoned plagioclase (PG) phenocrysts. P8/P9, BBD0612, 53.8 m.

C4: Deformed porphyritic granodiorite. P8/P9, BBD0529, 133.55 m.

Figure 7.2.6 Photographs of the Granodiorite and Granite Units



Source: Orezone (September 2023)

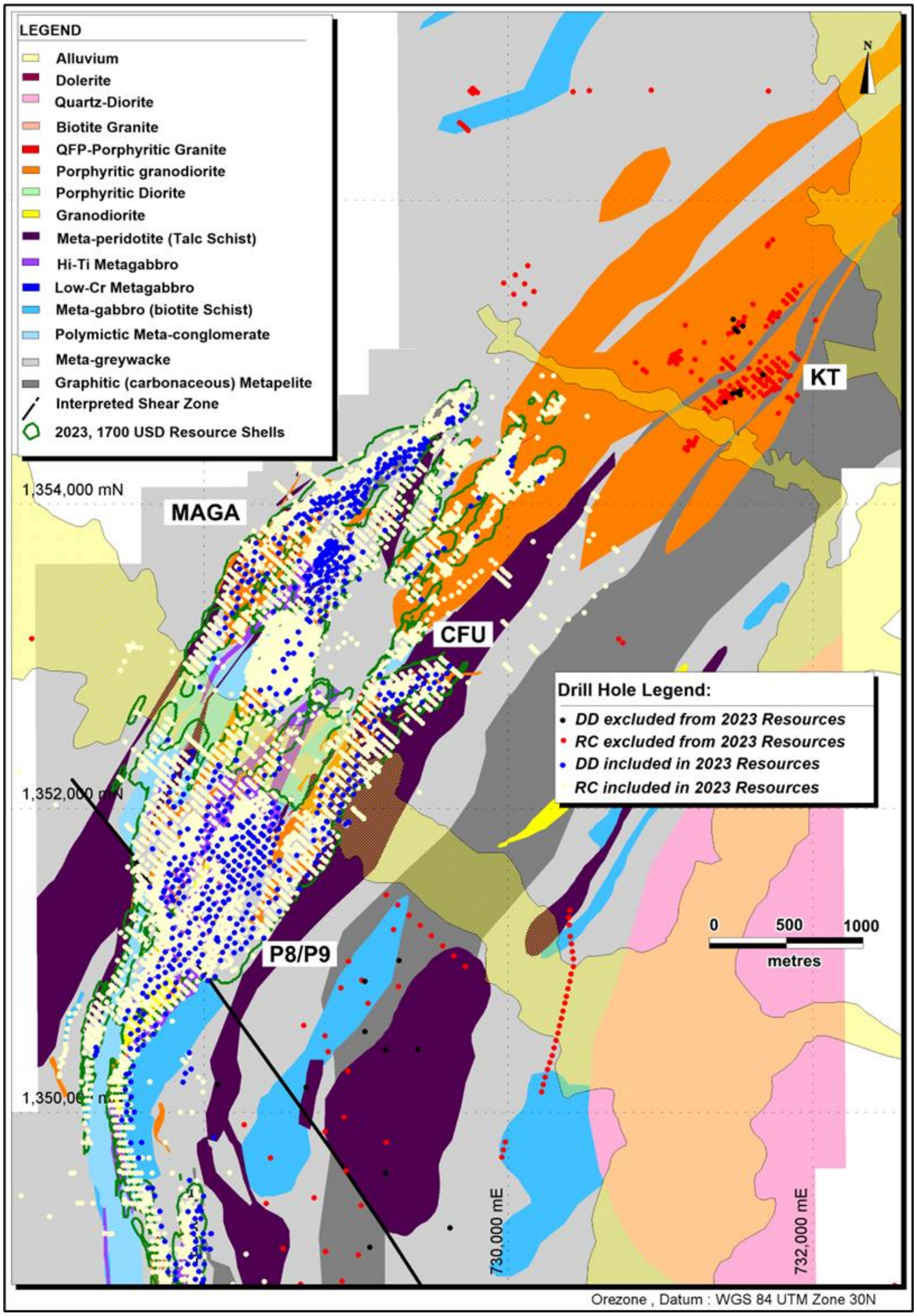
D1: Fine-grained deformed granodiorite. Plagioclase (PG) phenocrysts are altered and deformed. Pyrite (PY) veinlet. Maga, BBD0130, 83.0m.

D2: Late quartz feldspar porphyry (QFP) granite. P8/P9, BBD0039, 89.3 m.

D3: Fine-grained weathered granodiorite. Maga, BBD0055, 34.25 m.

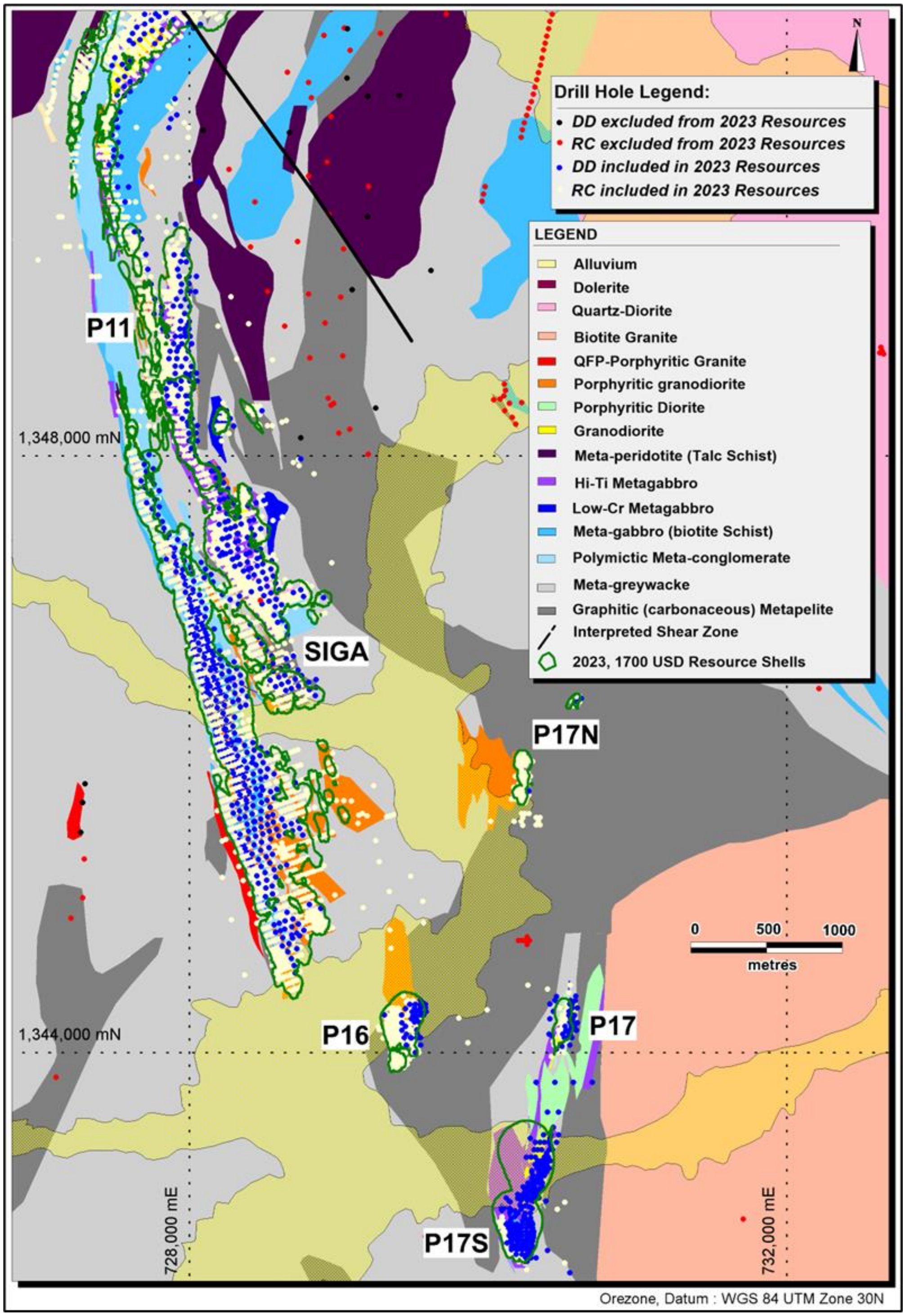
D4: Late QFP granite intruding a deformed biotitic meta-gabbro. P11, BBD0029.

Figure 7.2.7 Geology of the Northern Area Showing Collar Location of Exploration Boreholes



Source: Orezone (September 2023)

Figure 7.2.8 Geology of the Southern Area Showing Collar Location of Exploration Boreholes



Source: Orezone (September 2023)

7.2.3 Structural Geology

The gold mineralization on the Property is hosted in the BSZ, a major north-northwest to north-northeast trending structure. This shear zone has an arcuate shape and extends over tens of km beyond the limits of the Property. It is interpreted as a secondary structure to the Tiébélé-Dori-Markoye Fault, a regional north-northeast trending sinistral fault that represents a major discontinuity in the Birimian rocks, across which regions of contrasting structural styles are juxtaposed.

The BSZ is visible on the aerial photos and exhibits a strong signature on the geophysical (magnetic and induced polarization) maps. The curvature of this shear zone is interpreted to be caused by the moulding of the greenstone belt around the late quartz-diorite intrusion located along the eastern margin of the Property.

The BSZ is oriented 040° in the northern portion of the Property and 340° in the southern portion of the Property (Figure 7.2.9). Most of the syn- to late-tectonic dikes are located within the BSZ, together with the gold mineralized schists and barren quartz vein arrays. The dip of the main foliation and the main lithological contacts is approximately 65° towards the southeast in the northern portion of the Property (Maga, CFU, and P8/P9 Deposits), although it steepens in the Maga footwall area to approximately 75° and is approximately 55° towards the northeast in the southern portion of the Property (P11, Siga East, and Siga West-Siga South deposits).

The main foliation is oriented 360° to 010° and is sub-vertical in the southeast area, where the small satellite deposits (P16, P17N, P17) are located. At P17S, the main foliation is dipping approximately 55° towards the east-northeast and parallels the axial planes of W-shaped folds defined by the sequence of metasandstone-metagrabbro-metagranodiorite.

A north-south fault system is visible on satellite imagery, as well as an east-northeast and a west-northwest system. In addition, breaks in the magnetic data and apparent displacements in the mineralization support the presence of the systems oriented at 070° and 110° (Figures 7.2.9 and 7.2.10) responding to a late faulting event. Some of the gold mineralization appears to have been remobilized along the latter orientations. Fractures and quartz veins (sometimes auriferous) oriented roughly east-west are also noted on outcrops and in trenches.

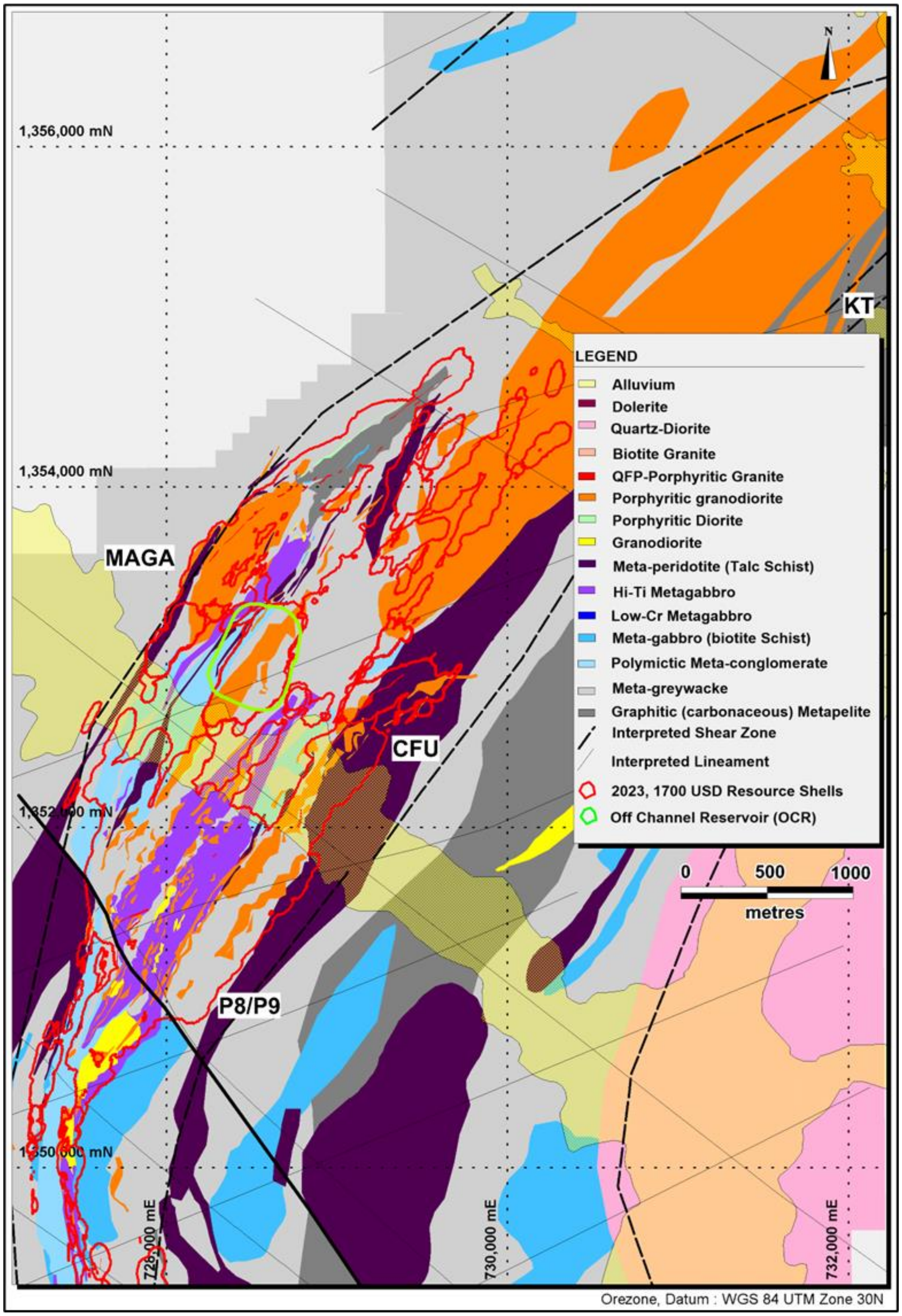
Observations in surface outcrops and borehole geophysical data suggest that rock units within the BSZ exhibit brittle-ductile behaviour with folds, transposition, and an anastomosing pattern typical of a shear zone environment. The presence of brittle structures, such as breccias and quartz veins, have also been recorded in the drill logs and surface maps. In the P11 and Siga areas where the S0-S1 fabric is oriented north-northwest to north, a northeast secondary cleavage has been observed in several localities, suggesting that the dominant northeast trending regional foliation might be overprinting earlier fabrics (S0 and S1).

In outcrop, the foliation fabric contains a stretching lineation plunging moderately (45° to 55°) to the north.

The current geological model, well constrained by oriented core observations, clearly shows that several units, including the auriferous meta-gabbro units, plunge moderately (45° to 55°) to the north-northeast, but recent work suggests that shallower plunges (20° to 35°) are typical of the south and southeast sectors of the project area.

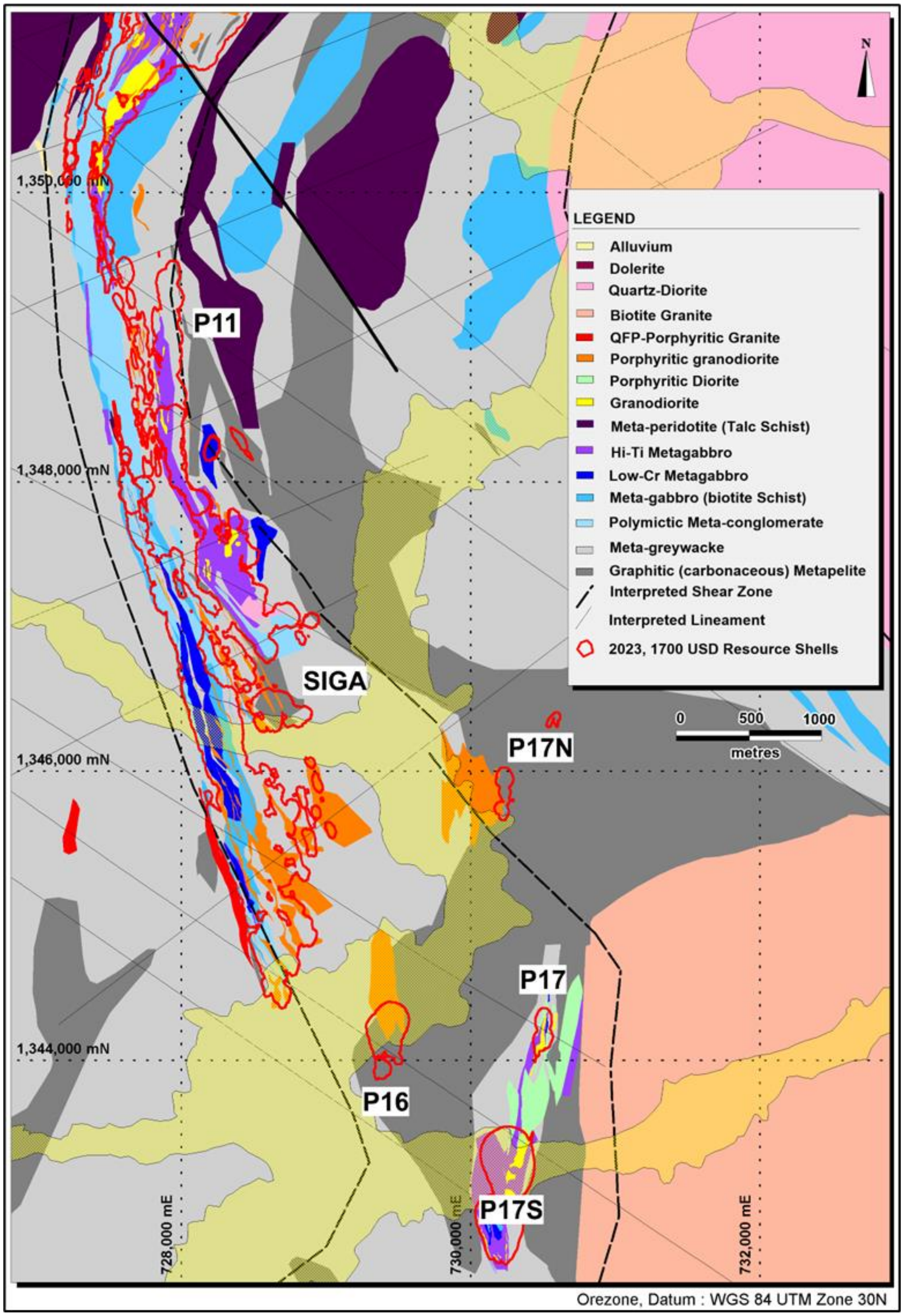
All rock units within the shear zone have been affected by a heterogeneous (brittle) - ductile strain, except the late QFP granitic dikes and some late dolerite dikes.

Figure 7.2.9 Geology of the Northern Area Showng Major Shear Zones and Lineaments



Source: Orezone (September 2023)

Figure 7.2.10 Geology of the Sourthern Area Showng Major Shear Zones and Lineaments



Source: Orezone (September 2023)

7.3 Mineralization

The Bomboré gold deposits occur within the regional BSZ, a major north to northeast trending structure considered as a subsidiary to the Tiébélé-Dori-Markoye Fault. Eleven separate auriferous deposits have been delineated by drilling within the 14-km segment of the BSZ located within the Property. The gold deposits were discovered by tracing gold-in-soil anomalies (Figure 7.1.2) to bedrock by drilling. The deposits are defined by geographic coordinates in Table 7.3.1.

The gold mineralization in the Property area is associated with arrays of structurally controlled quartz veins and veinlets and attendant silica, sulphide, and carbonate alteration developed within the BSZ. Most quartz veins are oriented sub-parallel to the foliation and exhibit strong strain, however, the presence of relatively unstrained quartz veins and breccia in drill core attest the protracted history of vein formation and deformation. Late west trending veins crosscutting the main foliation fabric are also observed. Locally, there is evidence suggesting that gold mineralization was remobilized into northeast and southeast dilation zones associated with late faults.

Table 7.3.1 Location of the Bomboré Gold Zones

Bomboré Gold Deposits	Easting*		Northing*	
	Minimum	Maximum	Minimum	Maximum
Kiin Tanga (“KT”)	731,000	732,000	1,354,350	1,355,500
Maga	727,800	730,500	1,352,600	1,354,900
Colline de Fusille (“CFU”)	728,800	729,800	1,352,600	1,353,050
P8/P9	727,150	729,350	1,350,100	1,352,850
P11	727,200	728,500	1,347,800	1,350,300
Siga East (“SE”)	727,700	729,100	1,346,250	1,348,500
Siga West-Siga South (“SW-SS”)	727,550	729,350	1,344,350	1,348,400
P16	729,250	729,650	1,343,850	1,344,400
P17N	730,100	730,650	1,345,650	1,346,400
P17	730,050	730,550	1,342,600	1,344,450

* UTM Projection – WGS84 datum, Zone 30 North in metres.

The quartz associated with the gold mineralization is milky white to smoky, locally vitreous and may contain tourmaline. The widths of the veins range from two centimetres to one metre, with an average of ten centimetres. The near surface gold mineralization with grades of up to 0.2 g Au/t is pervasive regardless of quartz veining.

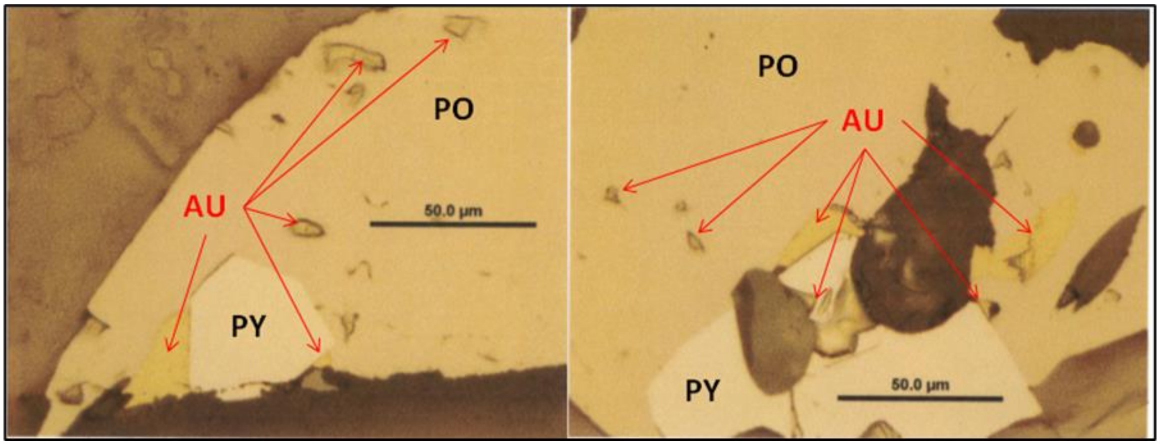
In the fresh zone, the gold mineralization is associated with fine disseminated sulphides, predominantly pyrite. Most of the quartz vein material is barren, as demonstrated by the scrubbing metallurgical test work completed on the Project.

Generally, the gold occurs as fine grain electrum (< 10 µm) but can be visible in outcrop. Artisanal mining over the 1990-2023 period attests to the existence of coarser gold locally. Gold occurs in places as free gold in late smoky quartz tension veinlets and stringers, and is mainly associated with pyrite, pyrrhotite, chalcopyrite, and arsenopyrite. Most sulphides occur as disseminations and fine stringers sub-parallel to the foliation fabric, which suggests development in an active shear zone or re-mobilization. Magnetite and graphite are present locally. Although the sulphide content can be as much as 5%, it is on average only 1% to 2% in fresh (i.e., non-weathered) mineralized rocks.

Gold mineralization is hosted by various rock types, but most commonly in the biotite schist (meta-gabbro), meta-sandstones, and also the granodiorite dikes that intrude the gabbros, although in the Maga North, P16 and P17N areas the meta-argillites are the main host rocks. The syn-tectonic granodioritic intrusives are also mineralized, although to a smaller extent than the biotite schist and the meta-argillites. The meta-conglomerate and meta-peridotite are unfavourable hosts. The meta-gabbro might represent the best chemical trap, given its high iron content (if gold was transported hydrothermally as a thio-complex), as suggested by the pervasive fine pyritic assemblage that is associated with the gold mineralization in the sulphide zone. Although much of the gold Mineral Resources defined within the Project area are hosted in the meta-gabbro unit, the deformed granodiorite, and its contact zone with the metagabbro host is where the higher-grade mineralization is concentrated.

Petrographic work on fresh rock samples in 2008 (Schandl 2008a, b and c) revealed that the gold mineralization is predominantly associated with silica and iron carbonate, although sericite is a ubiquitous and commonly an abundant alteration mineral in many of the gold-enriched rocks. Gold occurs as electrum, native gold, and gold telluride (calaverite). Small gold grains are included in pyrite, in fractures of pyrite grains (Figure 7.3.1), and as free gold in the fine-grained quartz-goethite matrix in the weathered zone. The major sulphides are pyrite and pyrrhotite with subordinate amounts of chalcopyrite, covellite, galena, and arsenopyrite. Pyrrhotite and chalcopyrite are found mainly in the biotite schist and arsenopyrite in the metasedimentary rocks. The gangue of the saprolitic mineralization consists of an assemblage of quartz, sericite, kaolinite, hematite, and goethite.

Figure 7.3.1 Primary Gold Mineralization (Sulphide Zone): Gold (Au) Occurring as Inclusions in Pyrite (PY) and Pyrrhotite (PO)



Source: Orezone (September 2023)

At a cut-off grade of 0.15 g Au/t, the gold mineralization exhibits reasonable continuity over a strike length of approximately ten kilometres. At this cut-off grade, the gold mineralization forms more restricted corridors (500 m to 1,000 m in length and 10 m to 100 m in width), defining anastomosing patterns parallel and slightly oblique to the general trend of the BSZ.

These higher-grade corridors formed the basis for defining geostatistical domains within each litho-domain considered for Mineral Resource estimation. One of the benefits of the 2010 to 2013 infill drilling programs was the delineation of higher-grade sub-domains based on a cut-off grade of 0.5 g Au/t with the broader low-grade domains based on a lower cut-off grade of 0.20 g Au/t. The higher-grade sub-domains have a strike length of up to 500 m and a width typically between 5 m and 30 m.

The typical texture of the gold mineralization host rocks in drill core is shown in Figure 7.3.2.

7.3.2 Sequence of Geological Events

Based on the work completed by Orezone since 2008, the following geological history is interpreted for the Property area:

- Sedimentation of a sequence of carbonaceous argillites followed by sandstone and capped by polymictic conglomerates.
- Intrusion of mafic (gabbro to diorite) sills and dikes, followed by ultramafic (peridotite) intrusives.

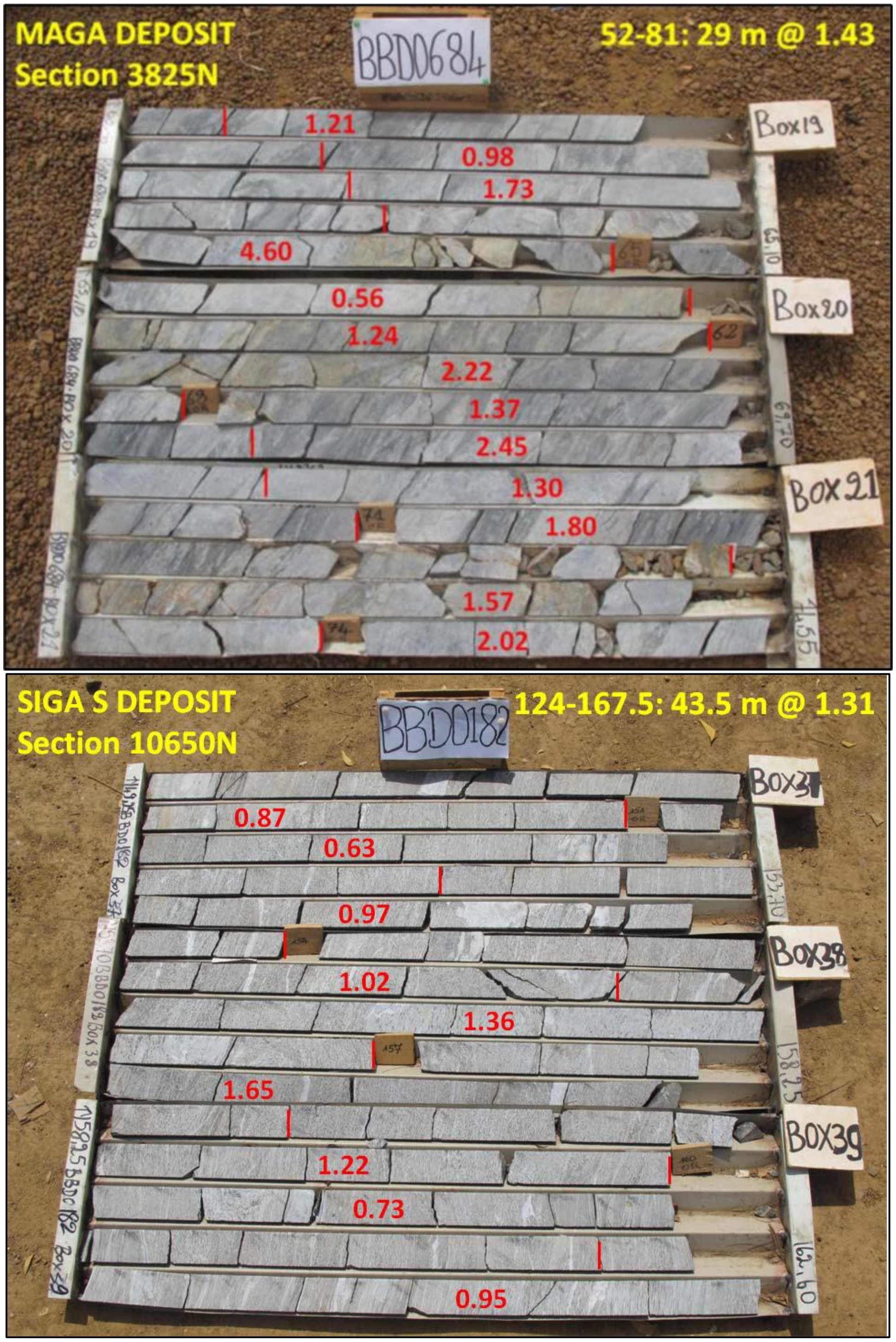
- Regional deformation and prograde metamorphism culminating under greenschist facies, biotite zone conditions, with pre- to syn-tectonic intrusion of fine-grained granodiorite dikes and small intrusion, and later sets of syn-tectonic dikes and larger intrusions of porphyritic granodiorite.

- Syn- to late-metamorphic albite-calcite-tourmaline-biotite-pyrite metasomatism, which is the main gold mineralizing event.

- Late-tectonic intrusion of QFP granitic dikes.

- Retrograde brittle-ductile deformation – local remobilization of gold in late quartz veins.

Figure 7.3.2 Typical Texture of the Gold Mineralization in the Core of the Maga Deposit Meta-Argillite (top) and Siga South Deposit Biotite (bottom)



Source: Orezone (September 2023)

8.0 DEPOSIT TYPES

8.1 Deposit Types

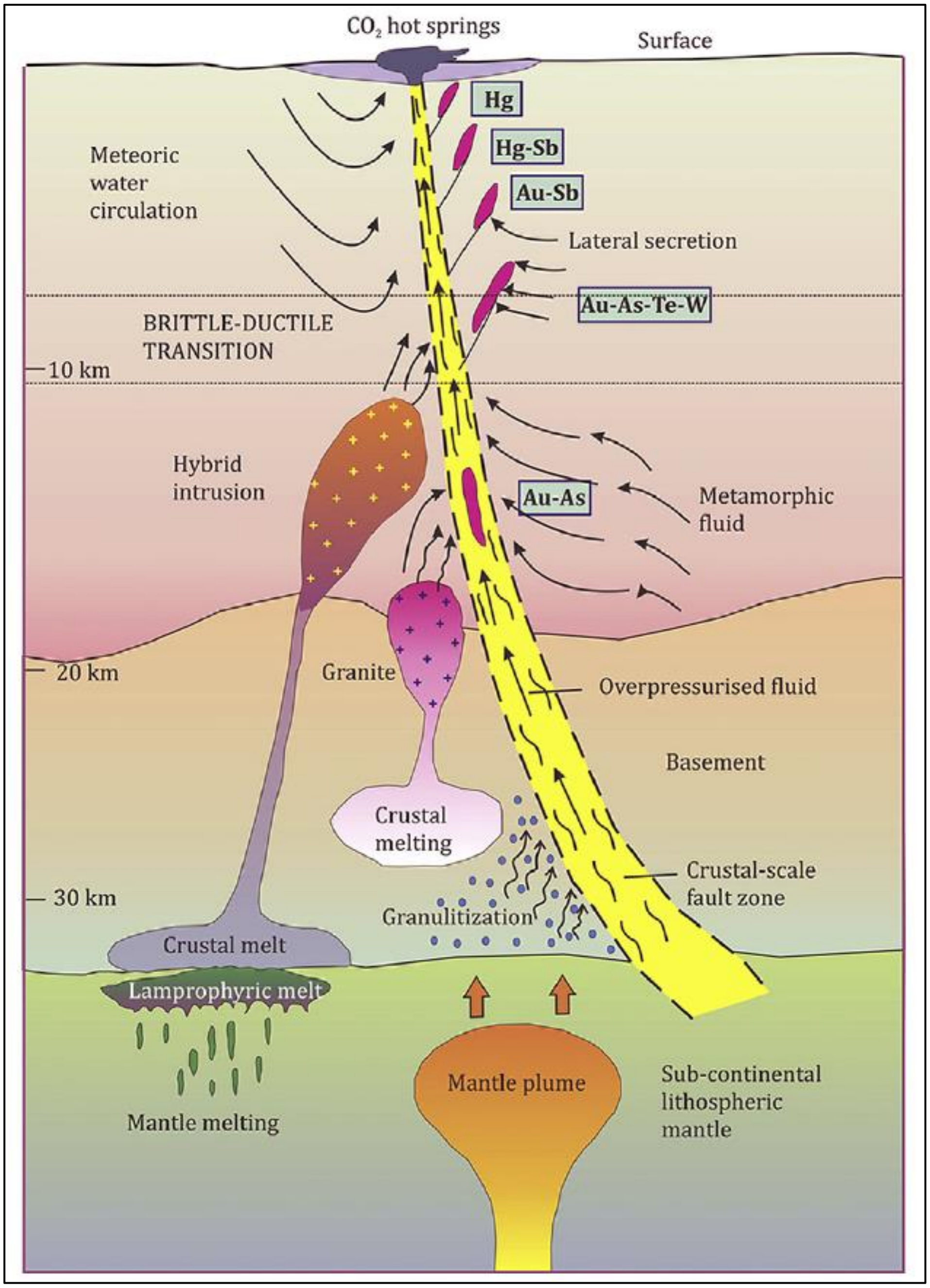
The Bomboré Gold Deposits are orogenic gold deposits (Goldfard and Groves, 2015; Groves and Santosh, 2016 (Figure 8.1.1). Like gold deposits globally and in the late Proterozoic Birimian terrains of West Africa, the Bomboré Gold Deposits exhibit a structural control and associated hydrothermal alteration mineral assemblages. The Deposits represent a large tonnage, low-grade gold mineralization system characteristics similar to other Birimian gold deposits, such as Kiaka in Burkina Faso, Damang, Yamfo-Selwi and Obuasi in Ghana, and Sadiola in Mali.

Orogenic gold deposits are typically products of structurally controlled hydrothermal mineralization systems. Such deposits exhibit strong relationship with regional arrays of major structures and shear zones connected to long-lived, crustal-scale faults and deformation zones. The gold mineralization is typically associated with a network of quartz veins containing subordinate amounts of carbonate, tourmaline, sulphides, and native gold. Gold in these quartz vein networks is typically free milling. The gold mineralization can also be associated with disseminated sulphides in strongly deformed alteration zones. Gold in the disseminated sulphides may be free milling but can also be held in the sulphide minerals and is therefore refractory.

The Bomboré Gold Deposits are essentially stratabound disseminated sulphide bodies hosted preferentially in the meta-gabbro, meta-argillite, and granodiorite lithologies. These rock units are interpreted to have acted as preferential gold traps during syntectonic deformation and metasomatism, due to their strong geochemical and rheological contrasts with the surrounding rocks (Hagemann and Cassidy, 2000).

The wet paleoclimate that preceded the current semi-arid climate in Burkina Faso resulted in extensive surface oxidation of bedrock and a deep weathering profile. Oxidized, weathered bedrock can occur up to a vertical depth of 100 m below surface. The gold mineralization occurs in both the surface oxide zone and the underlying sulphide zone. In the oxide zone, gold typically occurs in a free milling form. However, gold is grind-sensitive in the sulphide zone.

Figure 8.1.1 Integrated Model for Orogenic Gold Mineralization



Sources: Goldfarb and Groves (2015); Groves and Santosh (2016).

Figure Description: Schematic representation of the variety of proposed models for orogenic gold and fluid sources in the crust: from meteoric water circulation and lateral secretion, magmatic-hydrothermal fluid exsolution from various granite intrusion types, to granulitization and prograde metamorphic devolatilization processes during orogeny. The gold-bearing fluids ascend along crustal-scale faults (Tiébélé-Dori-Markoye Fault) and become trapped in splays (Bomboré Shear Zone), where they cool and mix with surface-derived fluids (i.e., meteoric waters) to form gold deposits.

9.0 EXPLORATION

The following is taken from the final land tenure report filed by Orezone for the Bomboré I exploration permit (Derra and Tamani, 2016) and has been updated to cover exploration activities on the current mining tenements. A summary of the activities undertaken at the Property from January 2003 to April 2023 by Orezone is presented in Table 9.3.1. Additional project development work is included in the summary, such as metallurgical test work, geotechnical drilling, and environmental studies.

9.1 Original Bomboré Exploration Permit

In 2002, Orezone entered into an option agreement with Channel and Solomon and assumed the funding and execution of exploration activities on the 150,000 ha Bomboré exploration permit until its expiry in January 2004. Exploration activities during this period consisted of data compilation and a RC drilling programme (Ackert, 2004). The work is described in a series of Orezone reports (Zongo, 2003a and b; and Marquis, 2003).

9.2 Historical Bomboré I and Toéyoko Exploration Permits

In February 2004, Orezone was granted the 25,000 ha Bomboré I exploration permit, which covered the most prospective portion of the former 150,000 ha Bomboré exploration permit, i.e. the BFT area. On 13 July 2011, Orezone was granted the 6,300 ha Toéyoko permit, adjacent to the Bomboré I permit and that covered the prospective extension of the BFT litho-stratigraphic sequence towards the southwest.

9.3 Current Bomboré Mining Lease and Bomboré II, Bomboré III, Bomboré IV and Bomboré V Exploration Permits

On 30 December 2016, Orezone was granted the 2,500 ha Bomboré Mining Lease, and on 17 January 2017, Orezone was granted the 1,815 ha Bomboré II, 4,810 ha Bomboré III and 1,235 ha Bomboré IV exploration permits which covered portions of the former Bomboré I permit that were outside of the limits of the Bomboré Mining Lease. On 24 November 2020, Orezone was granted the 4,618 ha Bomboré V exploration permit that covered the most prospective portion of the Toéyoko exploration permit, which expired on 12 July 2020.

Table 9.3.1 Summary of Exploration on the Bomboré Property

Period	Exploration Activities and Studies
2003	• RC drilling: Mankarga grid: eight boreholes (614 m); P8/P9: 11 boreholes (747 m); Kiin Tanga: 13 boreholes (640 m).
2004	• Compilation work. • Report on the 2003 RC drilling programme.

Period	Exploration Activities and Studies
2005	• 217 RC boreholes (13,829 m) at P8/P9, Maga and Kiin Tanga.
2006	• Establishment of a pair of trigonometric beacons in P8/P9 area and of survey control points from Kiin Tanga to Siga. • Survey of all RC and core boreholes that could be found. • Photogrammetric airborne survey (112 square kilometres). • 121 RC boreholes (8,770 m) at Maga, P8/P9, P11 and Siga. • Ground gradient induced polarization (IP) survey (153.6 km; 100 m line spacing; 25 m stations) at Maga, P8/P9 and CFU. • 1,450 check assays on the 2005 RC samples. • 614 RC composite samples collected for cyanidation metallurgical test work. • 39 core and 17 rock outcrop samples petrographic study.
2007	• Met-Chem Resource Estimate based on RC and core borehole data up to March 2007 – Initiated in August 2007, delivered in February 2008. • Systematic mapping, prospecting, sampling, and gold assaying of outcrops and gold workings. • 57 core boreholes (5,314 m November 2007 to February 2008) mostly within the 2007 Mineral Resource model core samples assayed for gold and also used for structural measurements, multi-element inductively coupled plasma (ICP) orientation study, petrographic study, and petrophysical analyses.
2008	• Systematic mapping, prospecting, sampling, and gold assaying of outcrops and gold workings. • 268 RC boreholes (19,963 m February to April 2008). • Quality assurance/quality control (QA/QC) report, 2007-2008 RC-core programmes. • Cyanidation test work under the supervision of H.C. Osborne and Associates (Commerce City, CO, USA) completed in September 2008. • Petrographic studies by Dr. Schandl (Toronto, ON, Canada) completed between April and September 2008. • 662 multi-element ICP analyses from core samples. • Re-logging of all RC and core boreholes to reconcile the surface mapping, petrography, and ICP data. • SRK Mineral Resource Estimate based on RC and drill core data up to August 2008 – Initiated in June 2008 and delivered in November 2008. • Compilation of all historical RC borehole detailed journals to create a penetration rate model. • Academic study of the petrography and structure of the Bomboré 1 Deposits by H. Zongo, under the supervision of Dr. Lompo from the Université de Ouagadougou – initiated in May 2008. • Check sampling of RC boreholes with poor QA/QC scores (3,211 samples).

Period	Exploration Activities and Studies
2009	• Re-logging of selected RC and core boreholes, revised geological model. • High-resolution (50 m x 10 m) resistivity surveys (237 km) over the Maga, P8/P9, P11, and Siga areas in March-April 2009. • Core drilling programme (April to June 2009), including 20 boreholes (4,502 m) to a vertical depth of 175 m in the P8/P9 area on two fences 200 m apart and three PQ boreholes (235 m) for metallurgical sampling. • Commissioning of bench-top rotary sample dividers for all pulverized samples. • Validation of all historical core specific gravity determinations, including new and more closely spaced determinations. • QA/QC reports on the April-June 2009 core drilling programme and on various check assay programmes. • Metallurgical test work (June to November 2009) by AMMTEC (Perth, WA, Australia) under the supervision of GBM (Twickenham, Middlesex, UK), including bottle tests on coarse material and milled material, flotation/leaching tests on milled material, gravity concentration tests, and column leaching, AMD, UCS, Bond impact, Bond abrasion, Bond rod mill, Bond ball mill and JK Drop-weight tests on two sets of composite samples representative of the oxide, transitional and fresh Bomboré mineralized material – final AMMTEC report January 2010. • Petrographic and structural study of the Bomboré Gold Deposits; Ph.D. progress report. • Environmental Baseline Study (April to July 2009) by Bureau d'Études des Géosciences, des Énergies et de l'Environnement (BEGE) – report July 2009. • Preliminary Environmental Impact Study by BEGE (April to September 2009) – Report September 2009. • Bench-top XRF programme commenced in April 2009. All available historical core pulp samples (> 10,000) were analyzed for a suite of 35 elements. • Core drilling programme (November-December 2009) with five boreholes (3,001 m) drilled to a vertical depth of 300 m in the Maga, P8/P9, and Siga areas. • Several programmes of check assays on RC (n = 130), core (n = 388) and leach tail (n = 914) samples.
2010	• Migration of all historical RC and core data to Datashed database. • QA/QC report on the November-December 2009 core drilling programme. • Mogtédo camp expansion and commissioning of laboratory with two rotary sample dividers (RSDs) for RC samples. • Commissioning of a dedicated laboratory equipped with three bench top RSDs at the Kossodo office. • Check assay programme on SGS Tarkwa and Abilab Ouagadougou RC samples (n = 354) with a soluble gold grade > 1 g/t. • 389-borehole auger drilling programme to investigate the overburden-saprolite interface and saprolite over the Siga South, P11-P8/P9 gap, and other targets (3,055 m). • 617 borehole RC definition and Mineral Resource expansion drilling programme (42,456 m).

Period	Exploration Activities and Studies
	• Establishment of four new pairs of trigonometric beacons at Kiin Tanga, Maga, Siga, and Siga South followed by a check survey of all historical survey control stations and corrections of all historical collar positions. • Several programmes of check assays on RC (n = 2,236), core (n = 141), and leach tail (n = 12,411) samples. • QA/QC report on the 2010 RC drilling programme and various check assay programmes. • Bench-top XRF analysis of approximately 40,000 samples. • Geological 3D model updated in Gemcom. • SRK Mineral Resource Estimate based on RC and core data up to April 2010. Initiated in July 2010 and delivered in October 2010. • 2,374 line-km high-resolution magnetometry and radiometry airborne survey over the Bomboré 1 permit by UTS-Aeroquest in November 2010. • Initiation of a major core drilling definition programme in November 2010 to better define the 2010 sulphide Mineral Resources on a 50 m by 50 m drilling pattern to a vertical depth of about 125 m. The programme was completed in June 2012 and totalled 770 boreholes for 116,795.5 m. • Initiation of the construction of a new base camp for the Project in November 2010. • High-resolution (50 m by 10 m) resistivity surveys (243 km) over the KT, Maga, CFU, P11, Siga South, P16 and P17 areas in December 2010 and January 2011. • Initiation of a major RC drilling programme in February 2011 to better define the 2010 oxide Mineral Resources on a 50 m by 25 m drilling pattern and to test several new targets. The programme was completed in June 2012 and totalled 2,375 boreholes and 135,167 m.
2011	• 2,547 borehole auger drilling programme between February 2011 and July 2011 to investigate the overburden-saprolite interface and the saprolite over several new targets (12,146 m). • Report from AccuMin Minerals Services on the lithostructural controls of the Bomboré gold mineralization. • High-resolution photogrammetric base map of the Bomboré permit generated by Photosat in May 2011. • PEA by GMSI delivered in June 2011. • Initiation of a detailed baseline environmental and impact study based on the PEA/Carbon-in-Leach (CIL) project. The socio-economic study by Société de Conseil et de Realisation pour la Gestion de l'Environnement (SOCREGE) commenced in May 2011 and the EIA by BEGE commenced in September. • Toéyoko permit granted to Orezone in July 2011. • Commissioning of a weather station at the new Bomboré camp in September 2011. • Initiation of a detailed CIL process metallurgical study in September 2011, using McClelland Laboratories, Inc. (McClelland), under the supervision of Woods Process Services. A suite of 76 samples representative of the various oxide, transition, and sulphide facies of each of the Deposits sent to McClelland. Final report delivered in February 2013.

Period	Exploration Activities and Studies
	• 1,901 line-km high-resolution magnetometry and radiometry airborne survey over the Toéyoko permit by UTS-Aeroquest in October 2011. • Delivery by SOCREGE of the report on the socio-economic study relevant to the 2011 PEA/CIL project. • High-resolution photogrammetric base map of the Toéyoko permit generated by Photosat in December 2011.
2012	• Initiation of an environmental testing study in February 2012 on a series of 28 composite waste samples by McClelland. Results received in July 2012. • 598 borehole auger drilling program during the April-May 2012 period to investigate the overburden-saprolite interface and saprolite over several new targets on the Bomboré 1 permit (2,299 m). • 587 borehole auger drilling program during the April-May 2012 period to investigate the overburden-saprolite interface and saprolite over several new targets on the Toéyoko permit (2,561 m). • Petrographic and mesoscopic catalogue of photographs of the Bomboré lithologies; Ph.D. progress report. • Report from Economic Geology Consulting on the mineralogy of the master composite samples used by McClelland for the detailed CIL process metallurgical study. • Delivery by BEGE in July 2012 of the report on the environmental baseline study relevant to the 2011 PEA/CIL project. • High-resolution (50 m 10 m) resistivity surveys (41 km) over the P17N area on Bomboré 1 permit in July 2012. • High-resolution (50 m 10 m) resistivity surveys (51 km) over the P17S area on Toéyoko 1 permit in July 2012. • Prospecting, outcrop sampling, and geological mapping on new regional targets on Bomboré 1 permit from March to July 2012 (401 samples). • Prospecting, outcrop sampling, and geological mapping on new regional targets on Toéyoko 1 permit from March to June 2012 (190 samples). • SRK Mineral Resource Estimate based on RC and core data up to March 2012 – Initiated in March 2012 and delivered in August 2012. • Initiation of a CIL process FS in June 2012, under the direction of GMSI from Brossard, QC, Canada. • Pit slope geotechnical study by Golder Associates Ltd. (Golder) from Montreal, Canada, initiated in August 2012. The final report was delivered by Golder in April 2013 • Core drilling definition programme started in September 2012 to define the 2012 Inferred sulphide Mineral Resources on a 50 m 50 m drilling pattern to a vertical depth of approximately 150 m. The first phase of the programme was completed in February 2013 and totalled 121 boreholes for 23,109.5 m. • QA/QC report, RC, core, and auger programmes completed from November 2010 to June 2012. • QA/QC report, RC, core, and check assay programmes completed from June 2012 to September 2012.

Period	Exploration Activities and Studies
	• RC drilling programme started in September 2012 to define the 2012 Inferred oxide Mineral Resources on a 50 m 25 m drilling pattern and to test several new targets. The first phase of the programme was completed in April 2013 and totalled 541 boreholes and 32,440 m. • Initiation in October 2012 by Golder from Montreal, Canada, of a geochemical characterization study of waste rock, tailings and potential construction material at the Project. Final report received in December 2013. • Initiation in November 2012 of site investigation by Golder from Montreal, Canada, for a feasibility level geotechnical study of the tailings and water management structures for the Project. Preliminary report delivered by Golder in April 2013. • Initiation in November 2012 of site investigation by Golder from Montreal, Canada, for a feasibility level geotechnical study of the design of foundations at the processing plant site and at the Nobsin and Bomboré bridges for the Project. Preliminary technical memorandum delivered by Golder in May 2013. • Completion of the preliminary scrubbing test work completed by Orezone under the direction of GMSI, targeting the saprolite gold resources. • QA/QC report, RC, core and check assay programmes completed from October 2012 to December 2012. • Initiation in December 2012 of a complementary comminution study with Hazen Research Inc., from Golden, CO, USA, on three granodiorite samples from the weathered zone. Report delivered in February 2013. • Initiation in December 2012 of a complementary comminution study with SGS Canada Inc from Lakefield, ON, Canada, on twenty sulphide samples. Report delivered in May 2013.
2013	• Initiation in January 2013 of an eight-borehole, 280 m saprolite PQ core drilling programme for a scrubber test work programme with Met-Solve Laboratories Inc. (Met-Solve) from Langley, BC, Canada. Report delivered in May 2013. • Delivery by SOCREGE of an interim socio-economic study relevant to the 2011 CIL Definitive Feasibility Study (DFS) project. • Final report of the study of the archaeological artifacts collected by BEGE. • Final report from McClelland on the CIL DFS process metallurgical study. • Initiation in March 2013 of a metallurgical study on pyrrhotitic samples with COREM from Quebec City, QC, Canada. Report delivered in May 2013. • SRK Mineral Resource Estimate updated based on RC and core borehole data up to November 2012 for the North and South models and March 2013 for the Southeast model. • Decision in June 2013 to interrupt the CIL process DFS due to adverse economic conditions. Initiation of a review of the 2011 Heap Leach (HL) PEA. • Interim report from BEGE on complementary botanical and archaeological studies completed for the CIL DFS. • Decision in August 2014 to update the 2011 HL PEA, under the supervision of GMSI, and with the support of Kappes, Cassiday & Associates (KCA) from Reno, NV, USA, and Golder Associates Inc. (Golder) from Reno, NV, USA for the process engineering. • QA/QC report, RC, core, and check assay programmes completed from November 2012 to June 2013.

Period	Exploration Activities and Studies
	• Preliminary PEA HL facility design delivered by Golder in November 2014. • Final DFS report from Golder in December 2013 on the CIL process geochemical characterization of waste rock, tailings, and potential construction material.
2014	• Release in January 2014 of the findings of the HL PEA completed under the direction of GMSI. • Decision to proceed with an HL DFS in January 2014. • Initiation in January 2014 of the HL metallurgical DFS under the supervision of KCA. This study includes one sample consisting of the coarse fraction scrubbed from the Met-Solve Laboratories Inc. 2013 oxide core samples scrubbing programme. Final report delivered in August 2014. • Preliminary design delivered by Golder in February 2014 of the HL facility at a new site retained for the HL DFS. • Initiation of the HL DFS geotechnical study under the supervision of Golder. Field report delivered in July 2014. The field programme executed in February and March 2014 included seven new core boreholes (170 m), eight new pressure metre boreholes (167 m), 51 new RC boreholes (2,167 m; a piezometer was installed in 8 boreholes), and 71 test pits (up to 5 m deep). All the samples were described. Laboratory test work was completed and reported from April to July 2014. All the samples were, if possible, used by Orezone as part of the sterilization programme, i.e., they were described, assayed for gold, and analyzed by XRF (multi-elements Orezone bench top Niton units). • Updated CIL process interim baseline environmental study from BEGE delivered in March 2014. • Report delivered in April 2014 of the audit completed by WSP Canada Inc. (WSP) on the ESIA and RAP work completed by BEGE and SOCREGE since 2011, and on the gaps to fill to complete the HL DFS. • Cyanide leach report delivered in April 2014 by Met-Solve on the fine fraction scrubbed from the 2013 oxide core samples. • Limited core drilling programme (1,114 m) in May 2014 in the CFU and P17S areas. • RC drilling programme (21,383 m) from May to July 2014, essentially infill definition drilling in the north area of the project. • Hiring KCA to coordinate and deliver the hybrid process DFS. • Complementary DFS comminution test work report delivered by SGS Lakefield Canada in June 2014. • Updated CIL process interim baseline environmental study from BEGE delivered in June 2014. • Updated CIL process interim socio-economic study from SOCREGE delivered in July 2014. • Decision to assess a hybrid process (HL and CIL) for the DFS based on the HL metallurgical study results to reduce the operational risks inherent to the high-cement agglomeration requirement for the saprolite material. • Release in June 2014 of the preliminary conclusions about the hybrid process based on the Met-Solve and KCA test work. Final report on the hybrid process preliminary test work delivered by KCA in November 2014.

Period	Exploration Activities and Studies
	• Hybrid design trade-off study delivered by Golder in June 2014. • Revised ESIA and RAP terms of reference relevant to the hybrid process submitted to the Ministry of Environment in July 2014. • PFS assessment of the hybrid process facility (tailings storage and heap leach pad) delivered by Golder in August 2014. • Preliminary assessment by WSP in August 2014 of the 2009 and 2014 Bomboré metallurgical results relevant to the environmental impacts of the hybrid process DFS. • Preliminary assessment by WSP in September 2014 of the baseline air quality conditions relevant to the environmental impacts of the hybrid process DFS. • Preliminary assessment by WSP in September 2014 of the baseline acoustic conditions relevant to the environmental impacts of the hybrid process DFS. • QA/QC report, RC, core, geotechnical and check assay programmes completed from July 2013 to August 2014. • DFS pit slope recommendations from Golder delivered in November 2014. • Initiation of the hybrid process DFS geotechnical study under the supervision of Golder. The field programme was executed in December 2014 and included 40 new RC holes (1,153 m; a piezometer was installed in one borehole) and 58 test pits (up to 5 m deep). All the samples were described. Laboratory test work was completed and reported from January 2015 to March 2015. All of the samples were, if possible, used by Orezone as part of the sterilization program, i.e. they were described, assayed for gold, and analyzed by XRF (multi-elements Orezone bench-top Niton units).
2015	• DFS Assessment, Hybrid Facility (Tailings Impoundment and Heap Leach Pad), final report delivered by Golder in January 2015. • Progress report delivered by BEGE in January 2015 on the complementary archaeological and ethnographic studies relevant to the hybrid process DFS. • QA/QC report, geotechnical and check assay programmes completed from September 2014 to February 2015. • Various reports delivered by Golder, such as the design of the waste rock dumps, plant foundations, bridge foundations, and surface water management infrastructure. • Final revision of the ESIA and RAP terms of reference relevant to the hybrid process submitted to the Ministry of Environment in February 2015.
2016	• A small drill programme completed from November to December 2016, including 3,162 m of RC drilling in the P13 and P17S areas, and 2,806 m of core definition drilling in the P17S area. • A small induced polarization and gravimetry test survey was also completed during the drill programme in the P17S area.
2017	• A metallurgical test work programme on one P17S granodiorite composite sample was initiated in July 2016 and completed in January 2017. The test work included head analysis, QEM-RMS mineralogy, Bond ball mill grindability, gravity separation, whole material, and gravity tailing cyanidation, flotation and concentrate cyanidation testwork. • Induced polarization surveys (various configurations) totalling 37, 425 m of profiles were completed by Sagax Afrique on the Toéyoko and Bomboré IV permits in the P17 and P17S Deposits area.

Period	Exploration Activities and Studies
	• Auger drilling programmes totalling 462 boreholes and 1,648 m were completed in January and February 2017 in the MV3, BV1 and BV2 areas before the mandatory 25% surface reduction of the Toéyoko permit. • A small reverse circulation drilling programme of 27 boreholes totalling 985 m was completed in March 2017 in the P17S area on Toéyoko permit to complete the definition of the shallow mineralization and follow up on some auger anomalies. • A core drilling programme of 52 boreholes totalling 7,457 m was completed from February to June 2017 in the P17S area on Toéyoko permit to advance the definition of the P17S Deposit. • An auger drilling programme totalling 144 boreholes and 528 m and a small reverse circulation drilling programme of 17 boreholes totalling 857 m were completed in February and March 2017 on the new Bomboré II permit to follow up on auger anomalies in the vicinity of planned resettlement sites, which were sterilized. • A small auger drilling programme totalling 76 boreholes and 339 m was completed in February 2017 on some weak geochemical anomalies within the limits of the new Bomboré III permit. • A core drilling programme of 13 boreholes totalling 3,275 m was completed from April to June 2017 in the P17S northeast extension on the new Bomboré IV permit to advance the definition of the gap between the P17 and P17S Deposits. • Petrographic descriptions from a series of 14 granodiorite core samples were completed in July 2017. • A reverse circulation drilling programme of 249 boreholes totalling 13,911 m was completed from July to October 2017 in the Siga South, Siga East, P11 and CFU oxide areas on the mining lease to tighten-up the drill spacing to 25 m by 25 m and better define discrete better-than-average gold zones present in these areas.
2018	• A small drill programme completed from January to March 2018 on the Bomboré II permit, including 1,009 m of RC drilling and 238 m of core drilling in the Kiin Tanga area to follow-up on past RAB and RC scout anomalies. • A drill programme completed from January to March 2018 on the Bomboré III permit, including 387 m of RC drilling and 1,348 m of core drilling, with much of the core drilling to further advance the definition of the gap between the P17 and P17S Deposits. • A small drill programme completed in December 2018 on the Bomboré IV permit, consisting of 219 m of RC drilling on across the Nobsin River to define the thickness of the alluvial deposits and the geology of the underlying Birimian units. • A drill programme completed in March 2018 on the Toéyoko permit, including 1,447 m of RC drilling and 510 m of core drilling on the P13 prospect. • A small geotechnical drill programme completed in June 2018 and consisting of 56 m of RC drilling in the Maga area on the Orezone Bomboré SA permit. • A geotechnical programme completed in November and December 2018, consisting of 8 test pits, 4 core holes (105 m) and 16 piezometer holes, including 13 on the Orezone Bomboré SA permit and 3 on the Toéyoko permit around the P17S Deposit.
2019	• A metallurgical study initiated at the end of 2018 was completed at Base Met Labs, BC, Canada in 2019 on a suite of 4 composite and 11 variability fresh (sulphide) samples from

Period	Exploration Activities and Studies
	various deposits included in the Bomboré Mineral Reserves, including Maga, P8P9, Siga and P17S. The study included comminution, leaching, pre-oxidation and thickening test work. • A metallurgical study was completed in 2019 at SGS Lakefield, ON, Canada, on a Bomboré oxide sample, in order to examine and optimize the carbon-in-leach (CIL) circuit design. • A drill programme of 1,311 m at a drill spacing of 12.5 by 12.5 m to test the Mineral Resource model at the grade control drill spacing contemplated in the 2018 feasibility study over three mining panels along the fringe of the Maga Main, Maga Hill and Siga East Deposits on the Orezone Bomboré SA permit. • A drill programme completed in September and October 2019 on the Maga Hill Deposit, Orezone Bomboré SA permit, including 608 m of RC drilling and 1,374 m of core drilling. • A petrographic study was initiated at the University of Fada, Burkina Faso, on a suite of 18 samples from the Bomboré III permit Kiin Tanga Deposit, 12 samples from the Bomboré III P17N Deposit, 20 samples from the Bomboré V P13 prospect, and 26 samples from the Bomboré V P17S Deposit. These samples were also submitted to the University of Nancy, France, for a lithogeochemistry study of major and trace elements, including rare earths.
2020	• A geotechnical programme consisting of 30 test pits was completed in March 2020 over the Orezone Bomboré SA permit TSF site. • A metallurgical study of the silver head grade and silver recovery was performed on the 2019 Base Met Labs samples at their laboratory. • A RC grade control programme of 19,759 m completed from October to December 2020 over the Orezone Bomboré SA permit OCR site, using the exploration logging, sampling and assaying protocols. • A resistivity survey totalling 16,000 m of profiles was completed by Sagax Afrique in December 2020 on the Bomboré II permit north of the Kiin Tanga Deposit. • A resistivity survey totalling 40,000 m of profiles was completed by Sagax Afrique in December 2020 on the Bomboré III and Bomboré V permits over the P13 prospect.
2021	• A resistivity survey totalling 10,000 m of profiles was completed by Sagax Afrique in January 2021 on the Bomboré IV permit over a potential bridge site. • A core drilling programme of 5,136 m on the Orezone Bomboré SA and Bomboré III permits to advance the definition of the P17S Deposit. • A small core drilling programme of 70 m completed in November and December 2021 on the Bomboré II permit. • A small geotechnical programme consisting of two test pits and one core hole of 20 m over the Bomboré III permit potential TSF expansion site. • A geotechnical programme consisting of ten test pits and two core holes totalling 29 m over the Orezone Bomboré SA TSF site. • A small core drilling programme of 180 m on the Orezone Bomboré SA P17S Deposit.
2022	• A resistivity survey totalling 41,700 m of profiles was completed by Sagax Afrique in January 2022 on the Bomboré V permit over the P13 prospect. • A core drilling programme of 9,065 m to advance the definition of the P17S and P17 Deposits on the Bomboré II permit from February to May 2022.

Period	Exploration Activities and Studies
	• A programme consisting of 37,201 m of core drilling and 21,660 m of RC drilling to convert Inferred Mineral Resources on the Orezone Bomboré SA permit. • High-resolution topographic drone surveys were completed by Orezone Bomboré SA over the P17, P17S, Siga and P11 Deposits. • A metallurgical study of Bomboré sulphide material generated by the 2022 core drilling programme was initiated and samples from the P17, P17S and Siga Deposits were sent to Maelgwyn in South Africa. The programme consists of variability testing test, GRG testing, oxidation, and leach optimization test work.
2023	• Metallurgical samples from the Maga Deposit were sent to Maelgwyn, South Africa to complete the programme initiated in 2022. • An RC drilling programme of 11,677 m to achieve a 25 m drill spacing within 2023 oxide mining panels on the Orezone Bomboré SA permit.

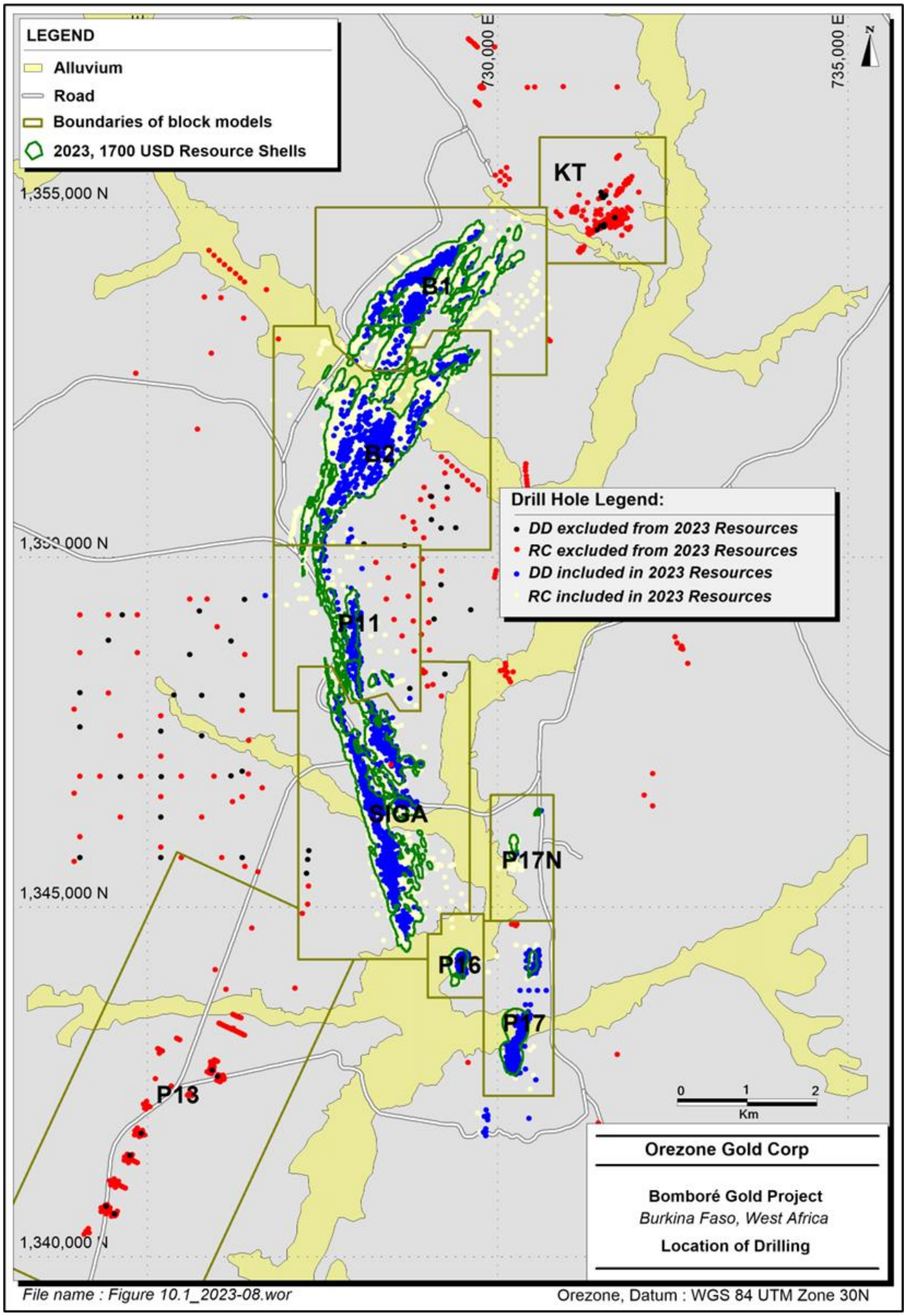
10.0 DRILLING

10.1 Drilling Programs

Orezone completed core, reverse circulation (RC), and auger drilling programs on the Bomboré Property from 2003 to April 2023 that supported the geological model used for the Mineral Resource Estimate presented in Section 14. Historical auger, trench, and Rotary Air Blast (RAB) sample assay results were not utilized for the current Mineral Resource estimation, but these drill data were utilized to interpret the geological model.

The location of the core and RC borehole collars is shown in Figure 10.1.1. A summary of drilling to April 2023 is presented in Table 10.1.1. Drilling by previous Property owners is summarized in Section 6 of this Report.

Figure 10.1.1 Orezone Drilling Locations



Source: Orezone (September 2023)

Table 10.1.1 Summary of Bomboré Project Drilling to 30 April 2023

Company	Years	Drilling Type	Number of Holes Drilled	Length m
Channel	1994 to 2000	RC	261	19,501
Channel	1997 to 1998	RAB	1,000	34,249
Channel	1998	Core	10	1,080
Orezone	2003 to 2006	RC	351	23,017
Orezone	2007 to 2008	RC	287	21,246
Orezone	2007 to 2008	Core	57	5,714
Orezone	2009	Core	29	7,738
Orezone	2010 to 2014	Core	939	142,616
Orezone	2011 to 2014	Auger	4,221	20,057
Orezone	2011 to 2014	RC	4,073	240,291
Orezone	2016 to 2018	RC	664	34,735
Orezone	2016 to 2018	Core	166	22,566
Orezone	2017	Auger	700	2,558
Orezone	2019	RC	65	1,919
Orezone	2019	DD	10	1,374
Orezone	2020	RC -GC	527	19,759
Orezone	2021	RC	16	931
Orezone	2021	DD	37	5,435
Orezone	2022	RC	377	21,660
Orezone	2022	DD	236	46,265
Orezone	2023	RC	178	11,877
Sub-total (Included in 31 March 2023 Mineral Resources)		Auger	2,244	11,368
		RAB	846	28,195
		RC	6,332	373,001
		Core	1,426	228,944
Sub-total (excluded from 30 March 2023 Mineral Resources)		Auger	2,677	11,248
		RAB	154	6,054
		RC	467	21,935
		Core	69	4,080
Totals as of 30 April 2023		Auger	4,921	22,616
		RAB	1,000	34,249
		RC	6,799	394,936
		Core	1,495	233,023

DD = diamond borehole; RC = reverse circulation borehole

A total of 536 RC (467) and diamond core (69) drill holes were excluded from the Mineral Resources estimation, because these data were located outside the Mineral Resources area.

10.2 Drilling Procedures

10.2.1 Type of Drilling

Orezone chose to drill significantly more RC holes than core holes because of the shallow and weathered nature of the targeted portion of the BSZ. Core drilling was used to define deeper targets within the sulphide zone.

Since 2010, core drilling has been completed by JMS Drilling Inc. (JMS) utilizing up to five Boart Longyear 44 rigs with an HQ core barrel for the weathered zone and an NQ core barrel for the fresh bedrock. In 2021 and 2022, Forajo Sarl also completed 21,229 m of core drilling at Bomboré.

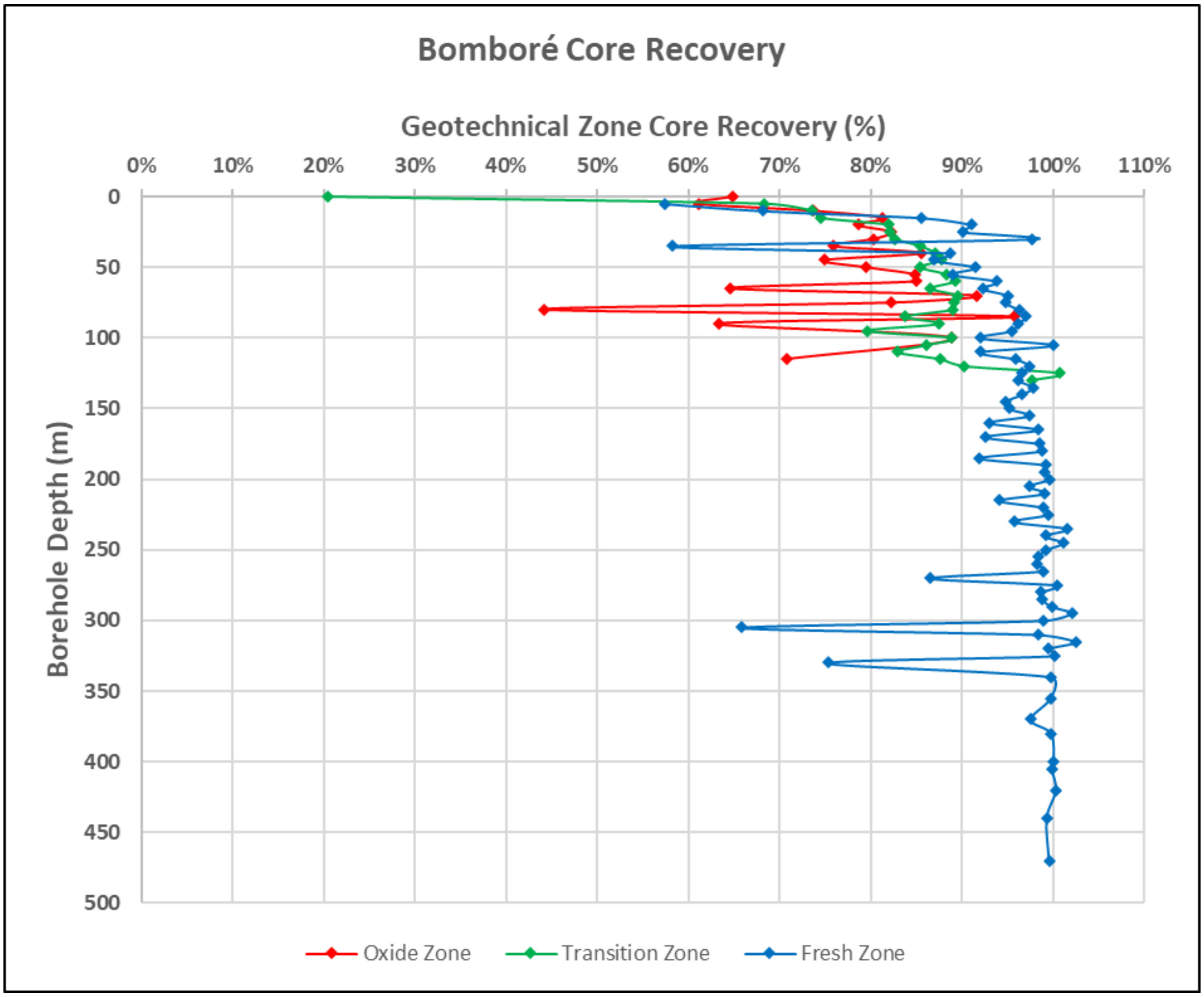
Prior to 2012, RC drilling was mostly completed by Boart Longyear utilizing either a CatMax or a DeltaBase RC drilling rig equipped with a 5.25 in. hammer bit. Since February 2012 to March 2021, all RC drilling has been completed utilizing Orezone's own Hardab rig operated by JMS until December 2014 and subsequently by Orezone. In 2022 and 2023, RC drilling was completed by JMS UDR650 rigs.

10.2.2 Water Table Elevation and Sample Recovery

The water table is encountered at an average depth of 20 m and is shallower in the southern (Siga Deposits or Bomboré South) area and deeper in the northern (Maga Deposits or Bomboré North) area. The RC rigs are equipped with compressors powerful enough to completely flush the borehole between rod additions and during bit advancement. Recovery from the RC boreholes is based on sample weight and has been estimated to average between 83% and 92% in the oxide zone, 91% and 95% in the transition zone, and 84% and 89% in the sulphide zone. These estimates are based on a theoretical volumetric density of 1.83 g/cm^3, 2.35 g/cm^3, and 2.86 g/cm^3, respectively, for each of those weathering zones.

Core recovery, based on detailed geotechnical logs representing 227,712 m of drilling, averages 75% in the Oxide zone and increases across the weathering profile to an average of 83% in the Transition zone and 95% in the Fresh (sulphide) zone (Figure 10.2.1).

Figure 10.2.1 Core Recovery by Vertical Depth



Source: Orezone (2023)

10.2.3 Borehole Orientation and Drilling Pattern

The RC and core boreholes were drilled towards the northwest in the Bomboré North area (Maga, CFU and P8/P9 Deposits), the west in the area of P11, P16, and P17 Deposits area, and the west-southwest in the Bomboré South area (Siga Deposits area). In all areas, the drilling direction is opposite to the dip and orthogonal to the average strike of the lithological units, major fabrics, and wireframed mineralized domains. The plunge of the boreholes at the collar is generally 50°±5°, thereby intersecting the lithological units, major fabric and wireframed mineralized domains at an angle between 65° and 90°.

The oxide resources have been defined along 50 m-spaced drill sections with 25 m between the drill collars.

The sulphide resources have been defined generally along 50 m-spaced drill sections with 50 m between the drill collars, but structurally more complex zones have been defined along 25 m-spaced sections with 25 m between the drill collars. In some areas, such as the Maga North and P8/P9 starter pits, the RC drill collars were drilled on a 25 m by 25 m pattern.

10.2.4 Planning and Borehole Implementation

Drilling programs are planned by the exploration team, under the supervision of the Senior Vice President of Exploration and the Exploration Manager. A handheld GPS with a precision of ±5 m is utilized by a technician to locate and prepare drilling pads. The borehole collars are spotted in the field and pegged using a Differential Global Positioning System (DGPS) set to achieve sub-metre accuracy.

When drilled, the casing is surveyed first using a DGPS, and then a Total Station instrument coupled with DGPS accessing the national network of CORS DGPS stations now installed in Burkina Faso. This system provides accurate coordinates over the entire Project area. The DGPS accuracy is validated on a known control station at the beginning and end of each work shift.

On completion, a three metre PVC pipe is inserted in RC holes and a six metre PVC pipe is inserted in core holes. The top of the pipe is capped by a concrete beacon on which the Hole-ID, the final depth, and the date of completion are recorded.

10.2.5 Borehole Trajectory

Orezone conducts downhole deviation surveys in open RC boreholes after drilling is completed, rods have been pulled and the rig moved, but before the PVC casing is capped. Readings are taken at 25 m increments starting at six metres below the collar. The reading at a depth of six metres is used to control the quality of the drill collar alignment.

Readings in core boreholes are taken once or twice a day with the instrument positioned six metres ahead of the drill string to avoid magnetic interference. If the distance between successive tests exceeds 30 m, rods are removed to take additional readings and to maintain on average 25 m between successive readings.

The path of the Orezone boreholes was surveyed using a Reflex Instrument that measures several parameters, including the plunge of the borehole and the three components of the magnetic field. It relies on a compass to read the azimuth. The azimuth angles are validated against the measured intensity of magnetic field, and an accelerometer reading to ensure the compass was stable when measurements were taken. The magnetic azimuth is converted to a geographic azimuth using the declination applicable at the time of the survey.

The path of RC boreholes on the Bomboré Property typically steepens up with depth, contrary to core holes that have the opposite behaviour. However, both types of boreholes deviate to the right of the collar azimuth. Borehole deviation is not a critical issue, because more than 75% of the boreholes are shorter than 70 m and 93% of the boreholes do not exceed a depth of 90 m. The Reflex Instrument occasionally produces incorrect results, which can be filtered out and the deviation path interpolated.

In the opinion of the Author of this Report section, the borehole survey method applied by Orezone conforms to industry best practice.

10.2.6 Description of RC Cuttings

RC holes are sampled at 1 m intervals by collecting 100% of the material reporting to the cyclone. Small samples of screened and washed chips from each 1 m run are saved in labelled plastic boxes (chip boxes). A log of RC chips is made at the drill site to monitor the drill advance and extend the borehole if required to penetrate the top of the sulphide zone. A large majority of RC holes reach a minimal depth of 50 m, even though the top of the sulphide zone can be significantly shallower.

10.2.7 QP Comments

In the opinion of the Author of this Report section, the drilling procedures employed by Orezone conform to industry best practice and the resultant drilling patterns are sufficient to interpret the geometry and the boundaries of the gold mineralization with confidence. Qualified personnel under the direct supervision of appropriately qualified geologists conducted all drilling sampling. The Author is not aware of any drilling, sampling, or recovery factors that could materially affect the Mineral Resource Estimate.

10.3 Drilling Results

Numerous drill programs have been completed on the Bomboré Property since the early 2000s (see Tables 9.3.1 and Table 10.1.1). Drilling assay results from 2016 to 2023 for many of the Bomboré gold deposits, including oxide and sulphide zones, are presented below. The assay results are compiled from Orezone press releases available at www.orezone.com and filed under Orezone's profile on SEDAR+ at www.sedarplus.ca, and from Orezone's drill hole database.

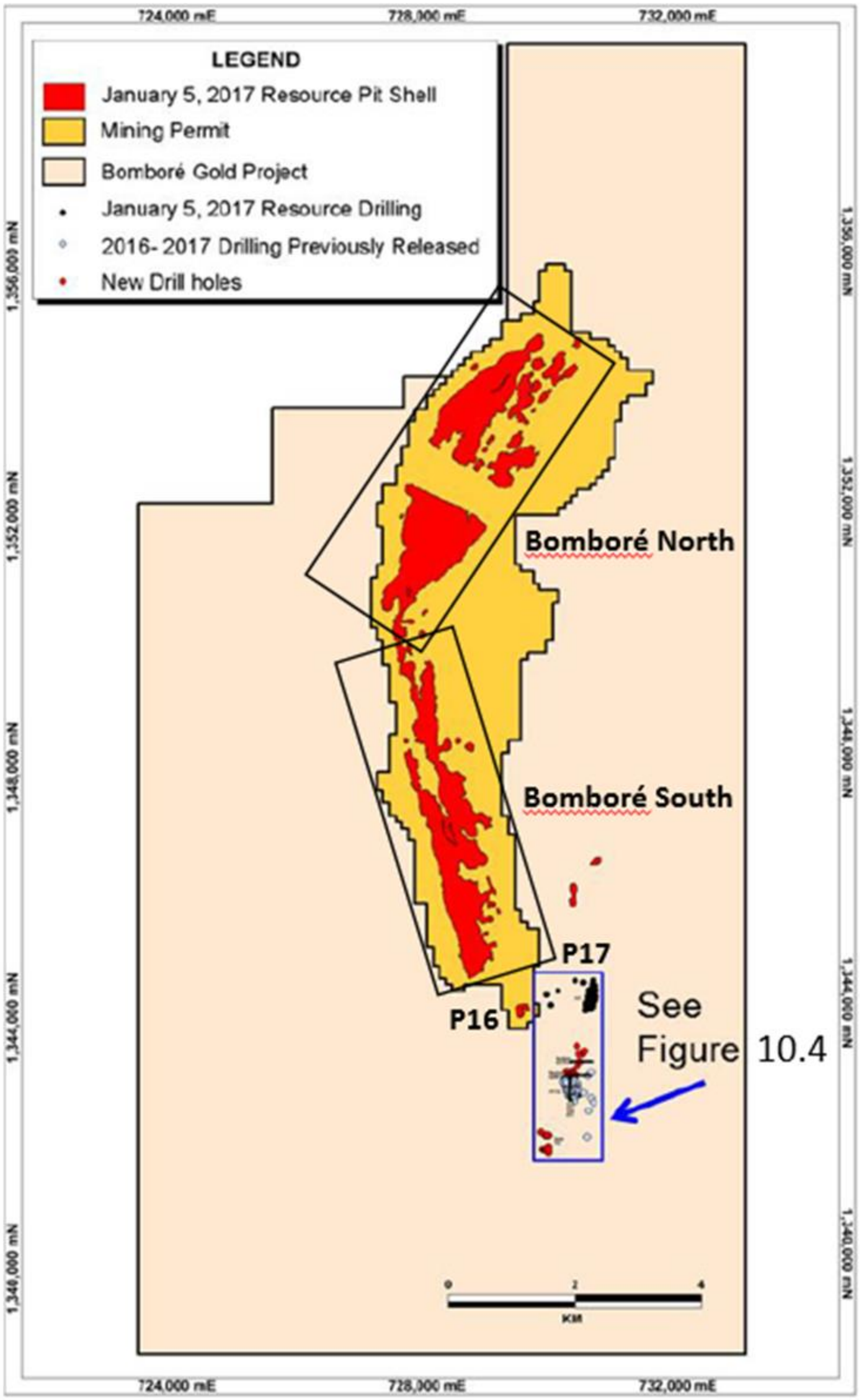
10.3.1 2016 Drilling Program

In November 2016, Orezone commenced an exploration and definition drilling program on the Bomboré Property that was designed to: 1) upgrade existing oxide Mineral Resources and test gaps between oxide pit shell models; 2) simulated grade control drilling for modelling purposes in representative areas of the oxide Mineral Resource model; and 3) further test regional oxide targets and higher-grade, near-surface, sulphide targets. The 2,000 m program was completed on the P17S Deposit and on resistive lineaments located to the east of the P17S Deposit. In total, 62 holes were drilled, and the P17S Deposit was extended from 60 m depth to 100 m depth below surface. For the assay results, see Orezone press release dated 17 January 2017 and available on Orezone's website.

10.3.2 2017 Drilling Program

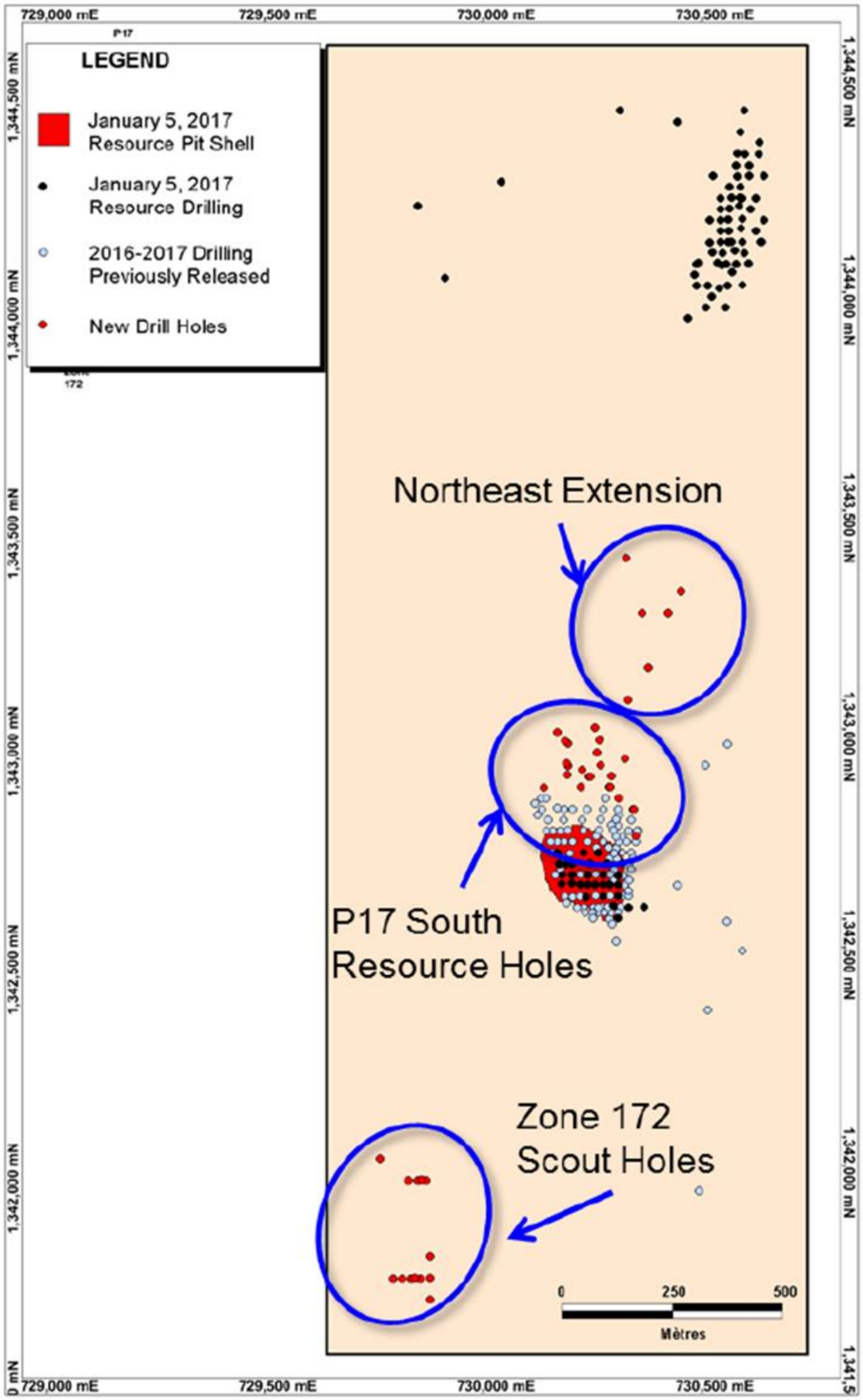
Following review of the 2016 results, drilling commenced in February 2017 and 40 RC and core holes were completed in the P17S Deposit area (Figures 10.3.1 and 10.3.2). The farthest step-out drill holes intersected mineralized granodiorite to a vertical depth of 150 m and indicate down-plunge continuity of the zone to >350 m. The P17S Deposit remained open at depth and to the north.

Figure 10.3.1 2017 Drill Plans



Source: Orezone press release dated 7 June 2017, modified by P&E (April 2022)

Figure 10.3.2 2017 Detailed Drill Plan



Source: Orezone press release dated June 7, 2017

Table 10.3.1 Highlights of the P17S 2017 Drill Results

Section	Borehole	Drill Hole Type[1]	From m	To m	Length m	Au g/t	Depth m
42875	TYD0061	DD	116.00	118.30	2.30	5.04	110
		DD	126.00	138.25	12.25	3.61	125
42850	TYD0059	DD	148.00	157.00	9.00	2.61	115
		DD	159.00	163.40	4.40	3.76	121
42825	TYD0057	DD	130.00	133.75	3.75	3.39	101
42787.5	TYD0071	DD	80.50	94.40	13.90	1.76	78
42775	TYD0055	DD	51.00	64.00	13.00	2.38	44
	TYD0062	DD	111.00	115.25	4.25	3.02	84
42750	TYD0054	DD	25.00	32.50	7.50	1.77	22
		DD	80.35	85.40	5.05	4.33	63
	TYD0063	DD	97.30	103.90	6.60	2.81	76
42737.5	TYD0161	RC	28.00	33.00	5.00	2.30	24
	TYD0160	RC	19.00	32.00	13.00	1.77	20
	TYD0069	DD	77.70	81.65	3.95	7.13	62
		DD	99.00	104.00	5.00	2.37	78
42712.5	TYC0163	RC	3.00	24.00	21.00	2.01	11
	TYCO162	RC	21.00	25.00	4.00	2.84	18
	TYD0068	DD	81.30	92.20	10.90	1.95	66
42687.5	TYD0067	DD	71.20	77.00	5.80	3.07	56
42675	TYD0051	DD	57.00	63.00	6.00	2.29	47
42662.5	TYD0066	DD	52.90	60.25	7.35	2.30	43

Source: Orezone press release (April 12, 2017)

[1] DD = diamond core borehole; RC = reverse circulation borehole

By June 2017, drilling at P17S Deposit had intersected mineralized granodiorite to a vertical depth of 150 m with a down-plunge continuity >425 m. Selected highlights of the 2017 drilling program at P17S follow below:

- Borehole TYD0073: 4.80 m @ 2.29 g/t Au and 6.30 m of 4.47 g/t Au.
- Borehole TYD0082: 13.30 m @ 2.06 g/t Au.
- Borehole TYD0083: 9.70 m @ 3.38 g/t Au.
- Borehole TYD0084: 2.00 m @ 14.10 g/t Au.

- Borehole TYD0087: 6.90 m @ 4.85 g/t Au.

In addition, mineralized granodiorite was intersected to the northeast of P17S in a series of reconnaissance drill holes that targeted the gap between the P17S and P17 Deposits, where the prospective sequence was confirmed in the drilling over an additional strike length of >400 m. Selected highlights of this drilling are listed below:

- Borehole BBD0980: 10.00 m @ 3.20 g/t Au.
- Borehole BBD0981: 8.20 m @ 1.89 g/t Au, and 7.80 m @ 1.54 g/t Au.
- Borehole TYD0077: 14.00 m @ 1.65 g/t Au.

Furthermore, at Zone 172, <1 km southwest of P17S (Figure 10.3.2), reconnaissance RC drilling of auger drilled gold anomalies and limited core drilling intersected mineralized granodiorite within a meta-gabbro sequence. All three Deposits (172, P17S and P17) span a corridor of >2.5 km long and approximately 400 m wide, which was interpreted to be very prospective for the higher-grade, granodiorite-hosted gold mineralization.

Following the consistent higher-grade intercepts at the P17S Deposit, Orezone reviewed all historical drilling data.

Previous historical drilling in the main Bomboré North-Bomboré South Deposits trend did return several high-grade intercepts, generally within or in the vicinity of the granodiorite units. Several of the historical intersections are of significant width and grade and they also appear to follow a similar plunge to that of the P17S zones. Furthermore, the high-grade intercepts occur along the entire strike of the main Bomboré North-Bomboré South Deposits trend and are within the oxide zone.

Orezone concluded that there was potential for several high-grade zones and shoots that had not previously been incorporated in the Mineral Resource modelling and which could have a positive impact on grade and tonnes of the Bomboré Mineral Resource model. Prior selected historical drilling highlights for each of the Siga South, Siga East / West, P11 and CFU Deposits are listed below.

Siga South (Bomboré South Area)

- Borehole BBC1388: 5 m @ 8.9 g/t Au from 18 m, incl. 2 m @ 19.6 g/t Au.
- Borehole BBC1393: 14 m @ 130.2 g/t Au from 24 m, incl. 1 m @ 1.78 g/t Au.
- Borehole BBC1333: 4 m @ 20.4 g/t Au from 16 m, incl. 1 m @ 79.7 g/t Au.
- Borehole BBC3262: 8 m @ 5.1 g/t Au from 30 m, incl. 3 m @ 11.1 g/t Au.

- Borehole BBD0744: 2.5 m @ 26.1 g/t Au, incl. 1.5 m @ 42.6 g/t Au.

At that time, true widths had yet to be determined.

Siga East and Siga West (Bomboré South Area)

- Borehole BMC0080: 16 m @ 9.0 g/t Au from 38 m, incl. 4 m @ 29.7 g/t Au.
- Borehole BMC0093: 4 m @ 16.2 g/t Au from 42 m, incl. 2 m @ 29.5 g/t Au.
- Borehole BBC1626: 5 m @ 3.1 g/t Au from 23 m, incl. 1 m @ 10.4 g/t Au.
- Borehole BBD0246: 10 m @ 3.1 g/t Au from 45.5 m, incl. 1 m @ 13.5 g/t Au.
- Borehole BBD0359: 4.5 m @ 23.0 g/t Au from 66.5 m, incl. 1.5 m @ 67.0 g/t Au.
- Borehole BBC0250: 1 m @ 44.5 g/t Au from 57 m.

At that time, true widths had yet to be determined.

P11 Deposit (Bomboré South area)

- Borehole BBC0568: 1 m @ 36.7 g/t Au from 3 m.
- Borehole BBC0568: 3 m @ 30.4 g/t Au from 72 m, incl. 1 m @ 89.9 g/t Au.
- Borehole BBC1061: 4 m @ 8.8 g/t Au from 1 m, incl. 1 m @ 32.2 g/t Au.
- Borehole BBC1050: 1 m @ 38.2 g/t Au from 59 m (hole stopped in mineralization).

At that time, true widths had yet to be determined.

CFU Deposit (Bomboré North area)

- Borehole BBC4365: 5 m @ 20.1 g/t Au from 39 m, incl. 2 m @ 47.9 g/t Au.
- Borehole BBD0921: 3 m @ 59.0 g/t Au from 75.5 m, incl. 1 m @ 175 g/t Au.
- Borehole BBC4223: 5 m @ 21.7 g/t Au from 60 m, incl. 1 m @ 103 g/t Au.
- Borehole BBC0778: Hole stopped in 4 m @ 22.7 g/t Au from 78 m, incl. 1 m @ 81 g/t Au.

At that time, true widths had yet to be determined.

On completion of further modelling, Orezone decided to systematically drill each of the four prioritized targets. The RC drilling program commenced on 8 July 2017 and consisted of in-fill drill holes to tighten up the hole spacing from 50 m by 25 m to 25 m by 25 m in the four target deposit areas. By October 2017, 249 drill holes totalling 13,900 m were completed on the Siga South, Siga East, P11 and CFU areas. The drill holes in the Siga South and Siga East Deposits area had collar azimuths of N250° with a plunge of -50°. All drill holes in the P11 area had collar azimuths of N270° with a plunge of -50°. All drill holes in the CFU Deposit area had collar azimuths of N312° with a plunge of -50°.

RC drilling results were reported first for the Siga East and Siga South Deposits (Bomboré South Area).

The drill holes were designed to better define the extent and the geometry of discrete high-grade mineralization within the oxidized portion of the Mineral Resources. Selected highlights from the Siga East and Siga South Deposits drilling follow below.

- Borehole BBC4505: 5 m @ 3.18 g/t Au from 6 m, incl. 1 m @ 12.7 g/t Au.
- Borehole BBC4506: 15 m @ 7.54 g/t Au from 40 m, incl. 8 m @ 13.2 g/t Au.
- Borehole BBC4523: 2 m @ 7.82 g/t Au from 41 m, incl. 1 m @ 14.7 g/t Au.
- Borehole BBC4547: 11 m @ 2.37 g/t Au from 28 m, incl. 2 m @ 6.2 g/t Au.
- Borehole BBC4564: 16 m @ 2.27 g/t Au from 0 m, incl. 1 m @ 11.0 g/t Au.
- Borehole BBC4573: 26 m @ 2.01 g/t Au from 16 m, incl. 1 m @ 10.1 g/t Au; and 1 m @ 10.1 g/t Au from 48 m.

At that time, true widths had yet to be determined.

Additional selected significant highlights from the 2017 drilling are listed below.

- Borehole BBC4494: 10 m @ 1.39 g/t Au from 7 m.
- Borehole BBC4497: 19 m @ 0.91 g/t Au from 6 m, incl. 1 m @ 6.3 g/t Au.
- Borehole BBC4502: 28 m @ 0.89 g/t Au from 15 m.
- Borehole BBC4557: 6 m @ 1.82 g/t Au from 60 m.
- Borehole BBC4561: 12 m @ 1.21 g/t Au from 24 m, incl. 1 m @ 6.9 g/t Au.
- Borehole BBC4581: 7 m @ 2.14 g/t Au from 24 m, incl. 1 m @ 5.5 g/t Au.

- Borehole BBC4612: 18 m @ 1.18 g/t Au from 1 m.

At that time, true widths had yet to be determined.

Drilling at the P11 and CFU Deposits (Bomboré North area) successfully intersected high-grade mineralization in the targeted areas. Selected highlights from this drilling follow below.

- Borehole BBC4580: 5 m @ 3.24 g/t Au from 30 m, incl. 1 m @ 10.7 g/t Au.
- Borehole BBC4608: 8 m @ 3.54 g/t Au from 23 m, incl. 2 m @ 8.7 g/t Au.
- Borehole BBC4615: 2 m @ 6.43 g/t Au from 20 m, incl. 1 m @ 10.4 g/t Au.
- Borehole BBC4644: 13 m @ 1.43 g/t Au from 31 m.
- Borehole BBC4673: 11 m @ 2.22 g/t Au from 10 m; incl. 1 m @ 5.6 g/t Au.
- Borehole BBC4695: 5 m @ 7.83 g/t Au from 2 m; incl. 1 m @ 36.4 g/t Au.
- Borehole BBC4716: 5 m @ 10.27 g/t Au from 3 m, incl. 1 m @ 45.8 g/t Au.

True width is approximately 85% of the drill length.

Additional selected significant highlights of this drilling follow below.

- Borehole BBC4623: 9 m @ 1.26 g/t Au from 8 m.
- Borehole BBC4648: 5 m @ 2.37 g/t Au from 40 m, incl. 1 m @ 7.9 g/t Au.
- Borehole BBC4709: 11 m @ 1.36 g/t Au from 11 m.
- Borehole BBC4712: 2 m @ 5.30 g/t 49 m Au, incl. 1 m @ 10.1 g/t Au.

True width is approximately 85% of the drill length. The mineralized drill hole intervals listed above are based on a lower cut-off grade of 0.45 g/t Au, a minimum width of 2 m, and up to a maximum of 1.5 m of internal dilution.

10.3.3 2018 Drilling Program

Orezone planned to complete 13,000 m of RC oxide drilling and 9,000 m of oxide and sulphide drilling at Bomboré in 2018. In-fill drilling targeting discrete high-grade oxide mineralization was planned, in order to better define the geometry of the high-grade zones for future Mineral Resource modelling purposes.

The Q1 2018 drill program outside of the established Mineral Resources consisted of 55 RC boreholes totalling 2,904 m and 34 diamond core boreholes totalling 4,916.5 m. The drill program targeted the P17S Deposit, KT Prospect and P13 Prospect areas.

P17S Deposit – February 2018

Between 25 May 2017 and 22 February 2018, 14 core holes totalling 3,135 m were completed in the P17S Deposit areas. Selected highlights from the P17S Deposit area drilling follow below.

- Borehole BBD0985: 13 m @ 2.26 g/t Au from 180 m, incl. 1.0 m @ 10.9 g/t Au.
- Borehole BBD0991: 4.6 m @ 15.96 g/t Au from 77 m, incl. 0.8 m @ 87.8 g/t Au.
- Borehole BBD0990: 5 m @ 6.38 g/t Au from 44 m, incl. 2.0 m @ 13.5 g/t Au.
- Borehole BBD0983: 16.3 m @ 1.97 g/t Au from 254 m.
- Borehole BBD0986: 8.2 m @ 1.88 g/t Au from 277.8 m.

True width is approximately 80% of drill intercept length.

The core drill results from the P17S target are in follow-up to previous drilling that defined several continuous, shallow plunging and higher-grade mineralized zones, which extend from surface to a depth of at least 215 m.

These higher-grade zones are associated with thickened fold hinges that plunge northwards with a down-plunge extent of at least 500 m from the main P17S Deposit outcrop, along a folded granodiorite unit, striking north-northeast between the P17S and P17 Deposits. The total strike extent is currently 1.8 km, including a 600-m gap that had yet to be drill tested.

In follow-up, 18 additional core holes totalling 3,040 m were planned to be drilled at P17S in 2018.

KT Prospect – February 2018

The KT Prospect is a near-surface target located northeast adjacent to the Bomboré North Area (see Figure 10.1.1). The 2018 RC drill program was in follow-up to historical reconnaissance drilling, specifically auger, RAB and RC higher-grade results outside of the Bomboré Mineral Resources, but in a similar geological environment.

This first 10 of 23 step-out RC holes planned were around five historical higher-grade intersections in the KT area. The drilling completed on three of the five KT targets was successful in determining that oxide mineralization remains open outside of the defined Mineral Resources, and that these new results are consistent with a shallow to moderate northeast-plunge for the discrete higher-grade mineralized zones at KT. Further follow-up drilling, including core holes to document the local structural setting of each target and the strike and dip of the discrete mineralized zones, was planned in the KT target area and other similar target areas on the Bomboré Property in 2018.

Highlights from the KT Target Area Drilling

Target KT-01 follow-up on BMB0864 RAB hole intersection of 4 m @ 3.76 g/t Au from 0 m:

- Borehole BBC4737: 3 m @ 2.03 g/t Au from 26 m.
- Borehole BBC4737: 4 m @ 1.23 g/t Au from 39 m.

Target KT-02 follow-up on BBC3839 scout RC hole intersection of 4 m @ 3.98 g/t Au from 8 m:

- Borehole BBC4734: 3 m @ 7.05 g/t Au from 20 m downhole, incl. 2 m @ 10.2 g/t Au.

Target KT-03 follow-up on BBC3618 scout hole intersection of 6 m @ 1.97 g/t Au from 36 m:

- Borehole BBC4740: 2 m @ 3.46 g/t Au from 43 m.

At that time, true widths had not been determined. The mineralized intervals are based on a lower cut-off grade of 0.45 g/t Au, a minimal width of 2 m, and up to a maximum of 2 m internal dilution.

Results from the KT Prospect show that the oxide resources remained open to expansion and add to the exploration potential of the overall Bomboré Project together with other similar targets, such as the P13 Prospect. Orezone planned to complete approximately 15,000 m of in-fill drilling to follow-up on the higher-grade drill results from 2017 and 7,000 m of drilling on outside targets to continue to expand the footprint of the Bomboré gold mineralization.

P17S Deposit – July 2018

The PS17S core drill results listed below are in follow-up to previous drilling (see P17S Deposit – February 2018 above) that defined several continuous shallow-plunging mineralized zones, which extend from surface to a depth of at least 215 m.

These mineralized zones are associated with thickened fold hinges plunging northwards with a down-plunge extent of at least 500 m for the main P17S zone outcrop, along a folded granodiorite unit, striking north-northeast between the P17S and P17 Deposits. The total strike extent is currently 1.8 km, including the 600 m gap that had yet to be drill tested. These new results continued to better define the extent and the geometry of the higher-grade P17S gold zones. The results highlighted below include final results from the 23 core hole (4,017 m) follow-up program completed at P17S in Q1 2018.

Selected Highlights from P17S Main Deposit Down-Dip and Down-Plunge Extension.

- Borehole TYD0112: 7.25 m @ 5.04 g/t Au from 148.3 m.
- Borehole TYD0106: 24.0 m @ 2.18 g/t Au from 91.0 m.
- Borehole TYD0102: 4.0 m @ 5.21 g/t Au from 83.0 m, incl. 2.0 m @ 9.9 g/t Au.
- Borehole TYD0108: 9.95 m @ 2.54 g/t Au from 126.3 m, incl. 1.0 m @ 8.2 g/t Au.
- Borehole TYD0103: 21.15 m @ 2.04 g/t Au from 5.5 m, incl. 5.2 m @ 3.8 g/t.
- Borehole TYD0103: 5.3 m @ 4.24 g/t Au from 63.5 m, incl. 2.0 m @ 6.0 g/t Au.

True width is approximately 80% of drill intercept length.

Selected highlights from P17S Northeast Extension Drilling.

- Borehole BBD0992: 7.0 m @ 2.56 g/t Au from 17.5 m, incl. 3.1 m @ 4.4 g/t Au.
- Borehole BBD1000: 18.0 m @ 2.71 g/t Au from 144.0 m, incl. 6.0 m @ 4.2 g/t; and 6.0 m @ 1.94 g/t from 190.0 m.
- Borehole BBD0996: 10.0 m @ 1.69 g/t Au from 153.0 m; 22.0 m @ 0.73 g/t Au from 169 m; 7.0 m @ 1.48 g/t Au from 249 m; and 15.4 m @ 1.01 g/t Au from 266 m.

At that time, true width had not been determined.

KT Prospect – July 2018

The 2018 RC drill program at KT was in follow-up to historical reconnaissance drilling by auger, RAB and RC (with high-grade hits) outside of the historically defined Mineral Resources, but in a similar geological environment. The KT drilling program consisted of 23 RC holes totalling 1,070 m and three core holes totalling 237.5 m completed in Q1 2018. This program successfully determined that the oxide mineralization remains open outside of defined Mineral Resources and that the higher-grade mineralized zones at KT consistently plunge shallowly to moderately northeast.

Highlights from the KT target area 2018 Drilling Program

<u>Target KT-01: follow-up on BMB0864 RAB hole that intercepted 4 m @ 3.76 g/t Au (from 0 m):</u>

- Borehole BBC4738: 2 m @ 1.31 g/t Au from 40 m.

<u>Target KT-03: follow-up on BBC3618 reconnaissance RC hole that intercepted 6 m @ 1.97 g/t Au (from 36 m):</u>

- Borehole BBC4741: 7 m @ 1.60 g/t from 15 m.
- Borehole BBD1001: 6 m @ 1.34 g/t from 27 m.

<u>Target KT-04: follow up on BBC3214 scout RC hole that intercepted 9 m @ 1.34 g/t Au (from 0m):</u>

- Borehole BBC4743: 10 m @ 12.88 g/t Au from 0 m, including 2 m @ 58.2 g/t from 2 m in colluvial material.

<u>Target KT-05: follow up on BMB1013 RAB hole that intercepted 4 m @ 20.7 g/t Au (from 0 m) in colluvial material:</u>

- Borehole BBC4753: 2 m @ 2.71 g/t Au from 9 m.
- Borehole BBC4751: 2 m @ 1.03 g/t Au from 29 m; and 4 m @ 1.14 g/t Au from 57 m.

At that time, true widths had not been determined. The mineralized intervals are based on a lower cut-off grade of 0.45 g/t Au, a minimal width of 2 m, and up to a maximum of 2 m internal dilution.

P13 Prospect

The P13 Prospect is located five km south-southwest of the Bomboré South Deposits area (see Figure 10 1 1).

The P13 RC drill program followed-up on historical reconnaissance drilling by auger and RC (with higher-grade results) outside of the defined Bomboré Mineral Resources, but in a similar geological environment. The drill program consisted of 25 RC holes totalling 1,447 m and six-core holes totalling 510 m completed at P13 in Q1 2018. This program successfully determined that oxide mineralization occurs along a potential strike length of approximately three km at P13. Selected drilling highlights from the P13 target area follow below.

<u>Target P13-02: follow-up on TYC0013 RC hole that intercepted 6 m @ 1.69 g/t Au from 23 m and TYC0012 RC hole that intercepted 4 m @ 1.95 g/t Au from 36 m:</u>

- Borehole TYD0115: 3 m @ 1.47 g/t Au from 48 m.
- Borehole TYC0178: 4 m @ 1.18 g/t Au from 10 m.
- Borehole TYC0177: 3 m @ 1.11 g/t Au from 26 m.

<u>Target P13-03: follow-up on TYC0089 RC hole that intercepted 3 m @ 1.00 g/t Au from 41 m and TYC0090 RC hole that intercepted 2 m @ 5.11 g/t Au from 48 m:</u>

- Borehole TYD0116: 2 m @ 5.74 g/t Au from 56.5 m.
- Borehole TYC0181: 3 m @ 1.41 g/t Au from 45.0 m.

<u>Target P13-05: follow-up on TYC0100 RC hole that intercepted 6 m @ 2.04 g/t Au from 14 m:</u>

- Borehole TYC0187: 4 m @ 3.67 g/t from 14 m; and 1 m @ 23.45 g/t from 37 m.
- Borehole TYD0117: 4 m @ 2.13 g/t Au from 44 m.
- Borehole TYC0186: 4 m @ 1.83 g/t Au from 19 m.
- Borehole TYC0185: 7 m @ 1.38 g/t Au from 2 m; and 3 m @ 1.13 g/t Au from 29 m.

<u>Target P13-06: follow-up on TYC0038 RC hole that intercepted 7 m @ 1.57 g/t Au from 25 m):</u>

- Borehole TYC0190: 3 m @ 1.50 g/t Au from 15 m.
- Borehole TYD0118: 6.5 m @ 1.12 g/t Au from 32.5 m.
- Borehole TYC0189: 7 m @ 1.58 g/t Au from 10 m.

<u>Target P13-09: follow-up on TYC0118 RC hole that intercepted 6 m @ 2.10 g/t Au (from 29 m):</u>

- Borehole TYC0201: 3 m @ 1.23 g/t Au from 16 m.
- Borehole TYD0120: 2 m @ 1.08 g/t Au from 31.5 m.
- Borehole TYD0118: 6.5 m @ 1.12 g/t Au from 32.5 m.
- Borehole TYC0200: 5 m @ 0.94 g/t Au from 3 m.

At that time, true width had yet to be determined. The mineralized intervals list above are based on a lower cut-off grade of 0.45 g/t Au, a minimal width of 2 m and up to a maximum of 2 m of dilution being included.

Results from targets KT and P13 show that the oxide resources also remain open to expansion and add to the exploration potential of the overall Bomboré Project.

Maga Deposit Area (Bomboré North Area)

Orezone's drilling program in Q2 2018 consisted of 196 RC boreholes totalling 12,171 m of oxide drilling and 30 diamond drill boreholes totalling 3,766 m of oxide and sulphide drilling in the Maga, P11 and Siga East Deposits areas. The drilling program was designed to demonstrate the presence, grade and continuity of discrete higher-grade zones within each of the Deposit areas. The results for each target deposit are summarized below.

Footwall Zone (Oxide and Sulphide Targets)

The drilling program targeted the northern portion of a 2.4-km Mineral Reserve pit shell, over a strike of 450 m in which 16 RC drill holes totalling 1,194 m and five diamond drill holes totalling 731 m were completed. In this area, the Footwall Zone mineralization is hosted in the nose of a mesoscopic, tightly folded sequence of carbonaceous meta-argillite interbedded with metasandstones over a strike length of approximately 1.2 km.

The geological sequence strikes N045° and dips approximately 75° south-southeast.

Selected Highlights from Maga Footwall Zone 2018 Drilling Program.

- Borehole BBC4935: 12 m @ 2.99 g/t Au from 63 m, including 1 m @ 26.4 g/t Au from 63 m.
- Borehole BBD1020: 7.10 m @ 2.30 g/t Au from 123.90 m (sulphide zone), including 1.15 m @ 8.9 g/t Au from 125.50 m; and 5.00 m @ 3.17 g/t Au from 161 m (sulphide zone), including 1.00 m @ 12.8 g/t from 161.00 m.
- Borehole BBC4936: 20 m @ 1.64 g/t Au from 23 m.

True width is approximately 90% of the drill intercept length and all intercepts are from the oxidized and semi-oxidized units, unless specified otherwise.

Maga Hill Zone (Oxide and Sulphide Targets)

The Maga Hill Zone is located on the Hanging Wall of the main Maga Footwall zone. The 2018 drilling program consisted of six RC holes totalling 367 m and ten diamond drill holes totalling 1,245 m, completed over a strike length of 175 m. The gold mineralization is hosted in a tightly folded sequence of meta-sedimentary rocks, biotite schist (meta-gabbro), and meta-granodiorite. The sequence strikes N045° and dips about 70° south-southeast.

Selected Highlights from Maga Hill Drilling Program.

- Borehole BBD1033: 9.00 m @ 3.09 g/t Au from 22 m, and 10.00 m @ 2.29 g/t Au from 93 m (sulphide zone).
- Borehole BBD1032: 5.00 m @ 2.07 g/t Au from 64.5 m.
- Borehole BBC4955: 16.00 m @ 4.62 g/t Au from 29 m, including 6 m @ 8.10 g/t Au.
- Borehole BBD1031: 7.50 m @ 1.96 g/t Au from 5.5 m; 15.50 m @ 1.21 g/t Au from 35 m.
- 6.55 m @ 2.80 g/t Au from 123 m (sulphide zone).
- Borehole BBD1030: 13.00 m @ 1.13 g/t Au from 16 m.
- Borehole BBD1024: 23.50 m @ 4.31 g/t Au from 21 m, including 4.20 m @ 19.0 g/t Au.
- Borehole BBD1025 2.20 m @ 5.76 g/t Au from 68.8 m (sulphide zone), including 1.20 m @ 8.9 g/t Au.

True width is approximately 95% of the drill intercept length and all intercepts are from the oxidized and semi-oxidized units, unless specified otherwise.

3653 Zone (Oxide targets)

The 3653 Zone is located on the Hanging Wall of the main Maga Footwall Zone. The 2018 drill program here consisted of eight RC holes totalling 777 m completed over a strike length of 100 m (the oxide zone is open at both ends). The gold mineralization is hosted along the Footwall contact of a porphyritic granodiorite intruding metasandstone. The geological sequence at 3653 Zone strikes N045° and dips approximately 75° south-southeast.

Selected Highlights from 3653 Zone

- Borehole BBC4943: 14 m @ 1.26 g/t Au from 20 m.

- Borehole BBC4944: 10 m @ 1.11 g/t Au from 56 m.
- Borehole BBC4947: 10 m @ 2.20 g/t Au from 21 m, including 1 m @ 13.4 g/t Au from 22 m.

True width is approximately 90% of the drill intercept length and all intersections are from the oxidized and semi-oxidized units.

3645 Zone - Oxide and Sulphide Targets

The 3645 Zone is located on the Hanging Wall of the main Maga Footwall Zone. The drilling program consisted of 16 RC holes totalling 1,201 m and three diamond drill holes totalling 435 m, completed over a strike length of 375 m. The gold mineralization is hosted within the same porphyritic granodiorite as the 3653 zone, on its structural Hanging Wall. The sequence strikes N025° and dips approximately 70° east-southeast.

Selected Highlights from 3645 Zone Drilling Program in 2018

- Borehole BBC4917: 11 m @ 1.03 g/t Au from 15 m.
- Borehole BBD1018: 4.0 m @ 17.80 g/t Au from 109 m (sulphide zone), including 1 m @ 64.4 g/t.
- Borehole BBC4906: 9 m @ 1.14 g/t Au from 37 m.

True width is approximately 95% of the drill intercept length and all intercepts are from the oxidized and semi-oxidized units, unless specified otherwise.

P11 Deposit Area (Bomboré South Area)

The 2018 program at the P11 Deposit was designed to complete the 25 m by 25 m definition drilling pattern in the oxide zone over a strike length of 1.2 km, and to step down into the top portion of the sulphide zone.

The gold mineralization is hosted in a sequence of metasedimentary rocks, biotite schist (meta-gabbro), and meta-granodiorite. This lithostratigraphic sequence defines a tight synformal fold open to the north, with a west limb striking N350° and an east limb striking N360°. The lithological contacts, regional foliation, and mineralized envelopes all dip approximately 65° to the east-northeast. The drilling program consisted of 74 RC drill holes totalling 4,015 m and 12 diamond drill holes totalling 1,346 m. Selected highlights from the P11 drilling program are listed below.

- Borehole BBC4901: 8 m @ 1.94 g/t Au from 61 m, including 1 m @ 5.4 g/t Au.
- Borehole BBC4897: 6 m @ 6.12 g/t Au from 43 m, including 3 m @ 9.9 g/t Au.

- Borehole BBC4867: 4 m @ 3.64 g/t Au from 28 m, including 1 m @ 10.0 g/t Au.
- Borehole BBD1012: 1 m @ 48.85 g/t Au from 33 m.
- Borehole BBD1007: 5 m @ 2.17 g/t Au from 43 m, and 13.00 m @ 0.82 g/t Au from 94 m (sulphide zone).
- Borehole BBC4835: 13 m @ 1.35 g/t Au from 1 m, including 1 m @ 8.8 g/t Au.

True width is approximately 95% of the drill intercept length and all intercepts are from the oxidized and semi-oxidized units, unless specified otherwise.

Siga East Deposit Area (Bomboré South Area)

The 2018 drilling program is a follow-up to the 2017 Siga East RC drilling program (see Orezone's press release of 12 September 2017). The 2018 program was designed to complete the 25 m by 25 m definition drilling pattern in the oxide zone over a strike length of 800 m along the main Footwall Zone, and to advance the definition of various Hanging Wall mineralized zones. The gold mineralization is hosted in a sequence of metasedimentary rocks, biotite schist (meta-gabbro), and meta-granodiorite. This lithostratigraphic sequence defines a large S-fold with long limbs striking N350° and a short limb striking N360°. The drilling program consisted of 71 RC drill holes totalling 4,301 m.

Siga East Footwall Zone - Oxide Targets

The Siga East Footwall Zone is located along the west limb of the S-fold. The lithological contacts, the regional foliation, and the mineralized envelopes all dip approximately 55° east-northeast.

Selected Highlights from Siga East Footwall Zone 2018 Drilling

- Borehole BBC4829: 18 m @ 0.98 g/t Au from 0 m.
- Borehole BBC4830: 13 m @ 1.17 g/t Au from 10 m, including 1 m @ 6.2 g/t Au.
- Borehole BBC4831: 3 m @ 24.96 g/t Au from 52 m, including 1 m @ 74.0 g/t Au.
- Borehole BBC4806: 2 m @ 7.59 g/t Au from 4 m.
- Borehole BBC4807: 4 m @ 14.36 g/t Au from 12 m, including 1 m @ 54.7 g/t Au.
- Borehole BBC4803: 3 m @ 3.45 g/t Au from 55 m, including 1 m @ 8.1 g/t Au.
- Borehole BBC4804: 11 m @ 1.02 g/t Au from 33 m.

- Borehole BBC4788: 10 m @ 1.47 g/t Au from 16 m, including 1 m @ 5.1 g/t; and 15 m @ 1.34 g/t Au from 55 m.

True width is approximately 100% of the drill intercept length and that all intersections are from the oxidized and semi-oxidized units.

Siga East Hanging Wall Zones - Oxide Targets

The Siga East Hanging Wall Zones are located along the short limb of the S-fold.

Selected Highlights from Siga East Hanging Wall Zones 2018 Drilling Program.

- Borehole BBC4824: 7 m @ 1.52 g/t Au from 12 m.
- Borehole BBC4799: 2 m @ 10.64 g/t Au from 63 m.
- Borehole BBC4785: 4 m @ 3.65 g/t Au from 43 m, including 1 m @ 12.2 g/t Au.
- Borehole BBC4777: 14 m @ 1.28 g/t Au from 0 m.
- Borehole BBC4779: 20 m @ 1.09 g/t Au from 0 m.

True width had not been determined and all intercepts are from oxidized and semi-oxidized units).

The mineralized intervals are based on a lower cut-off grade of 0.45 g/t Au, a minimal width of 2 m, and up to a maximum of 2 m of internal dilution between samples above the lower cut-off grade.

The 2018 drilling program was successful in better defining the geometry (strike, dip and plunge) and continuity of the shallow portion of these Deposit areas. The drilling program completed the 25 m by 25 m definition drilling pattern in the oxide zone and stepped down into the top portion of the sulphide zone.

Based on these results, further infill and step out definition drilling was warranted in all four deposit areas, in both the oxidized and sulphide zones.

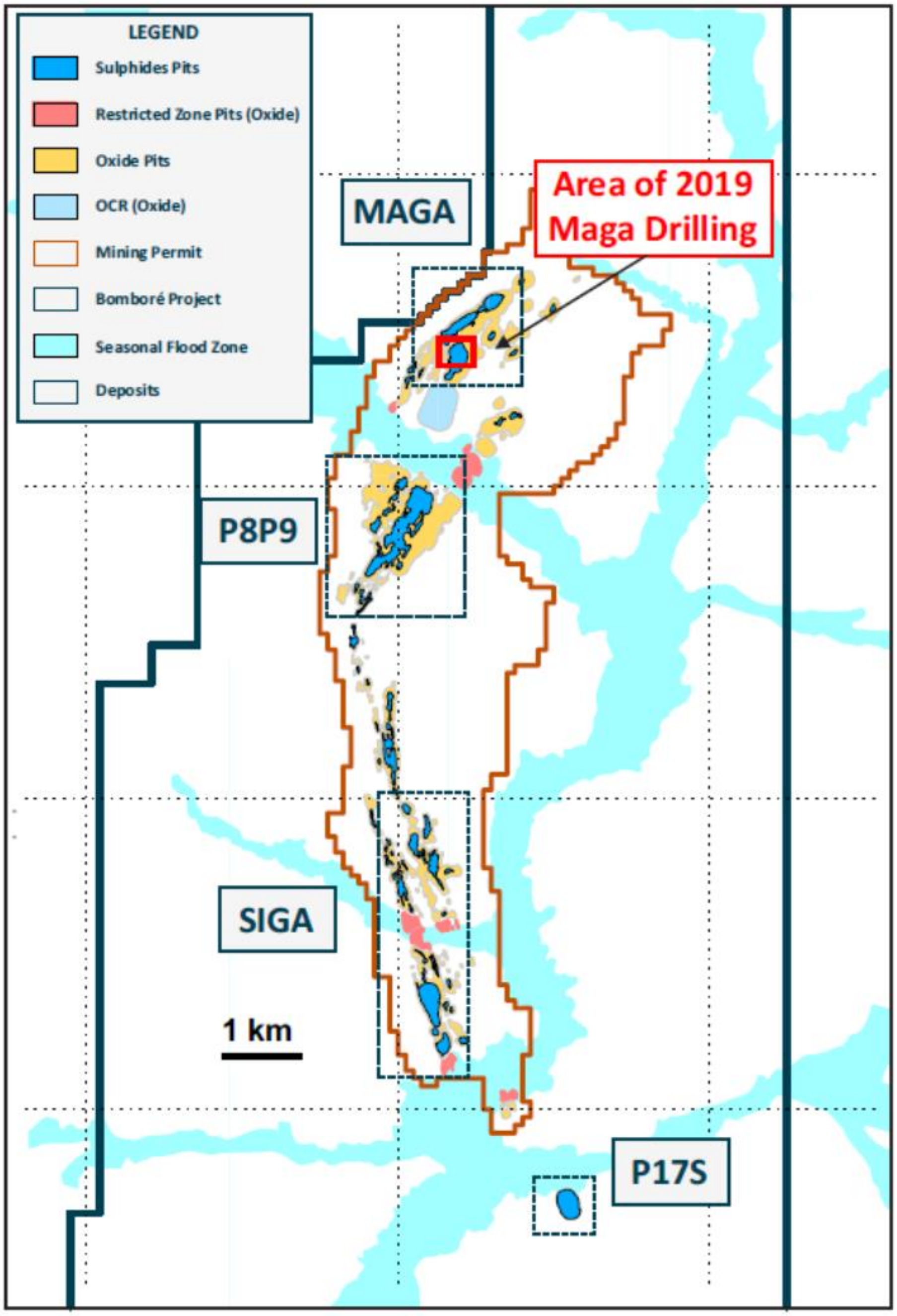
10.3.4 2019 Drilling Program

In September 2019, Orezone commenced a 2,000 m RC and DD drilling program at the Maga Hill Deposit area (Figure 10.3.3), in follow-up to higher than the average grade mineralization intersected in 2018. The 2019 drilling program consisted of 1,374 m of diamond drilling and 608 m of RC drilling. Its purpose was to advance the definition of high-grade, shallowly plunging gold mineralization on the Hanging Wall of the main Maga Deposit.

In addition, a small grade-control drilling program was also completed on the Maga Hill Deposit in May 2019, specifically on the western fringe of the main Maga Hill Footwall zone.

Highlights of the pre-2018 drilling in the Maga Deposit area follow below (Table 10.3.2). Highlights from the 2018 drilling program are listed above in the section Maga Deposit Area (Bomboré North Area).

Figure 10.3.3 2019 Drilling Program at Maga Deposit Area (Bomboré North Area)



Source: Orezone press release dated September 9, 2019.

Note: The mining lease shown is that of 2018.

Table 10.3.2 Highlights of Pre-2018 Maga Deposit Area Drilling

Borehole	Year	Section	From m	To m	Length m	Au g/t
BBC0117	2011	3675	35	52	17.00	2.38
BBC0117	2011	3675	125	137	12.00	2.58
BBC2998	2012	3650	34	55	21.00	2.51
BBD0803	2012	3625	43.5	49	5.50	7.07
BBD0803	2012	3625	105	113	8.00	5.88
BMC0063	2008	3625	8	34	26.00	2.06
BBC3004	2012	3600	22	31	9.00	4.76
BBD0052	2008	3600	50	56	6.00	5.36
BBD0911	2012	3600	153	160	7.00	6.04
BBC0720	2012	3575	6	12	6.00	5.26
BBD0118	2005	3575	118	128	10.00	4.80
BBC2898	2012	3550	12	26	14.00	2.53
BBC3001	2012	3550	8	16	8.00	10.36
BBD0903	2010	3550	112	117	5.00	7.15
BBD0133	2011	3475	97.5	111	13.50	3.50

The majority of the established Mineral Resources at the Bomboré Property were contained within a low-grade shear-zone drilled over a strike length of 11 km and width of 200 m. Drilling in 2017 and 2018 at Maga, P11 and Siga East areas demonstrated the presence of discrete higher-grade zones within the Mineral Resource model.

The 2019 drilling program tested and validated the geological interpretation of a higher-grade plunging gold system within the lower-grade mineralization. Higher-grade gold mineralization was intersected in most of the drill holes. Significant results from the 2019 drill program are presented below in Table 10.3.3.

Table 10.3.3 2019 Drill Program Results

Borehole	Section	From m	To m	Length m[1,2]	Au g/t
BBD1034	3650	122.80	131.70	8.90	3.21
including		127.70	128.70	1.00	16.25
BBD1035	3662.5	12.50	21.20	8.70	1.52
including		16.00	17.00	1.00	4.05
and		77.60	87.10	9.50	2.15
including		80.10	84.80	4.70	2.98
BBD1037	3662.5	108.30	115.50	7.20	2.81
including		111.40	112.40	1.00	8.85
BBD1038	3662.5	129.00	141.00	12.00	1.94
including		139.00	140.00	1.00	15.10
BBD1039	3675	63.00	75.00	12.00	1.35
including		71.80	73.00	1.70	4.82
BBD1040	3675	152.00	161.60	9.60	2.20
including		158.65	459.65	1.00	7.66
BBD1041	3687.5	70.00	84.80	14.80	1.63
including		81.00	82.00	1.00	7.89
BBC4995	3650	8.00	16.00	8.00	3.04
including		9.00	12.00	3.00	5.71
BBC4996	3650	27.00	45.00	18.00	3.68
including		34.00	42.00	8.00	7.31
BBC4997	3650	35.00	39.00	4.00	3.74
including		35.00	36.00	1.00	13.35
BBC4999	3662.5	22.00	33.00	11.00	1.89
including		29.00	33.00	4.00	4.53
BBC5000	3662.5	38.00	50.00	12.00	3.09
including		39.00	45.00	6.00	5.16
BBC5001	3662.5	52.00	54.00	2.00	6.90
including		52.00	53.00	1.00	12.95
and		57.00	62.00	5.00	7.70
including		57.00	61.00	4.00	9.48
BBC5004	3687.5	51.00	66.00	15.00	1.38
including		54.00	57.00	3.00	3.44
GCC0047	3525	31.00	38.00	7.00	1.50
GCC0052	3537.5	5.00	19.00	14.00	1.84

Borehole	Section	From m	To m	Length m[1,2]	Au g/t
GCC0054	3512.5	7.00	14.00	7.00	1.48
GCC0055	3512.5	15.00	20.00	5.00	2.38
including		16.00	17.00	1.00	7.43

Source: Orezone press release (November 17, 2019)

[1]The mineralized intervals are based on a lower cut-off grade of 0.45 g/t Au, a minimal width of 2 m, and up to a maximum of 2 m of dilution being included between samples above the lower cut-off grade.

[2]True width of the gold mineralization is approximately 90% of the drilled length.

10.3.5 2021 Drilling Program

The 2021 drilling program was planned to be executed in two phases that were focused on expansion of the P17S Deposit. The Phase I drilling was completed to the northeast of the P17S 2019 Mineral Reserves and Mineral Resources and intersected several wide, higher-grade gold intersections near-surface and outside the Mineral Reserves (Table 10.3.4 and Figure 10.3.4). Phase I drilling also successfully intersected broad zones of gold mineralization at depth, extending the down-plunge strike of the P17S NE Deposit into the untested “Gap Zone” by 150 m (Figure 10.3.5). The P17S NE Deposit remains open at depth and to the north towards P17, which occurs at surface.

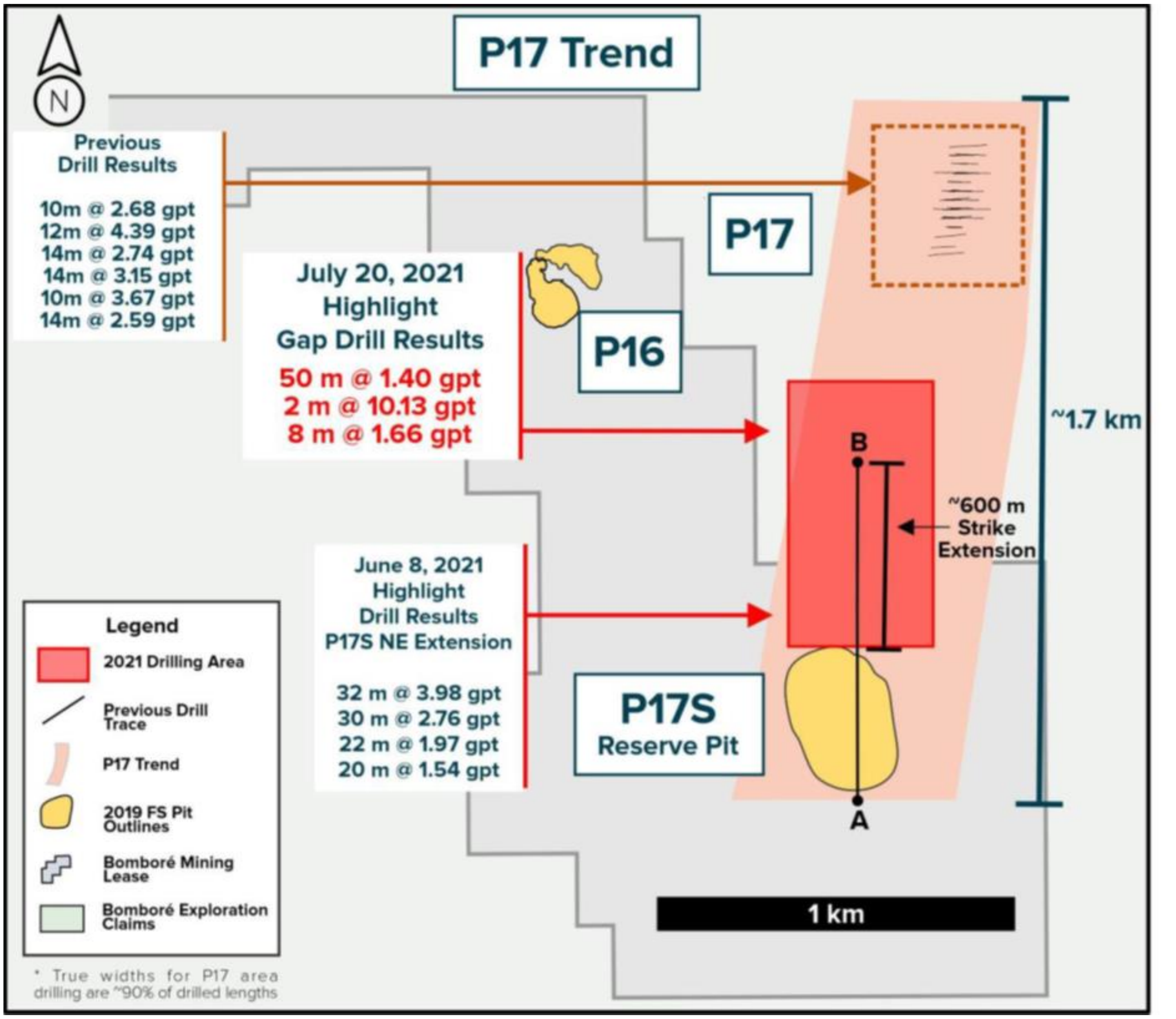
Table 10.3.4 P17S Area 2021 Selected Phase I Drill Results

Borehole	From m	To m	Length m[1,2]	Grade g/t Au
BBD1048	15.00	45.00	30.00	2.76
including	17.00	25.00	8.00	5.47
BBD1060	54.00	74.00	20.00	1.54
including	54.00	70.00	16.00	1.70
BBD1066	25.00	57.00	32.00	3.98
including	34.00	40.00	6.00	14.67
BBD1068	88.00	109.00	21.00	1.56
and	124.00	140.45	16.45	2.35
including	126.00	129.00	3.00	6.12
BBD1070	13.00	22.00	9.00	2.06
including	15.00	17.00	2.00	5.48
and	46.00	66.60	20.60	0.95
BBD1074	246.00	257.00	11.00	1.51
and	269	284	15.00	2.85
and	290	296	6.00	1.90

[1]The mineralized intervals are based on a lower cut-off grade of 0.45 g/t Au, a minimal width of 1.5 m, and up to a maximum of 2.0 m of dilution being included between samples above the lower cut-off grade.

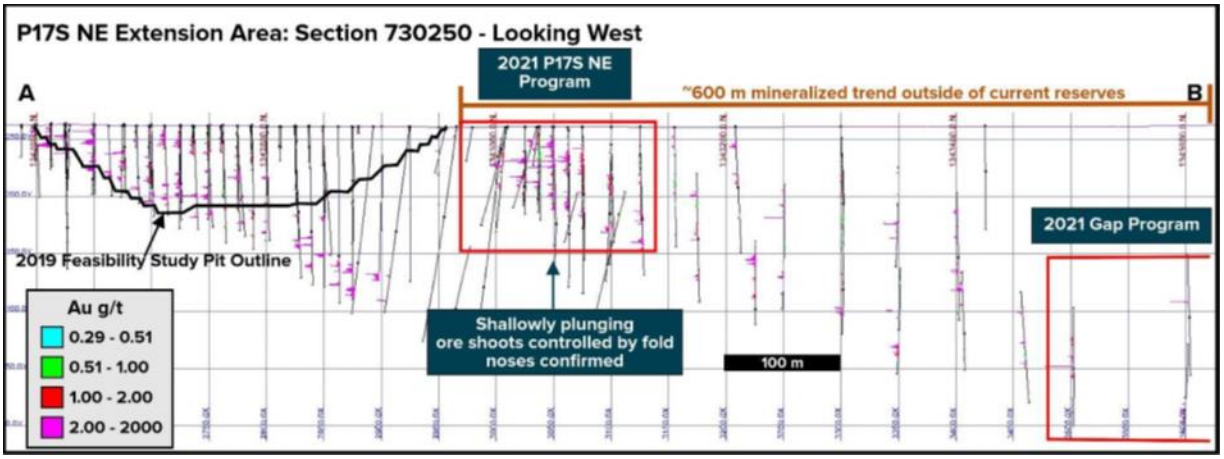
[2]True widths for P17S trend area are approximately 90% of drilled lengths.

Figure 10.3.4 P17 Trend Plan Map



Source: Orezone press release dated July 20, 2021.

Figure 10.3.5 P17S NE Extension Long Sectional Projection 730,250 m – Looking West



Source: Orezone press release (June 8, 2021).

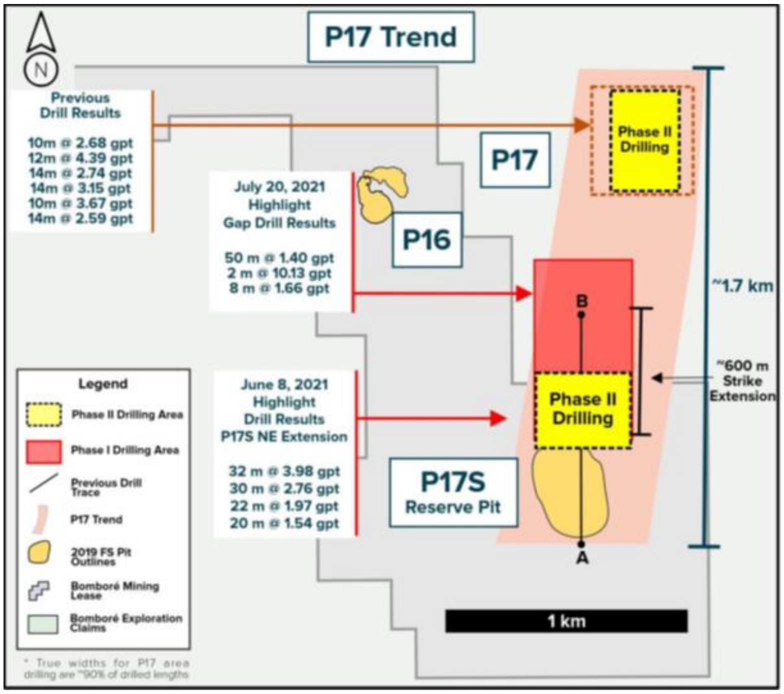
Additionally, the Phase I drilling program also tested the continuity of higher-grade mineralization into the 'Gap Zone' between the P17S and P17 Deposits. This 600 m gap between the P17S NE extension and P17 to the north had not previously been drilled. Phase I drilling here (see Figure 10.3.5) intersected broad zones of mineralization at depth, extending the down-plunge strike of the P17S NE deposit into the Gap Zone by 150 m. The P17S NE Deposit remains open at depth and to the north. The Phase I drilling program was completed in June 2021.

The Phase II drilling was planned to test the possible down-dip and lateral extensions of the P17 area, previously drilled by Orezone between 2007 and 2014, but not fully explored during that time. Phase II drilling commenced in December 2021 and continued into 2022.

10.3.6 2022 Phase II Drilling Program

The Phase II drilling program was focused in the areas of the P17S Zone Extension Zone and at the P17 Deposit. An objective of Phase II was to better define the plunge of the high-grade folds and the continuity between the P17S and P17 Deposits, approximately 1.7 km to the north. This drilling program includes near-surface drilling to the north of the P17S Mineral Reserve pit to follow-up on Phase I drilling, which intersected a new near-surface mineralized zone just outside the pit and where the best hole on the Bomboré Project to date was drilled (i.e., drill hole BBD1066 returned 32 m @ 3.98 g/t Au) (Figure 10.3.6).

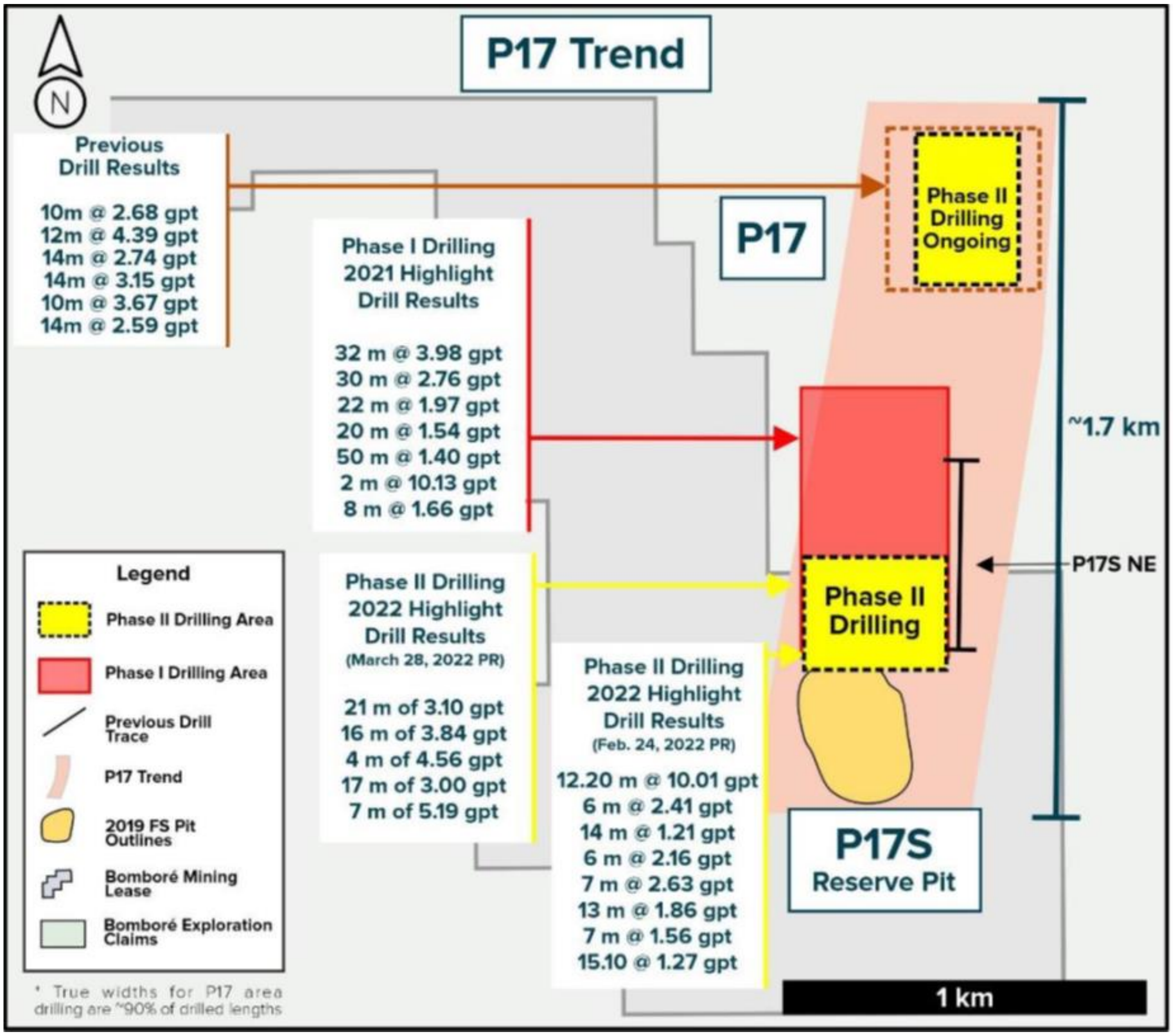
Figure 10.3.6 P17 Area Map showing Phase II Drill Program Plan for 2021-2022



Source: Orezone press release dated June 20, 2021.

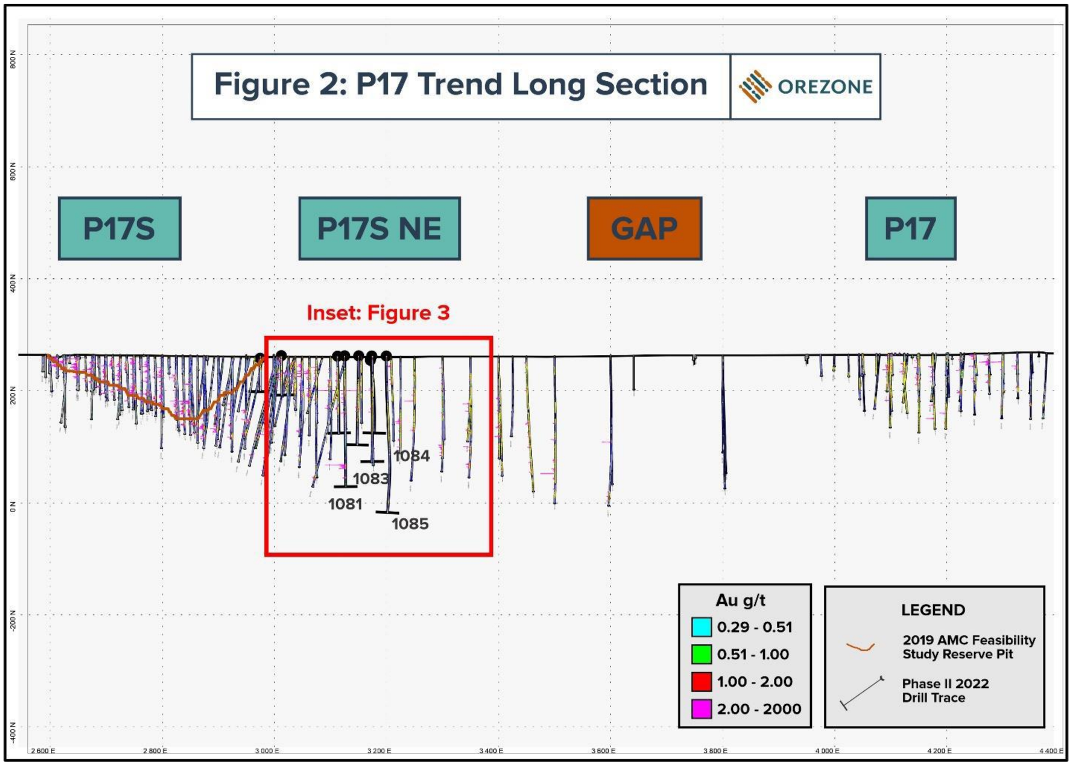
The Phase II drilling program results are presented in Figures 10.3.7 to 10.3.9 and Table 10.3.5. The drilling results confirmed the continuity of mineralization at surface and down-dip along the P17S-NE Zone. The mineralized system remains open to expansion by drilling at depth and to the north.

Figure 10.3.7 Phase II Drill Results at P17 Deposit Trend



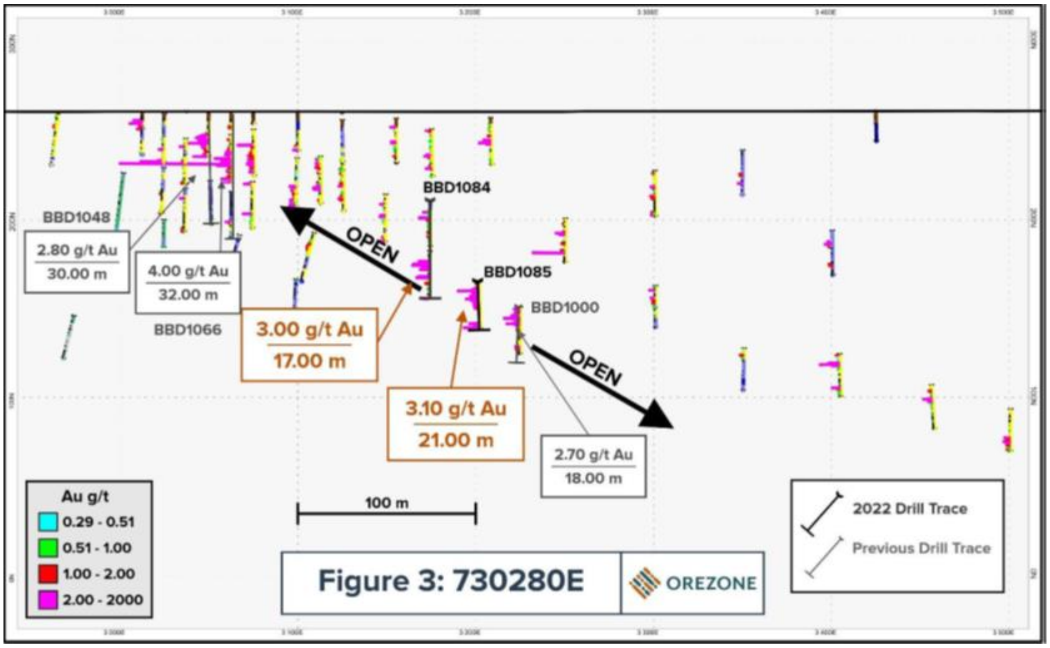
Source: Orezone press release dated March 28, 2022.

Figure 10.3.8 P17 Deposit Trend Longitudinal Section Projection



Source: Orezone press release dated March 28, 2022.

Figure 10.3.9 Longitudinal Section Projection 730280E – Continuity of High-Grade Shoots



Source: Orezone press release dated March 28, 2022.

Table 10.3.5 P17S Deposit Area Phase II Drilling Highlights

Deposit / Area	Borehole	From m	To m	Length m[1,2]	Au g/t
P17S - NE Extension	BBD1077	33.00	47.00	14.00	1.21
	and	51.00	57.00	6.00	2.16
	and	86.00	100.00	14.00	1.12
	and	117.00	124.00	7.00	2.63
	incl.	121.00	122.00	1.00	5.29
	BBD1078	5.00	18.00	13.00	1.85
	incl.	8.00	9.00	1.00	6.85
	BBD1079	47.00	54.00	7.00	1.56
	BBD1080	13.00	28.10	15.10	1.33
P17S Main Zone at Depth	BBD1081	79.00	85.00	6.00	2.41
	incl.	84.00	85.00	1.00	6.83
	and	245.80	258.00	12.20	10.04
	incl.	246.60	257.15	10.55	11.52
	and	276.10	279.70	3.60	3.27
	incl.	278.50	279.70	1.20	5.55

Deposit / Area	Borehole	From m	To m	Length m[1,2]	Au g/t
P17/S-NE	BBD1084	108.00	125.00	17.00	3.03
	incl.	108.00	115.00	7.00	5.19
	incl.	119.00	121.00	2.00	6.22
	BBD1085	123.00	144.00	21.00	3.06
	incl.	128.00	144.00	16.00	3.84
	and	154.00	158.00	4.00	4.56

[1]The mineralized intervals are based on a lower cut-off grade of 0.45 g/t Au, a minimal width of 1.5 m, and up to a maximum of 2.0 m of dilution being included between samples above the lower cut-off grade.

[2]True widths for P17S area drilling are approximately 90% of drilled lengths.

Based on the positive exploration results, Orezone significantly increased its exploration budget for the remainder of 2022 by US$9 million. For this Phase III drilling program, a total of ~77,000 m of definition and exploration drilling was planned on the Project over the next nine months. A second drill rig was mobilized to continue drilling at P17S and a third rig was mobilized in late Q2 2022.

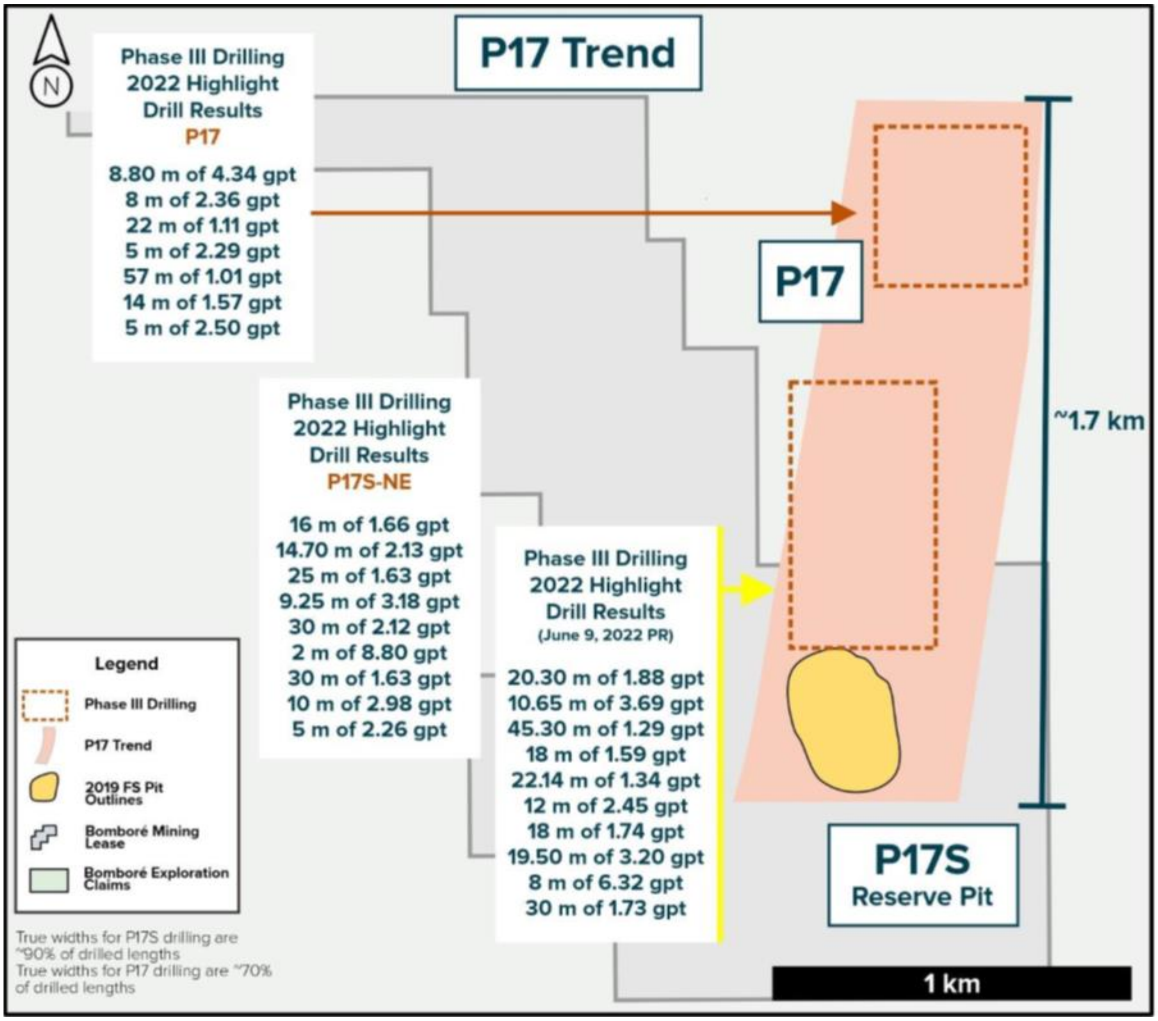
The primary focus of the Phase III drilling was to convert Inferred Mineral Resources into Measured and Indicated Mineral Resources. In addition to the P17 Deposit trend, drilling was also completed at Maga, Siga and other key deposits within the mining concession.

10.3.7 2022 Phase III Drilling in the P17S Area

The focus of the Phase III drilling on the P17 Deposit Trend was to better define the continuity of the repeated, folded high-grade granodiorite mineralized zones at P17S and to test the continuity and expansion potential of P17 located 1 km to the north of P17S, which was previously drilled between 2007 and 2013. The drilling results are shown in Figures 10.3.10 to 10.3.12 and Table 10.3.6.

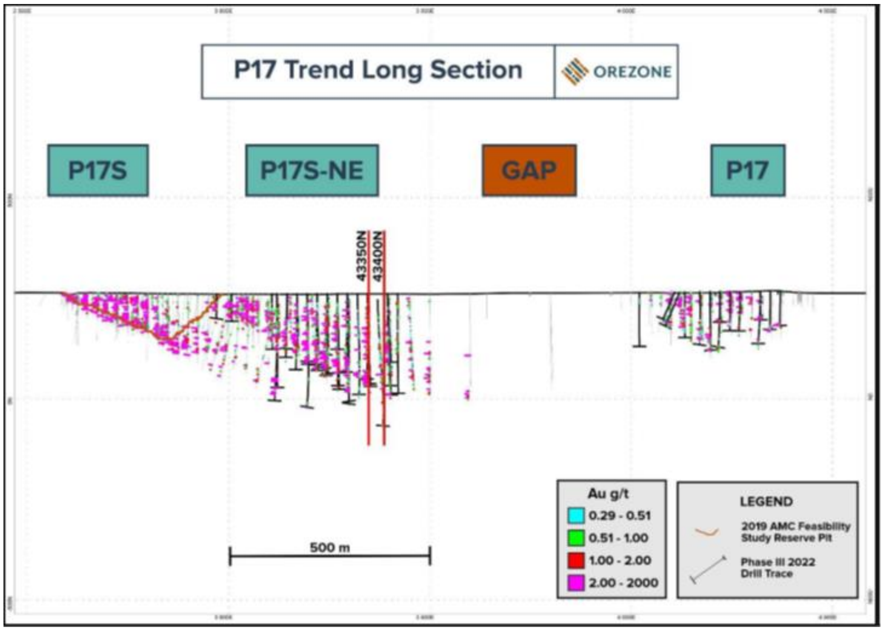
The drill results confirmed the following five features of the P17 Deposit Trend: 1) the down-plunge continuity of the mineralized zones at P17S; 2) the mineralized system at P17S remains open above and below the drilled portion of the granodiorite on most sections, which warrants follow-up definition drilling; 3) a significant new intercept on the deeper extension of the main zone beneath the P17S-NE deposit and outside of the current Mineral Resource envelopes indicate good potential to further expand the main P17S deposit; 4) results from P17 have confirmed continuity of existing shallowly north-plunging mineralized zones and have further extended the P17 Trend, with a new high-grade shallow intercept to the north; and 5) P17 remains open to expansion by drilling at depth and to the north.

Figure 10.3.10 P17 Deposit Trend 2022 Phase III Drilling Results



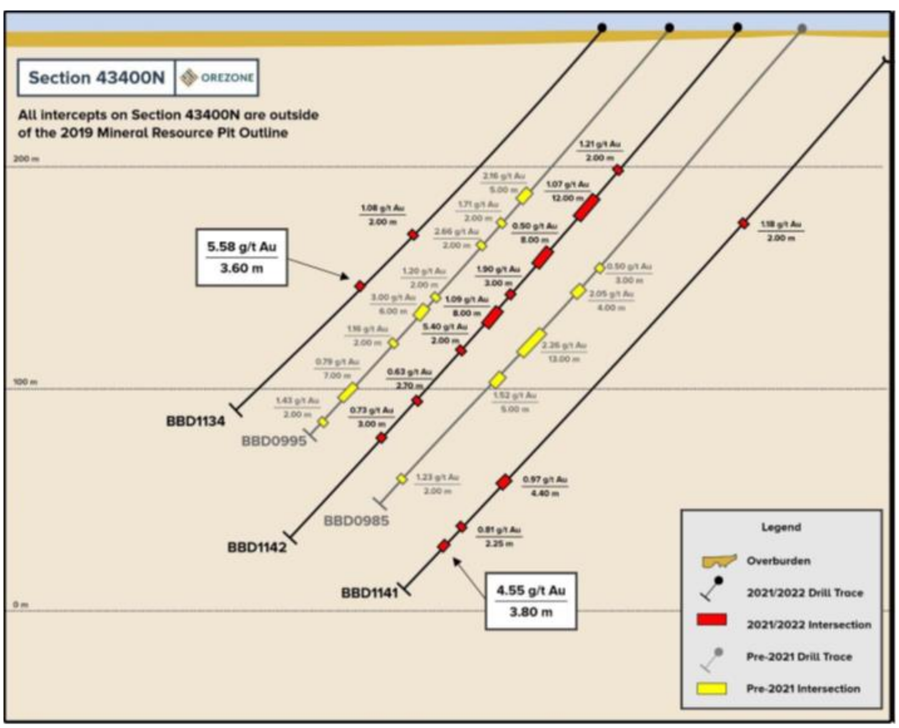
Source: Orezone press release dated June 14, 2023.

Figure 10.3.11 P17 Deposit Trend Longitudinal Section Projection Looking West



Source: Orezone press release dated September 13, 2022.

Figure 10.3.12 Cross Section Projection 43350 Looking North



Source: Orezone press release dated September 13, 2022.

Table 10.3.6 P17 Phase III Drilling Results

Deposit / Area	Borehole	From m	To m	Length m[1,2,3]	Au g/t
P17	BBD1086	80.20	89.00	8.80	4.34
	incl.	82.00	87.00	5.00	6.20
	BBD1088	59.00	67.00	8.00	2.37
	incl.	66.00	67.00	1.00	12.35
	BBD1094	124.00	146.00	22.00	1.11
	incl.	124.00	129.00	5.00	2.29
	BBD1096	34.00	91.00	57.00	1.01
	incl.	54.00	68.00	14.00	1.57
	and incl.	86.00	91.00	5.00	2.50
P17S NE Extension	BBD1100	48.00	64.00	16.00	1.66
	incl.	55.00	56.00	1.00	13.60
	BBD1102	190.00	204.70	14.70	2.13
	BBD1103	42.00	50.00	8.00	1.31
	and	54.00	66.00	12.00	0.97
	and	74.00	79.00	5.00	1.78
	incl.	83.00	108.00	25.00	1.63
	and incl.	106.10	107.10	1.00	25.15
	BBD1114	99.00	129.00	30.00	2.12
	incl.	100.00	102.00	2.00	8.48
	BBD1115	124.00	154.00	30.00	1.63
	incl.	124.00	134.00	10.00	2.98
	and	166.00	171.00	5.00	2.26
P17S Main	BBD1113	294.30	303.55	9.25	3.18
P17S	BBD1104	134.00	143.00	9.00	2.69
	incl.	171.00	178.00	7.00	2.13
		182.00	191.30	9.30	2.45
	BBD1108	103.30	135.00	31.70	1.23
	BBD1108	106.70	109.00	2.30	4.66
	incl.	106.70	107.80	1.10	9.04
	and	190.00	200.65	10.65	3.69
	incl.	193.00	198.20	5.20	6.40
	BBD1110	189.00	207.00	18.00	1.75
	incl.	193.00	207.00	14.00	2.15

Deposit / Area	Borehole	From m	To m	Length m[1,2,3]	Au g/t
	incl.	232.00	234.30	2.30	8.56
	BBD1118	101.00	111.00	10.00	2.28
	and	134.00	153.70	19.70	1.49
	and	179.85	202.00	22.15	1.34
	BBD1119	29.00	32.00	3.00	3.44
	and	76.00	88.00	12.00	2.45
	incl.	82.00	83.00	1.00	9.95
	BBD1120	124.00	143.00	19.00	1.74
	incl.	126.00	127.00	1.00	11.74
	BBD1131	230.50	250.00	19.50	3.35
	incl.	233.30	240.00	6.70	6.68
	BBD1132	127.00	135.00	8.00	6.77
	incl.	127.00	133.00	6.00	8.47
	and	245.00	275.00	30.00	1.89
	BBD1124	144.00	162.00	18.00	0.67
	and	245.00	257.00	12.00	2.13
	incl.	245.00	246.00	1.00	19.37
	BBD1125	257.00	280.00	23.00	2.84
	incl.	269.00	280.00	11.00	5.26
	BBD1133	86.00	94.00	8.00	3.40
	incl.	88.00	89.00	1.00	17.99
	and	101.00	112.00	11.00	0.96
	incl.	110.00	111.00	1.00	4.35
	and	130.00	133.00	3.00	4.15
	incl.	131.00	132.00	1.00	10.13
	and	167.00	176.00	9.00	1.53
	incl.	174.00	175.00	1.00	6.17
	BBD1134	157.00	160.60	3.60	5.58
	incl.	158.00	160.60	2.60	7.34
	BBD1141	312.20	316.00	3.80	4.55
	BBD1143	186.00	188.00	2.00	0.97
		192.00	200.00	8.00	2.21
	incl.	192.00	194.00	2.00	7.54

[1]The mineralized intervals are based on a lower cut-off grade of 0.45 g/t Au, a minimal width of 1.5 m, and up to a maximum of 3.0 m of dilution being included between samples above the lower cut-off grade.

[2]True widths for P17 drilling are approximately 70% of drilled lengths.

[3]True widths for P17S NE Extension, Main and P17S drilling are approximately 90% of drilled lengths.

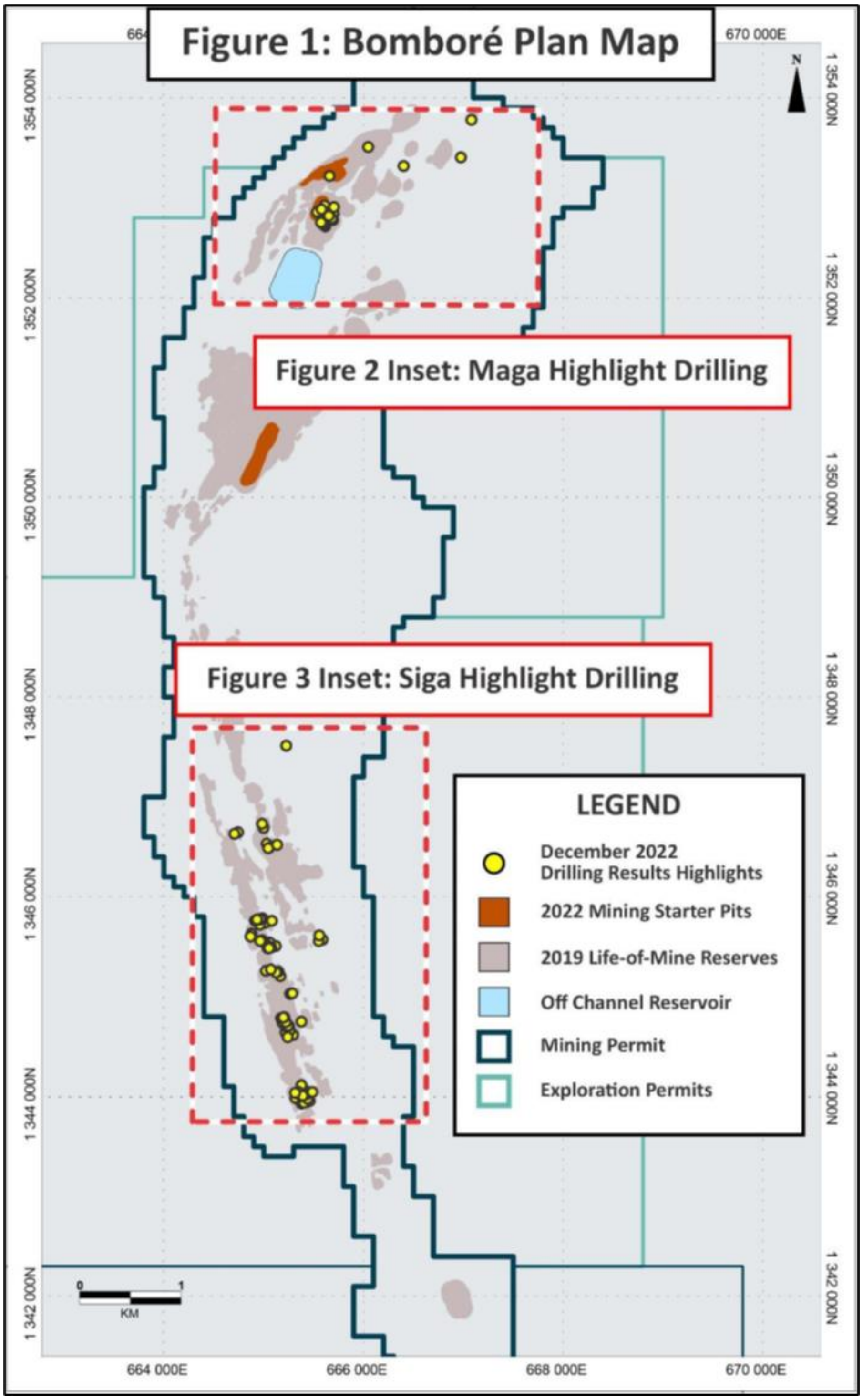
10.3.8 2022 Phase III Drilling at Maga Hill, Maga Main and Siga

Outside of the P17S Deposit area, the Phase III drilling in 2022 focused on the Maga Hill, Maga Main, P8P9 and Siga Deposits.

The results of the Phase III Mineral Resource definition drill program at the Maga Hill, Maga Main, Siga and P8P9 Deposits are summarized in this sub-section of the Report. The focus of the Phase III drilling was the conversion of Inferred Mineral Resources to Measured & Indicated Mineral Resources ahead of the upcoming sulphide expansion study. A key aspect of the infill program at Maga Hill was targeting higher-grade granodiorite folded structures that were intercepted in previous drilling of the near-surface oxide zones.

The drilling results confirmed the presence of high-grade mineralization within the sulphide zone at Maga Hill and down-dip continuity at the main hanging wall Siga Zone. Drilling within the Inferred mineralized zones at Siga South and Siga West was successful and at Maga Main and Maga Hill were also encouraging. Drill hole collar locations are shown in Figure 10.3.13. Highlight drilling results for Maga Hill and Siga are listed in Tables 10.3.7 and 10.3.8, respectively.

Figure 10.3.13 2022 Phase III Drill Collar Locations at Maga and Siga Zones



Source: Orezone press release dated December 21, 2023. See that press release for the Figures 2 and 3 referred to here.

Table 10.3.7 Maga Deposit 2022 Drill Assay Highlights

Borehole	From m	To m	Length m[1,2]	Au g/t
BBD1217	198.00	240.00	42.00	1.11
incl.	219.00	229.00	10.00	1.72
BBD1218	189.00	194.10	5.10	0.91
and	248.00	253.00	5.00	0.99
and	260.00	277.20	17.20	1.38
BBD1219	166.00	186.00	20.00	3.30
BBD1220	139.00	143.00	4.00	1.84
and	150.00	171.00	21.00	5.35
BBD1241	111.00	122.00	11.00	1.75
and	143.20	152.00	8.80	0.86
and	193.00	211.00	18.00	1.05
BBD1246	124.00	131.20	7.20	8.75
incl.	130.00	131.20	1.20	49.73
BBD1247	194.00	212.00	18.00	3.10
and	256.00	260.00	4.00	3.17
BBD1248	156.00	160.00	4.00	2.48
and	184.00	195.75	11.75	2.22
BBD1249	31.00	39.50	8.50	3.73
incl.	31.00	32.50	1.50	18.07
and	129.00	140.00	11.00	3.89
incl.	137.20	140.00	2.80	13.44

[1]The mineralized intervals are based on a lower cut-off grade of 0.45 g/t Au, a minimal width of 1.5 m, and up to a maximum of 3.0 m of dilution being included between samples above the lower cut-off grade.

[2]True widths for Maga Deposit area drilling are approximately 85% of drilled widths.

Table 10.3.8 Siga Deposit 2022 Drill Assay Highlights

Borehole	From m	To m	Length m[1,2]	Au g/t
BBD1145	67.75	74.00	6.25	3.46
incl.	70.00	71.00	1.00	16.66
and	156.10	169.00	12.90	1.40
incl.	168.00	169.00	1.00	9.71
BBD1148	9.10	27.00	17.90	0.85
and	92.00	96.00	4.00	4.36

Borehole	From m	To m	Length m[1,2]	Au g/t
incl.	92.00	93.00	1.00	16.05
BBD1150	1.50	7.50	6.00	3.09
incl.	4.70	5.80	1.10	14.94
and	169.00	185.00	16.00	1.40
incl.	169.00	170.00	1.00	5.73
and incl.	184.00	185.00	1.00	9.74
BBD1157	56.10	59.50	3.40	2.94
incl.	58.70	59.50	0.80	11.63
and	68.00	82.80	14.90	0.88
incl.	78.00	81.90	3.90	1.87
and	87.00	91.00	4.00	5.82
incl.	89.00	90.00	1.00	19.79
BBD1158	78.00	82.00	4.00	3.52
incl.	81.00	82.00	1.00	12.43
and	137.60	156.00	18.40	0.62
incl.	153.00	156.00	3.00	1.07
BBD1161	21.00	33.00	12.00	1.50
incl.	31.00	33.00	2.00	5.12
and incl.	41.00	41.85	0.85	6.02
and	77.80	93.20	15.40	0.76
BBD1163	59.00	84.00	25.00	0.89
incl.	71.00	84.00	13.00	1.25
BBD1169	275.00	300.00	25.00	1.04
incl.	285.00	293.00	8.00	1.78
BBD1174	12.00	16.00	4.00	5.32
incl.	14.60	16.00	1.40	14.50
and	179.65	224.00	44.35	0.53
incl.	194.00	224.00	30.00	0.57
and	231.00	234.00	3.00	0.97
	239.00	246.10	7.10	0.71
BBD1180	73.00	77.00	4.00	4.16
incl.	73.00	74.00	1.00	10.16
and	174.50	191.75	17.25	0.97
and	213.00	245.00	32.00	0.79
BBD1186	1.20	9.00	7.80	1.24
and	205.00	234.00	29.00	1.11

Borehole	From m	To m	Length $m^{1,2}$	Au g/t
incl.	227.70	228.70	1.00	10.97
BBD1191	203.00	231.75	28.75	1.01
incl.	219.00	231.75	12.75	1.55
BBD1195	221.15	223.15	2.00	7.60
incl.	221.15	222.15	1.00	14.54
and	255.00	258.00	3.00	3.87
incl.	255.00	257.00	2.00	5.54
and	265.00	279.70	16.30	1.61
BBD1196	283.00	297.00	14.00	1.07
incl.	284.00	292.00	8.00	1.49
BBD1198	213.85	230.00	16.15	0.61
and	248.00	267.20	19.20	1.19
incl.	253.00	263.60	10.60	1.67
BBD1199	203.80	246.25	42.45	1.08
incl.	224.00	238.00	14.00	1.80
BBD1200	228.80	249.00	20.20	1.33
incl.	237.00	245.00	8.00	2.20
and	257.30	262.90	5.60	1.97

[1]The mineralized intervals are based on a lower cut-off grade of 0.45 g/t Au, a minimal width of 1.5 m, and up to a maximum of 3.0 m of dilution being included between samples above the lower cut-off grade.

[2]True widths for Siga Deposit area drilling are approximately 90% of drilled widths.

10.3.9 2023 Phase III Drilling at P8P9 and Maga Deposits

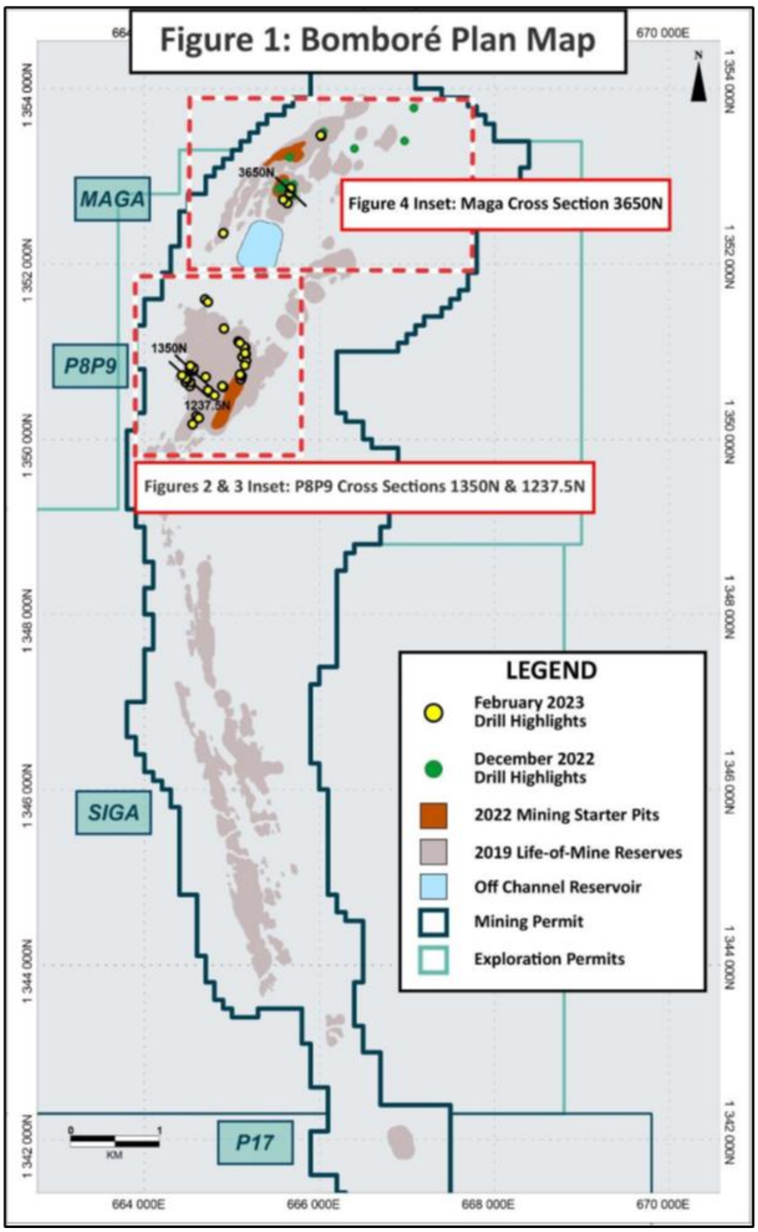
The Phase III Mineral Resource definition drill program was completed in early 2023. The focus of Phase III was expansion of the P17 Deposit and conversion of Inferred into Measured & Indicated Mineral Resources ahead of an upcoming sulphide expansion study.

Delineation drilling at the P8P9 Deposit revealed a higher-grade zone of plunging mineralization. This zone has been traced over 150 m along strike and remains open at depth and towards surface. The higher-grade zone is similar to that observed in the high-grade P17 Zone, 8 km to the south, as it is associated with and (or) proximal to, a granodiorite intrusion. Several intercepts at P8P9 returned thick well-mineralized intervals. The Phase III drilling at P8P9 was also successful in further demonstrating the continuity of mineralization below the current shallow Mineral Reserve pit, where the mineralization was locally traced to a depth of approximately 250 m below surface.

Additional drilling at the Maga Deposit continued to return high-grade intercepts in the sulphide zone, further demonstrating down-dip continuity. These results follow the more extensive set of Maga drill results from the drilling in 2022.

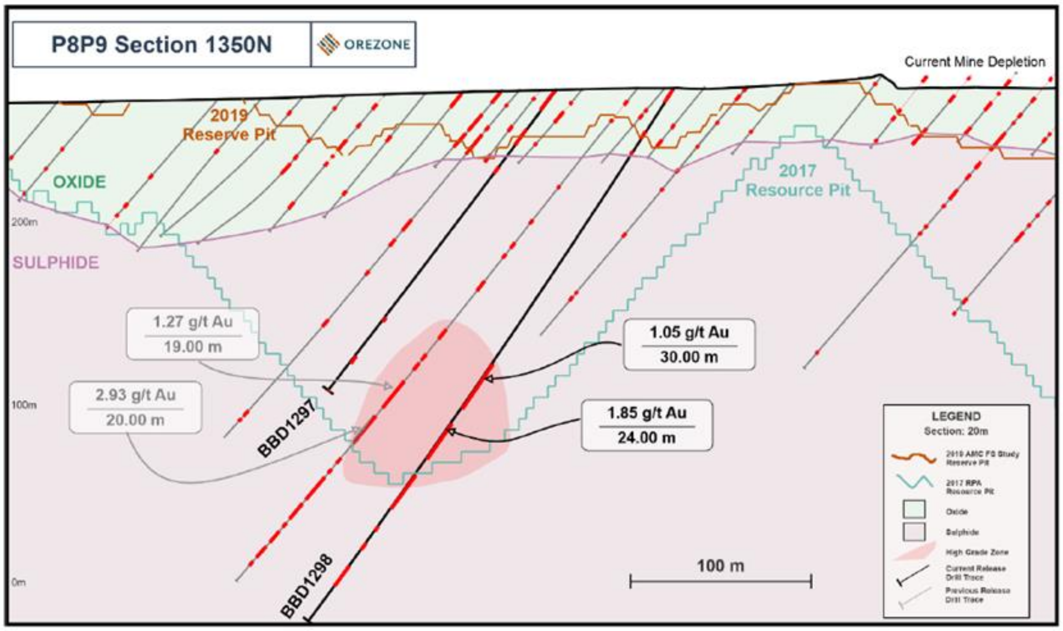
Drill hole collar locations for P8P9 and Maga are shown in Figures 10.3.14 to 10.3.16. Highlight drill assay results are listed in Tables 10.3.9 and 10.3.10.

Figure 10.3.14 Bomboré Plan Map Overview of 2023 Drill Hole Collar Locations



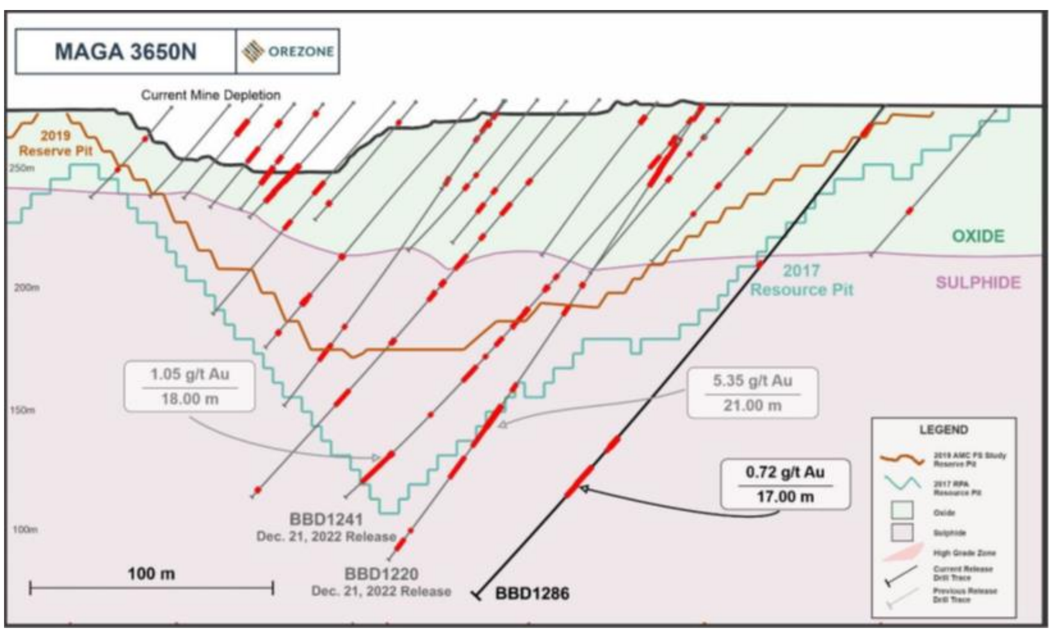
Source: Orezone press release dated March 6, 2023

Figure 10.3.15 P8P9 Cross Sectional Projection 1350N



Source: Orezone press release dated March 6, 2023

Figure 10.3.16 Maga Deposit Cross Sectional Projection 3650N



Source: Orezone press release dated March 6, 2023

Table 10.3.9 P8P9 Deposit 2023 Drill Results

Borehole	From m	To m	Length m[1,2]	Au g/t
BBD1288	83.00	104.00	21.00	1.16
and	109.60	141.00	31.40	0.80
incl.	138.00	140.00	2.00	3.18
and	223.40	233.00	9.60	1.28
BBD1289	170.50	180.80	10.30	2.08
and	191.00	193.00	2.00	6.17
BBD1293	63.10	84.00	20.90	0.90
incl.	72.00	73.00	1.00	6.18
and	166.00	180.40	14.40	1.37
BBD1294	17.40	51.00	33.60	0.98
incl.	20.20	40.00	19.80	1.31
and	99.00	139.40	40.40	1.72
incl.	126.75	139.40	12.65	3.24
and	144.60	145.60	1.00	12.81
and	157.00	159.00	2.00	14.23
BBD1298	188.00	212.00	24.00	0.94
incl.	188.00	189.00	1.00	9.38
and	222.00	246.00	24.00	1.85
incl.	225.00	236.00	11.00	3.04
and	257.00	269.00	12.00	0.62
and	273.00	280.00	7.00	0.69
BBD1305	39.30	93.00	53.70	1.49
incl.	57.00	70.00	13.00	2.01
BBD1309	31.80	43.50	11.70	1.98
incl.	39.00	43.50	4.50	4.59
BBD1312	260.00	304.00	44.00	1.06
incl.	271.00	279.00	8.00	2.75
and	345.65	349.85	4.20	3.01

[1]The mineralized intervals are based on a lower cut-off grade of 0.45 g/t Au, a minimal width of 1.5 m, and up to a maximum of 3.0 m of dilution being included between samples above the lower cut-off grade.

[2]True widths in the P8P9 Deposit area drilling are approximately 70 to 85% of drilled lengths.

Table 10.3.10 Maga Deposit 2023 Drill Results

Borehole	From m	To m	Length m[1,2]	Au g/t
BBD1283	208.00	210.00	2.00	9.78
and	303.00	305.00	2.00	10.83
BBD1285	197.00	215.00	18.00	3.14
BBD1287	213.00	217.00	4.00	3.01
and	221.00	237.00	16.00	5.27
incl.	226.00	227.00	1.00	62.21

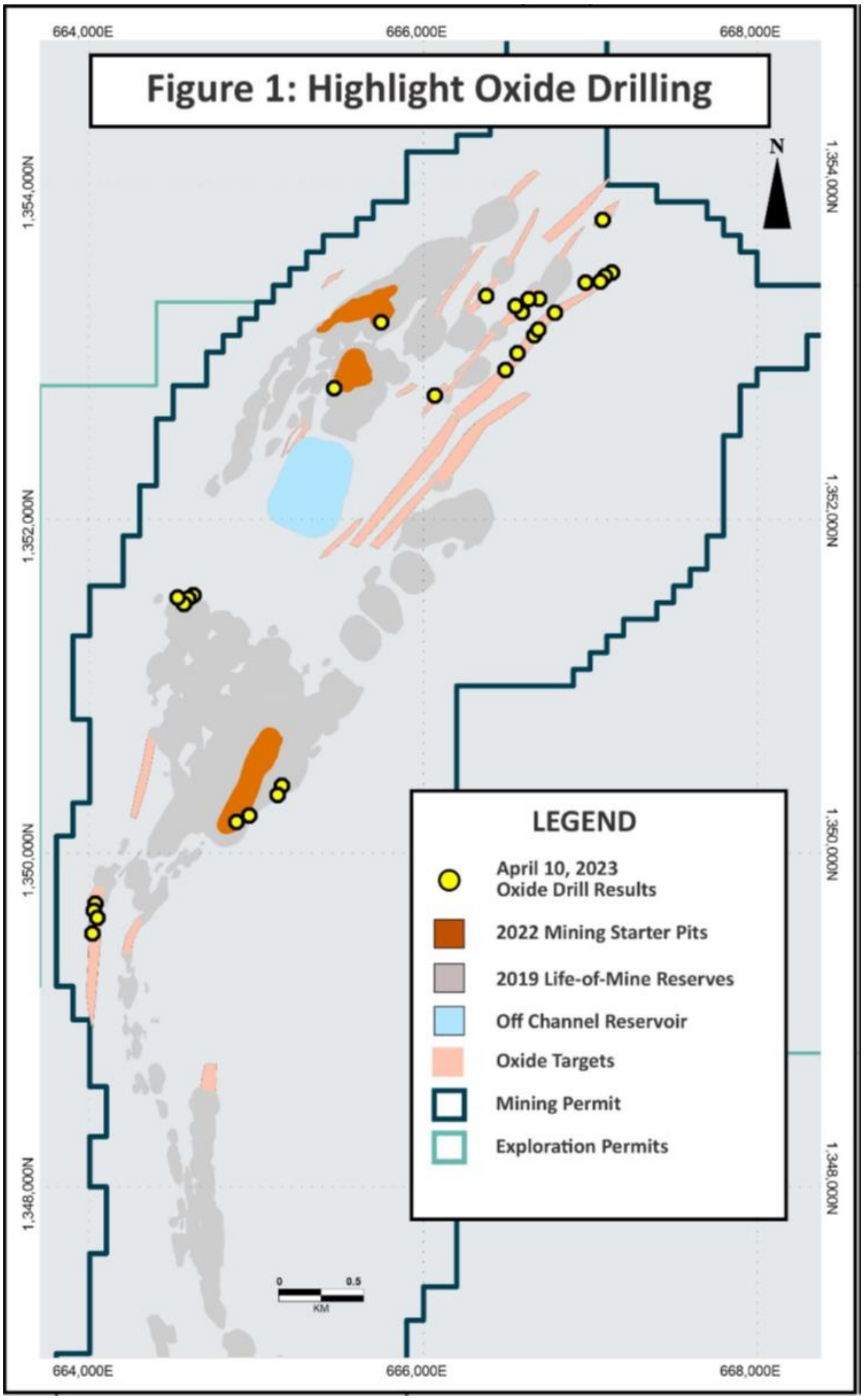
[1]The mineralized intervals are based on a lower cut-off grade of 0.45 g/t Au,a minimal width of 1.5 m, and up to a maximum of 3.0 m of dilution being included between samples above the lower cut-off grade.

[2]True widths in the Maga Deposit area drilling are approximately 85% of drilled lengths.

10.3.10 2022 Oxide Drilling Program

Drilling during the 2022 program included Phase III Oxide Mineral Resource definition program and testing of oxide targets outside of the current Mineral Resources along know mineralized trends. Drill hole collar locations are shown in Figure 10.3.17. Highlight oxide drill assay results are listed in Table10.3.11.

Figure 10.3.17 Bomboré Oxide Collar Locations



Source: Orezone press release dated April 10, 2023

Table 10.3.11 2023 Oxide Drill Result Highlights

Prospect / Area	Borehole	From m	To m	Length m[1,2]	Au g/t
Maga Main	BBC5263	63.00	74.00	11.00	1.26
	incl.	70.00	73.00	3.00	2.99
Maga Hill	BBC5272	31.00	40.00	9.00	1.67
	incl.	34.00	39.00	5.00	2.58
Maga HW	BBC5078	11.00	22.00	11.00	2.05
	incl.	11.00	12.00	1.00	10.63
	BBC5174	32.00	56.00	24.00	1.01
	incl.	33.00	41.00	8.00	1.76
	BBC5287	19.00	24.00	5.00	30.86
P8P9	BBC5392	11.00	36.00	25.00	1.42
	incl.	19.00	25.00	6.00	3.15
	and	48.00	53.00	5.00	2.18
	BBC5336	0.00	16.00	16.00	1.56
	incl.	13.00	14.00	1.00	14.42
	BBC5338	20.00	40.00	20.00	1.31
	BBD1279	46.25	48.00	1.75	42.19
Siga E	BBC5041	41.00	50.00	9.00	2.00
	incl.	45.00	46.00	1.00	12.82
Siga W	BBC5044	14.00	24.00	10.00	1.69

[1]The mineralized intervals are based on a lower cut-off grade of 0.28 g/t Au, a minimal width of 1.5 m, and up to a maximum of 3.0 m of dilution being included between samples above the lower cut-off grade.

[2]True widths for all oxide drill zones are approximately 75 to 80% of drilled lengths.

The drill results successfully confirmed the continuity of near-surface oxide mineralization within the established Mineral Resource area, demonstrated potential to further expand the oxide Mineral Resources along strike, and highlighted the potential to delineate additional oxide Mineral Resources, in order to replace mine depletion. These targets will be further tested and defined in ongoing drilling in 2023.

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

The following Technical Report Section considers Orezone borehole data from 2003 through 2023. No records are available that detail the sample preparation, analyses, and security protocol undertaken by Channel Mining (Barbados) Company Ltd., throughout their earlier exploration campaigns from 1994 to 2000. Consequently, the Author has not reviewed data prior to that collected by Orezone in 2003.

11.1 Orezone Sampling 2003 – 2023

Samples were collected at the drill rig cyclone at 1-m intervals during the 2003 RC drilling program. Each sample was logged, then split with a riffle splitter and recombined in a 2-m composite sample that was submitted to Abilab Afrique de l'Ouest s.a.r.l. in Bamako, Mali.

From 2005 to 2023, each RC sample is given a unique incremental identification number, rather than setting aside specific series of numbers for different projects or sample type, thereby reducing the risk of error. The number sequence incorporates pre-determined numbers assigned to the quality control samples.

The RC samples are systematically collected every metre for the entire length of the boreholes. The material from the cyclone underflow is placed in polypropylene bags and labelled with borehole and identification numbers.

From 2010 to 2019, an ESSA 020L rotary sample divider (RSD) has been used to generate a RC sub-sample with a mass of ±2.1 kg. The field duplicates are prepared at this stage and inserted into the sample stream. The following information is entered into the sample book and tabulated in computer files: date, borehole number, interval, sample number, sample weight, magnetic susceptibility readings, sampler name, and assay type requested. The RSD is cleaned between each sample with compressed air. The damp samples are dried in the sun. Sample rejects are kept until assay results are available. Reject samples are all saved in thick polypropylene bags, and when all the check assays and umpire assays have been completed, the samples grading >0.1 g/t Au are kept and other samples, that are below that threshold over multi-metre widths, are discarded. Coarse reject RC sample bags are safely stored in a shed on a concrete slab or elevated compacted laterite ground, in piles wrapped in thick plastic sheets tied down with ropes.

The sub-samples are transported in rice bags to the Orezone sample preparation facility warehouse in Ouagadougou for insertion of additional quality control samples (blanks and certified reference materials (CRM) prior to submission to the preparation laboratory.

Since 2021, RC rigs equipped with a 3-chute splitter have been used to generate directly RC sub-samples of ±2.1 kg in triplicate. The rest of the sampling and analytical protocols have remained the same, except that all in-house work that was done at the Ougadougou Kossodo facility has progressively been transferred to the new Bomboré site Exploration facility between September 2022 and March 2023.

Drill core samples are collected from half-drill core cut lengthwise with a diamond saw, at regular 1-m intervals for HQ drill core and 1.0 m or 1.5 m intervals for NQ drill core, although the sample length can be adjusted according to the geological contacts, the drill core recovery, or the Rock Quality Designation zones. Like RC samples, core description includes detailed information about geology, alteration, mineralization, and magnetic susceptibility readings taken for every sample, as well as structural and geotechnical readings and descriptions. Sampling protocols for core are generally similar to that for RC sampling, except that no field duplicate is collected as ½-drill core cut is kept as a witness sample.

All drilling sampling was conducted by appropriately qualified personnel under the direct supervision of appropriately qualified geologists.

Until June 2011, Orezone used two gold analytical procedures for drilling samples collected at the Bomboré Gold Project. Drill core assay samples from the fresh or non-weathered zone were assayed using a standard fire assay procedure on 50-g pulverized sub-samples. RC assay samples and core samples from the weathered (laterite, oxide and transition) zone were subjected to bottle-roll cyanide extraction with LeachWELL assaying. From July 2011, all new samples (RC and core) were subjected to bottle-roll cyanide extraction with LeachWELL assaying.

Orezone has utilized various analytical laboratories throughout the time they have been actively drilling at the Property, including:

- ABILAB Burkina s.a.r.l. (ALS), a subsidiary company of the ALS Group in Ouagadougou.
- SGS Burkina Faso SA (SO), a subsidiary company of the SGS Group in Ouagadougou.
- BIGS Global Burkina s.a.r.l. (BIGS), in Ouagadougou.
- ACTLABS Burkina Faso s.a.r.l. (ACT), a subsidiary company of the ACTLABS Group in Ouagadougou.
- SGS Burkina Faso SA (SGS-B), a subsidiary company of the SGS Group operating the Orezone Bomboré sample preparation facility.

ALS is not an accredited laboratory, but its hub laboratory in South Africa is accredited with ISO/IEC 17025:2005. SO is accredited in accordance to ISO/IEC 17025:2005. BIGS is not accredited. SGS-B is accredited in accordance to ISO 9001:2008. All laboratories are independent of Orezone, the Author and P&E.

A summary of Orezone’s sample preparation and analytical activities is presented in Tables 11.1.1 and 11.1.2.

Table 11.1.1 Summary of Sample Preparation Activities by Laboratory and Type of Sample for the Period from October 2007 to May 2023

Sample Type	ALS	SO	BIGS	ACT	SGS-B	Total
Primary Samples	109,668	208,910	106,184	8,674	176,993	610,429
Auger	4,209	3,843	2,065			10,117
Drill Core (definition)	18,248	99,681	40,727	6,993	52,685	218,334
Drill Core (geotechnical)		29	106		169	304
Metallurgy	6				190	196
RC (definition)	87,187	100,481	62,827	1,507	121,866	373,868
RC (geotechnical)		4,044			1,890	5,934
Rock (outcrop)	18	513	418	174	19	1,142
Trench (geotechnical)		319	41		174	534
Check Samples	4,043	6,294	344	124	2,057	12,862
Auger					16	16
Drill Core (definition)		47	3		69	119
RC (definition)	4,043	6,201	341	124	1,972	12,681
RC (geotechnical)		46				46
Bottle-roll Cyanidation Leach Residue Samples	506	492	158,276	0	0	159,274
Auger			269			269
Drill Core (definition)		30	49,283			49,313
Core (geotechnical)			5			5
Metallurgy	410		2,605			3,015
RC (definition)	60	462	105,361			105,883
RC (geotechnical)			701			701
Rock (outcrop)	36		8			44
Trench (geotechnical)			44			44
Umpire Samples	353	17	0	0	130	500
Drill Core (definition)						0
RC (definition)	353	5				358
RC (definition ≥ 5 g/t Au)		12			130	142
Total	**114,570**	**215,713**	**264,804**	**8,798**	**179,180**	**783,065**

ALS: ABILAB Burkina s.a.r.l., a subsidiary company of the ALS Group in Ouagadougou.

SO: SGS Burkina Faso SA, a subsidiary company of the SGS Group in Ouagadougou.

BIGS: BIGS Global Burkina s.a.r.l. in Ouagadougou.

ACT: ACTLABS Burkina Faso s.a.r.l., a subsidiary company of the ACTLABS Group in Ouagadougou.

SGS-B: SGS Burkina Faso SA, a subsidiary company of the SGS Group operating the Bomboré sample reparation facility.

Table 11.1.2 Summary of Analytical Activities by Laboratory and Type of Sample for the Period from October 2007 to May 2023

Sample Type	LeachWELL 1-kg			FA-AAS 50-g			FA-GRAV 50-g		
	ALS	SGS	BIGS	ALS	BIGS	SGS	ALS	BIGS	SGS
Primary Samples Sub-Total	**571**	**24,024**	**585,555**	**14,683**	**570**	**12,490**	**0**	**0**	**0**
Auger			10,444						
Drill Core (definition)		2	203,809	14,360	570	12,224			
Drill Core (geotechnical)			321						
Metallurgy			2,706	228		144			
RC (definition)	571	24,022	360,790						
RC (geotechnical)			6,098						
Rock (outcrop)			828	95		122			
Trench (geotechnical)			559						
Check Samples Sub-Total	**110**	**1,583**	**14,942**	**962**	**258**	**1,426**	**0**	**0**	**0**
Auger			16						
Drill Core (definition)		15	1,203	832	168	1,336			
RC (definition)	110	1,568	13,663	130	90	90			
RC (geotechnical)			48						
Trench (geotechnical)			12						
Bottle-roll Cyanidation Leach Residue Samples Sub-Total	**0**	**2**	**0**	**44,188**	**18,161**	**101,323**	**0**	**0**	**0**
Auger				301		812			
Drill Core (definition)				5,175	10,260	35,815			
Core (geotechnical)					4	7			
Metallurgy				1,110		995			
RC (definition)		2		37,595	7,887	62,959			

Sample Type	LeachWELL 1-kg			FA-AAS 50-g			FA-GRAV 50-g		
	ALS	SGS	BIGS	ALS	BIGS	SGS	ALS	BIGS	SGS
RC (geotechnical)						657			
Rock (outcrop)				7		40			
Trench (geotechnical)					10	38			
Umpire Samples Sub-Total	**0**	**9,657**	**0**	**0**	**118**	**331**	**1,633**	**369**	**438**
Drill Core (definition ≥ 5 g/t Au)							682	290	148
Drill Core (definition)		3,052			118	331			
RC (definition ≥ 5 g/t Au)							947	79	284
RC (definition)		6,491							
RC (geotechnical ≥ 5 g/t Au)							4		6
RC (geotechnical)		114							
Total	**681**	**35,266**	**600,497**	**59,833**	**19,107**	**115,570**	**1,633**	**369**	**438**

LeachWELL 1-kg Bottle-roll cyanidation with LeachWELL on a 1-kg sample over ten hours.

ALS: ABILAB Burkina s.a.r.l., a subsidiary company of the ALS Group in Ouagadougou.

SO: SGS Burkina Faso SA, a subsidiary company of the SGS Group in Ouagadougou.

BIGS: BIGS Global Burkina s.a.r.l. in Ouagadougou.

Orezone applied strict security measures throughout the sampling, sample preparation, and analytical stages. The RC samples and the drill core retrieved by the drillers are collected and handled at the drill site by Orezone personnel. The sample bags are transported by a dedicated driver to a secured storage area in the Bomboré Gold Project area, split and sent to the Orezone warehouse in Ouagadougou. The sample bags are never left unattended. The sample storage area at the Bomboré Gold Project is fenced and a watchman provides full time security. The warehouse in Ouagadougou is an enclosed building. From there, the samples are checked, sent to the preparation facilities in Ouagadougou and thereafter returned to Orezone's warehouse. Finally, the samples are dispatched to the analytical laboratories. The samples are continually under the direct control of Orezone staff, who monitor the preparation and shipment of the samples. This procedure ensures reasonable chain of custody by Orezone from the drill sites to the analytical laboratory.

11.1.1 RC Drilling Sampling

The ±2.1 kg RC sub-samples are shipped from the Project site to Orezone’s sample preparation facility warehouse in Ouagadougou with shipments monitored by the Project geologist.

The information provided includes the Project number, sample type, weight, and interval.

The form is sent by e-mail and a hard copy. The sample preparation and analyses procedures are outlined as flow sheets in Figures 11.1.1 to 11.1.4.

At Orezone’s warehouse (Kossodo), samples are checked against the shipment form and inspected before adding the in-house blanks and CRMs, and then forwarded to the various laboratories for preparation.

Between September 2022 and March 2023, the Kossodo activities were progressively transferred to a new Bomboré site facility. As a result of this relocation, samples are now collected at site and then returned to site by representatives of the sample preparation laboratory. Samples are also collected at site by representatives of the analytical laboratory, which also must return to site the leach residue samples that have been selected for leach residue assays.

The sample preparation procedure is essentially similar. After drying, the entire ±2.1 kg sample is crushed to 6 mm and pulverized in a vertical continuous Keegor disc pulverizer or LM2 shatter box to achieve 85% passing 106 µm. The pulverized sample is then returned to the Orezone facility.

At the Orezone facility, pulverized samples are rotary split using a Rocklab benchtop rotary divider to 1-kg for use as original samples, blind duplicates, and lab-aware duplicate samples. The blind duplicate control samples are also inserted into the sampling stream. The remaining portion is stored for future reference. All prepared samples are then dispatched to the various laboratories for analyses using submittal forms indicating the batch and samples numbers, the number of samples and bags.

Figure 11.1.1 Field Sampling Procedure and Kossodo Sample Reception and Preparation Procedure

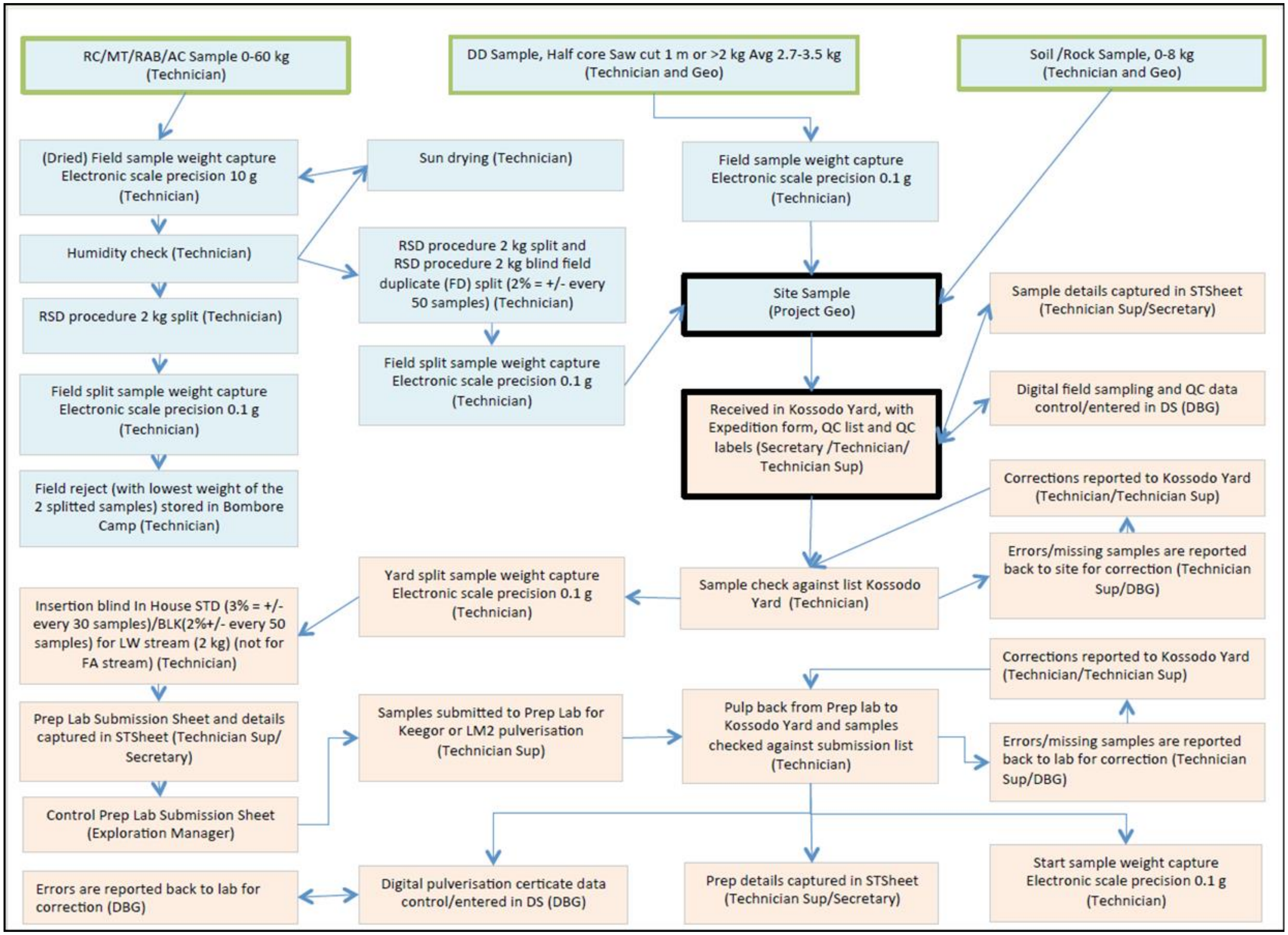


Figure 11.1.2 Leachwell Stream Rotary Sample Division and QA/QC Procedure

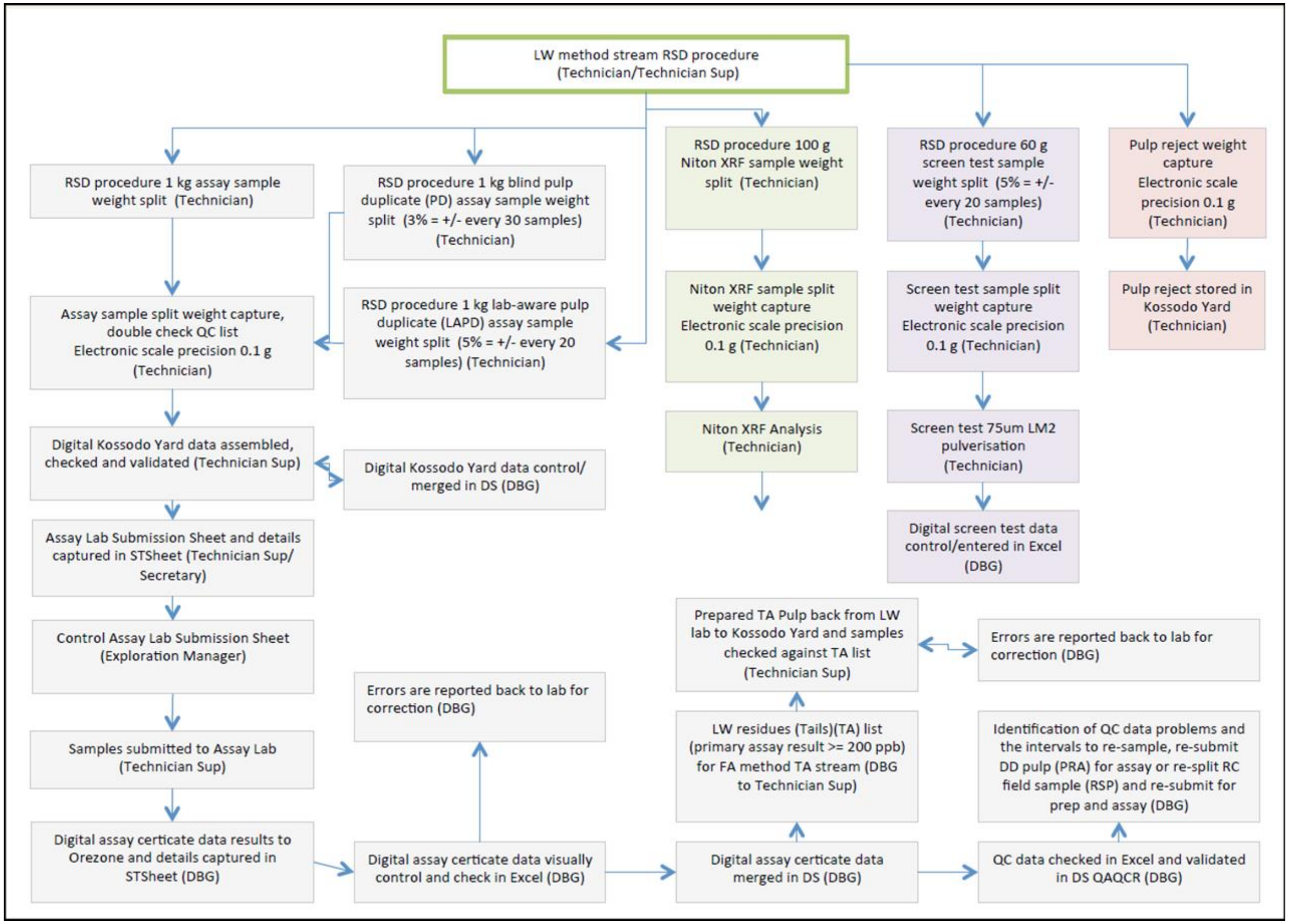


Figure 11.1.3 FA Stream Sample Division and QA/QC Procedure

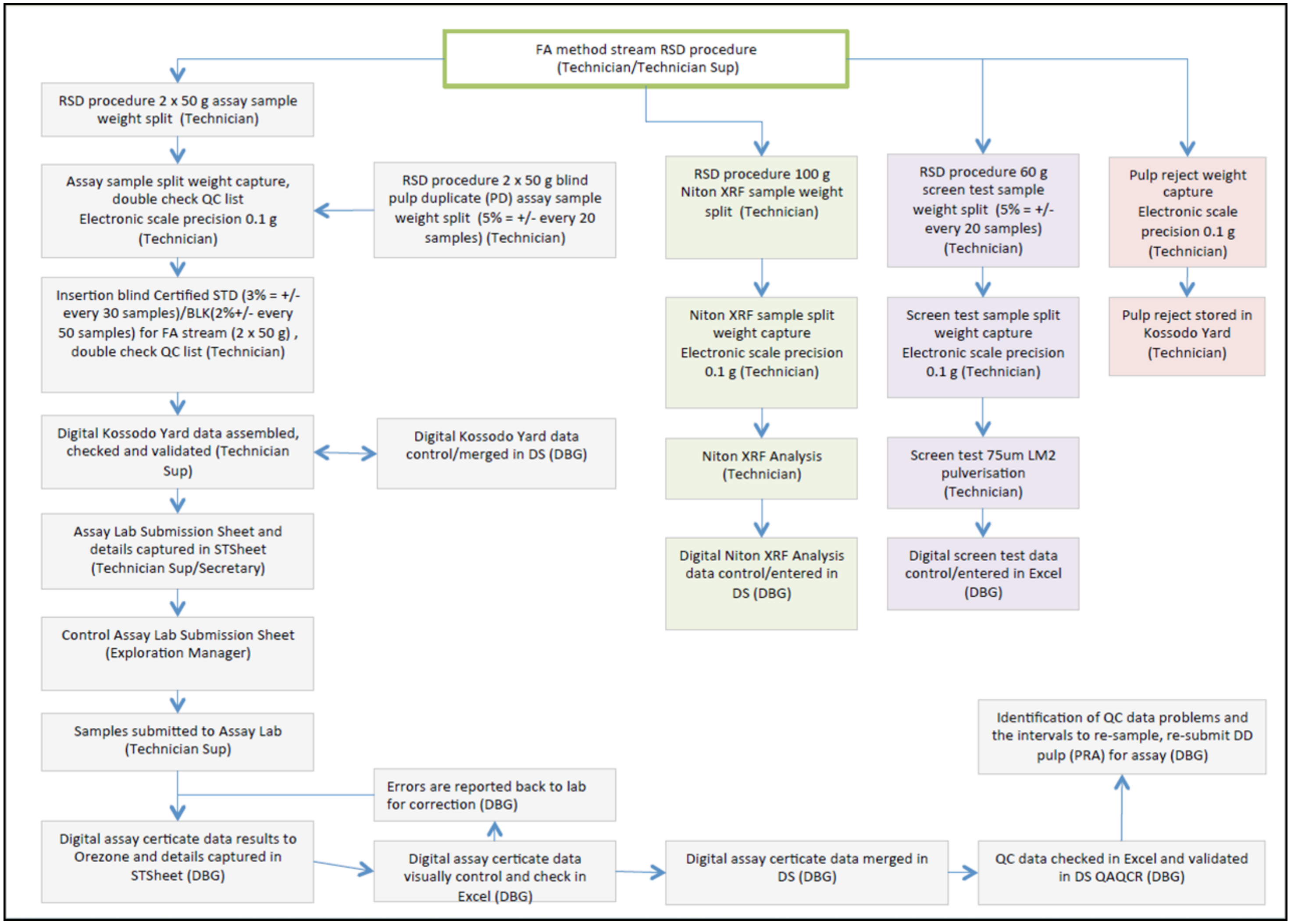
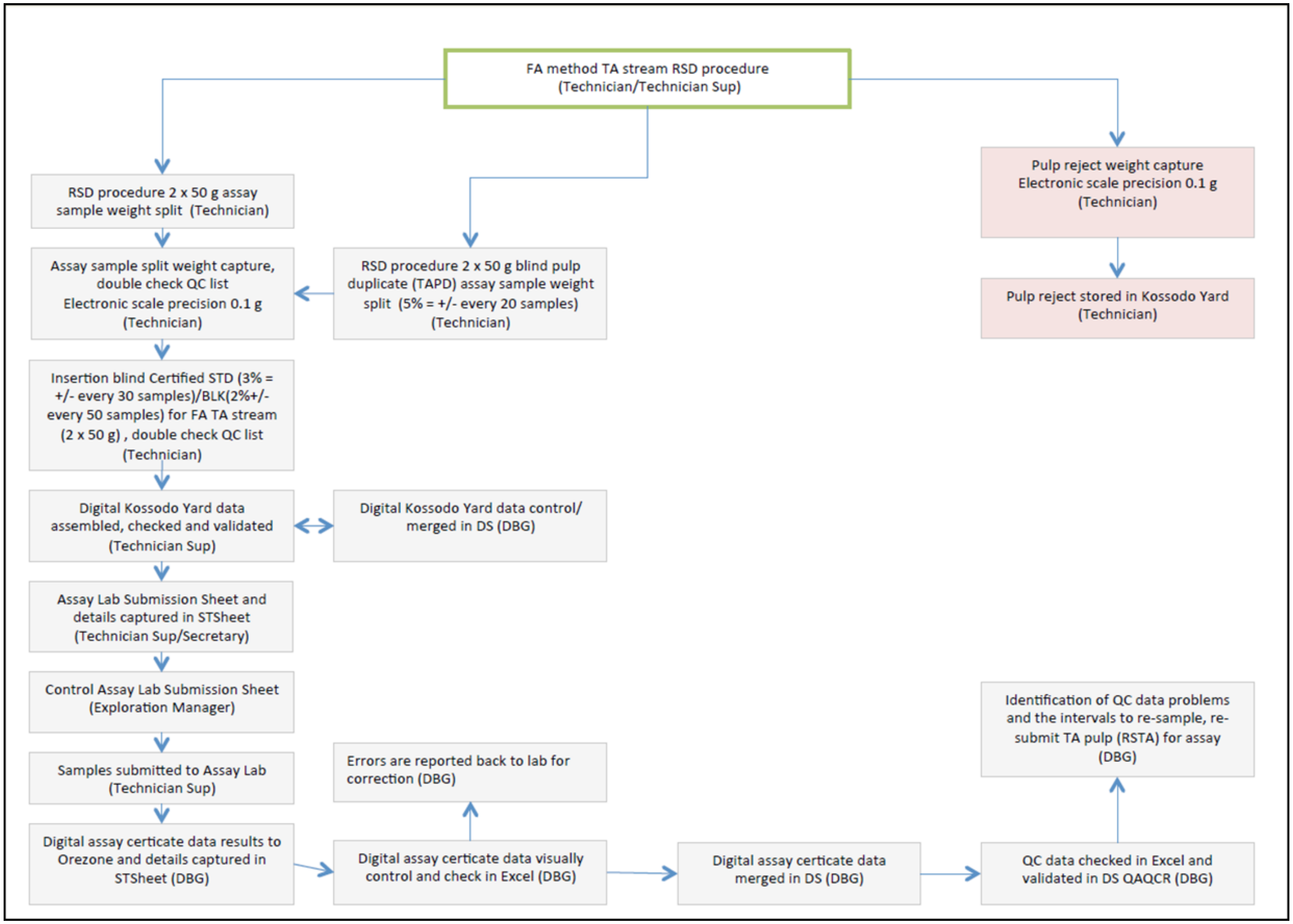


Figure 11.1.4 Leach Residue Rotary Sample Division and QA/QC Procedure



Several steps are taken in order to prevent sample-to-sample contamination at the preparation stage. The sample bags are emptied into metal pans that have been blown clean with compressed air. The samples are recovered at the underflow of the pulverizer in a clean pan lined with a single-use thin plastic bag, at Orezone's request. The saved fraction will be left in this bag inserted into a new, thick plastic bag. The apparatus is cleaned by feeding it barren material and by using vacuum and compressed air.

Prepared 1-kg RC samples, analyzed by bottle roll cyanide leach method, are poured into a bottle with water, leaching solution and LeachWELL tablets, and are then placed on a roller for 10 to 12 hours. LeachWELL is a catalyst formulated to increase the dissolution rate of gold in the samples. The gold is dissolved through formation of its cyanide complex, which can be concentrated through the process of solvent extraction. Gold content in the solution (liquor) is determined using atomic absorption analysis. For all the samples having liquor grade >0.5 g/t (from 2007 to 2009) or >0.2 g/t Au (from 2010 to 2023), the tail is washed, dried and a 50-g charge is split and submitted for assaying using a conventional fire assay procedure on 50-g sub-sample.

The laboratories provide the assay results in electronic text files. The original files are imported by Orezone's database geologists in a Datashed database, where assay results are automatically merged with the drill logs. Analytical certificates are validated according to their Quality Assurance/Quality Control (QA/QC) score. Samples from certificates or portions of certificates that have passed the Orezone QA/QC test are assigned a Priority = 1 code in the Datashed assay database, whereas those that failed are assigned a Priority = 0 code. Only Priority 1 samples are exported for use in Mineral Resource modelling.

11.1.2 Core Drilling Sampling

Drill core samples were prepared for assaying (drying, crushing and pulverization) by ALS, SGS, BIGS and Actlabs laboratories in Ouagadougou, as well as the Orezone sample preparation facility operated by SGS at Bomboré from June 2012 until June 2022. Prepared samples were returned to the Orezone facility for the insertion of control samples before their submission to the analytical laboratory.

Following a pre-programme round robin testing of the various Ouagadougou laboratories and ongoing monitoring of the quality during the programme, Orezone chose to forward initially most of the core samples to the ALS Ouagadougou laboratory, but then progressively switched to the SGS laboratory for fire assay (FA) analysis on 50-g charges with an atomic absorption spectroscopy finish. From January 2022, BIGS laboratory became the primary laboratory for the FA analyses.

11.2 Bulk Density Data

The bulk density database includes 131,229 records generated by Channel and Orezone from measurements on core from 1,454 boreholes.

Bulk density measurements were conducted onsite using the water displacement method. Generally, a single piece of drill core, 10 cm to 15 cm in length, is selected in each core box prior to splitting. Wax or thin film coating is applied to the sample whenever necessary, and it is subsequently weighed in air and in water. Bulk density was calculated and subsequently classified by rock and material type.

Samples that are not fully dried prior to the water displacement test and the moisture present in the drill core samples could potentially bias the bulk density measurement, particularly in the oxide and transition zones. Orezone have estimated this moisture content by weighing samples at the sample preparation laboratory before and after drying. The average loss of moisture in the oxide drill core samples is 5.7%, in the transition drill core samples 2.8%, and in the fresh drill core samples 0.2%. A reduction factor can therefore be applied to individual samples in the oxide and transition zones to compensate for potential bias.

Orezone has recognized that bulk density increases with depth through the weathering profile and is then fairly homogeneous within the fresh zone for each lithologic unit.

11.3 Quality Assurance / Quality Control

This Technical Report reviews the analytical quality control measures implemented by Orezone between 2007 and 2023.

Detailed reviews of analytical QA/QC measures implemented since December 2012 (and March 2013 for the P16 Deposit) are provided in (Maes, October 2014) and (Maes, February 2015). Detailed reviews of analytical QA/QC measures implemented prior to December 2012 (and prior to March 2013 for the P16 Deposit) are provided by Met-Chem (Buro and Saucier, 2008) and SRK (Cole and El-Rassi, 2008; Cole and El- Rassi, 2010; Cole et al., 2012; Gourde, 2014; Defilippi et al., 2015).

Historical sampling, analytical, and QA/QC protocols used on the Property are summarized in Tables 11.3.1 and 11.3.2.

In all instances, the percentage is based on the total stream of samples submitted to the analytical laboratory i.e., the primary samples plus the FD, CD, PD, BLK, and STD, but excluding the LAPD.

Table 11.3.1 Summary of the Sampling, Analytical and QA/QC Protocols Used on the RC Programs Since 1994

Period	Sampling m	Division	QA/QC Samples					Analyses Method / Weight
			FD	LAPD	PD	BLK	CRM	
1994 to 2000	19,501	Riffle?	No	No	No	No	No	FA 30-g
2003	1,387	Riffle	1%	0%	0%	No	0,2%	FA 50-g
2005	13,829	Riffle	2%	8%	4%	2% NC	2% NC	BLEG 2-kg
2006 to 2007	8,770	Riffle	2%	9%	4%	2% NC	2% NC	LW 2-kg
2008	19,663	Riffle	2%	10%	4%	2% IHC	2% IHC	LW 1-kg
2010	42,456	RSD	4%	10%	4%	3% IHC	4% IHC	LW 1-kg
2011 to 2014	197,835	RSD	2%	5%	3%	2% IHC	3% IHC	LW 1-kg
2016 to 2023	90,881	RSD	2%	5%	3%	2% IHC	3% IHC	LW 1-kg

RSD: Rotary Sample Divider

FD: Field Duplicate, blind to the preparation laboratory

PD: Pulp Duplicate, blind to the analytical laboratory

LAPD: Lab-Aware Pulp Duplicate, known to the analytical laboratory

BLK: Blank, blind to the preparation laboratory

CRM: CRM; blind to the preparation laboratory

NC: Non-certified reference material

IHC: In-house referenced material

FA: Fire Assay, with AAS finish

BLEG: Bulk Leach Extractable Gold

LW: LeachWELL™

Table 11.3.2 Summary of the Sampling, Analytical and QA/QC Protocols Used on the Core Drilling Programs Since 1998

Period	Sampling (m)	Division	QA/QC Samples					Analyses Method / Weight
			CD	LAPD	PD	BLK	CRM	
1998	1,080	Saw	No	No	No	No	No	FA 50-g
2007 to2008	5,714	Saw	2%	No	4%	2% CM	2% CM	FA 50-g
2009	7,738	Saw; RSD	2%-0%	9-10%	4%	2%-3% IHC	2%-6% IHC	LW 1kg; FA 50-g
2010 to 2011	71,097	Saw; RSD	No	5%	3%-5%	2% IHC	3% IHC	LW 1kg; FA 50-g
2012 to 2014	71,520	Saw; RSD	No	5%	3%	2% IHC	3% IHC	LW 1-kg
2016 to 2023	75,874	Saw; RSD	No	5%	5%	2% IHC	3% IHC	LW 1-kg

RSD: Rotary Sample Divider

CD: Crush Duplicate, blind to the analytical laboratory

PD: Pulp Duplicate, blind to the analytical laboratory

LAPD: Lab-Aware Pulp Duplicate, known to the analytical laboratory

BLK: Blank, blind to the preparation laboratory

CRM: CRM; blind to the preparation laboratory

CM: Certified Reference Material

NC: Non-certified reference material

IHC: In-house reference material

FA: Fire Assay, with AAS finish

BLEG: Bulk Leach Extractable Gold

LW: LeachWELL™

Orezone has partially relied on the internal analytical QA/QC measures implemented by ALS, BIGS and SGS during past drilling campaigns. In addition, Orezone implemented external analytical control measures on all reverse circulation, diamond borehole and auger sampling consisting of using control samples, and duplicate sampling in all sample batches submitted for assaying.

Commercial CRMs and in-house certified material (ICM), and blanks were used on drill core samples and RC tail samples analyzed by FA by BIGS, SGS and ALS. In-house blanks and ICMs were used on RC and drill core samples analyzed by LeachWELL by BIGS and SGS. Field duplicates were used on RC samples analyzed by LeachWELL by BIGS and SGS. Pulp duplicates were used on all sampling including tails and were run by all laboratories. Lab-aware pulp duplicates were used on RC and drill core samples analyzed by LeachWELL by BIGS.

The type and location of the control samples in the sample stream has been determined on the basis of randomly generated numbers.

Orezone also undertook check assaying on RC and drill core samples analyzed by LeachWELL by BIGS and on drill core samples analyzed by FA by ALS, SGS or BIGS, at secondary umpire laboratories, including SGS, BIGS and ALS (Table 11.1.2).

11.3.1 LeachWELL Assaying

Since October 2007, Orezone introduced a procedure of internal certification for the reference material inserted in the stream of 1-kg samples analyzed by the LeachWELL method.

The ICM is made of barren saprolite 'spiked' with CRM.

This method allows the insertion of blind QA/QC samples in the stream of samples and the monitoring of the accuracy of the analytical results; it was possible and economically affordable to implement this new procedure given the presence of three reliable commercial analytical laboratories in Ouagadougou from 2007.

Orezone has been using two different sources of barren oxidized material for the preparation of the ICM; saprolite of coarse-grained granite and lateritized coarse-grained granite. The material is collected in a quarry in batches of approximately 100 kg, crushed, dried, and split into 2-kg bags. Five out of 50 2-kg samples are submitted to BIGS in Ouagadougou for sample preparation and LeachWELL analysis of the gold content and, to be accepted as a blank batch, all five samples must return a gold analysis less than or equal to the detection limit (i.e., ≤1 ppb Au).

When a batch of blank material is accepted, it can be used as a blank in the stream of samples to be prepared, or a blank base to be spiked, with CRM. Using various certified materials and various proportions of barren blank and CRM, Orezone can create reference material with a theoretical gold grade within the range anticipated for the samples of a given exploration project. Orezone has focused on a range of grades between 0.2 g/t and 1.5 g/t Au.

The list of CRM used for the preparation of the ICMs since 2007 is presented in Table 11.3.3.

The ICM used are listed in Table 11.3.5.

Table 11.3.3 Specifications of Control Samples Used to Produce ICM Used for LeachWELL Analysis by Orezone from October 2027 to May 2023

Certified Reference Material	Source	In-House Reference Material	Certified Grade Au g/t	
			Mean	Standard Deviation
Amis23	Amiso	AA23*	3.57	0.13
Amis43	Amiso	AA43*	1.65	0.085
BLKORZ	Orezone	BLKORZ	<0.001	--
HiSilK2	Rocklabs Ltd	ARK2*	3.474	0.017
HISILK4	Rocklabs Ltd	ARK4*	3.463	0.013
HISILK6	Rocklabs Ltd	HISILK6*	3.446	0.0165
HiSilP1	Rocklabs Ltd	ARP1*	12.05	0.065
HISILP3	Rocklabs Ltd	ARP3*	12.24	0.0355
HISILP5	Rocklabs Ltd	ARP5* or HISILP5*	12.051	0.032
OREAS 15h	Analytical Solutions Ltd	AO16h*	1.019	0.025
OREAS 15Pb	Analytical Solutions Ltd	AO15PB*	1.06	0.008
OREAS 60b	Analytical Solutions Ltd	AO60b*	2.57	0.023
OREAS 62c	Analytical Solutions Ltd	AO62c*	8.79	0.21
OREAS 67a	Analytical Solutions Ltd	AO67a*	2.24	0.096
OxA89	Rocklabs Ltd	AR89*	0.0836	0.00125
OxC109	Rocklabs Ltd	AR109*	0.201	0.001
OxE101	Rocklabs Ltd	AR101*	0.607	0.0025
OxE106	Rocklabs Ltd	AR106*	0.606	0.002
OxJ47	Rocklabs Ltd	AR47*	2.384	0.01
OxK48	Rocklabs Ltd	AR48*	3.557	0.0095
OxL159	Rocklabs Ltd	OxL159*	5.849	0.021
OxN173	Rocklabs Ltd	OxN173*	7.668	0.046
OxP172	Rocklabs Ltd	OxP172*	15.057	0.0575
OxQ170	Rocklabs Ltd	OxQ170*	24.939	0.091
OxQ75	Rocklabs Ltd	AR75*	50.03	0.19
SG56	Rocklabs Ltd	AR56*	1.027	0.0055
SG66	Rocklabs Ltd	AR66*	1.086	0.0045

Certified Reference Material	Source	In-House Reference Material	Certified Grade Au g/t	
			Mean	Standard Deviation
SH55	Rocklabs Ltd	AR55*	1.375	0.007
SL108	Rocklabs Ltd	AR108*	5.744	0.021
SL123	Rocklabs Ltd	SL123*	5.899	0.0195
SL34	Rocklabs Ltd	AR34*	5.893	0.0285
SL46	Rocklabs Ltd	AR46*	5.867	0.033
SL51	Rocklabs Ltd	AR51*	5.909	0.0235
SL61	Rocklabs Ltd	AR61*	5.931	0.0285
SL76	Rocklabs Ltd	AR76*	5.96	0.026
SN117	Rocklabs Ltd	SN117*	8.443	0.018
SN26	Rocklabs Ltd	AR26*	8.543	0.036
SN38	Rocklabs Ltd	AR38*	8.573	0.0305
SN50	Rocklabs Ltd	AR50*	8.685	0.031
SN91	Rocklabs Ltd	AR91*	8.679	0.028
SP116	Rocklabs Ltd	SP116*	18.091	0.0515
SP122	Rocklabs Ltd	SP122*	18.044	0.061
SQ36	Rocklabs Ltd	AR36*	30.04	0.12
SQ48	Rocklabs Ltd	ARS48*	30.25	0.085
SQ87	Rocklabs Ltd	ARS87*	30.87	0.105
SQ88	Rocklabs Ltd	SQ88*	39.72	0.14

Note: LeachWELL™

* ICM comprised of blank base with spiked CRM

Orezone detected an increase in ICM failures from late April 2022 on BIGS LeachWell jobs and, in response, all analytical work was halted to understand and resolve the issue. Two series of umpire samples were submitted from duplicates of the April 2022 failed jobs. One series was analyzed by FA with AAS finish, and a second series of higher-grade samples were analyzed by FA with gravimetric finish. The umpire assaying has confirmed that there were no analytical issues with the original BIGS assays, and further investigation identified preparation issues of the ICMs by a new team. Internal preparation procedures were adjusted to improve the monitoring of the in-house ICM fabrication performance, after which the vast majority of ICM failures were obvious ICM ID recording errors. In addition, the Bomboré Mine Department (also using BIGS for the LeachWell analysis of grade control samples) confirmed that no analytical issues were detected with BIGS analyses during this same period, which provides further corroboration that the failures were a result of in-house issues, rather than the lab analyses themselves. In response to investigation findings, Orezone accepted all BIGS LeachWELL batches with failed ICMs from the April-May 2022 period.

Changes to in-house ICM preparation procedure from May 2022 include:

- Tracking of the CRM pot number for each of the ICM samples prepared, the date and identifying the team members involved in preparation.
- When an ICM is inserted into the sample stream, a photograph of the ICM pouch, with its preparation ID, as well as the sample ticket used in the stream of samples to be sent to the preparation laboratory, is taken, and archived. This ensures the tracking from the CRM pot to the sample number of the ICM inserted in a LW job. The jobs impacted by this problem were from the P17S drilling program.

11.3.2 Fire Assaying

A number of commercial CRMs and commercially certified blank reference materials, sourced mostly from Rocklabs Ltd., (Rocklabs), and to a lesser extent from Analytical Solutions Ltd. and AMISO were inserted into drill core sampling sample streams for fire assay. These reference materials are listed in Table 11.3.4.

Table 11.3.4 Specifications of Control Samples Used for Fire Assay Analysis of Primary Samples from October 2007 to May 2023

Reference Material	Type	Source	Certified Grade Au g/t		Number of Samples Used
			Mean	Standard Deviation	
Amis43	CRM	Amiso	3.57	0.130	4
AR36-210	ICM	Orezone	1.502	0.083	1
BLK10	Certified Blank	Rocklabs Ltd	<0.002	--	33
BLK12	Certified Blank	Rocklabs Ltd	<0.002	--	182
BLK13	Certified Blank	Rocklabs Ltd	<0.002	--	93
BLK24	Certified Blank	Rocklabs Ltd	<0.002	--	195
BLK31	Certified Blank	Rocklabs Ltd	<0.002	--	14
BLK35	Certified Blank	Rocklabs Ltd	<0.002	--	80
BLK44	Certified Blank	Rocklabs Ltd	<0.002	--	4
OREAS 15h	CRM	Analytical Solutions Ltd	1.019	0.025	1
OREAS 65a	CRM	Analytical Solutions Ltd	0.52	0.017	1
OREASBLK	Certified Blank	Analytical Solutions Ltd	<0.024	--	12
OxA71	CRM	Rocklabs Ltd	0.0849	0.0011	8
OxC88	RMCRM	Rocklabs Ltd	0.203	0.0015	111
OxD73	CRM	Rocklabs Ltd	0.416	0.0025	22

Reference Material	Type	Source	Certified Grade Au g/t		Number of Samples Used
			Mean	Standard Deviation	
OxE86	CRM	Rocklabs Ltd	0.613	0.0035	39
OxG83	CRM	Rocklabs Ltd	1.002	0.0045	97
OxJ47	CRM	Rocklabs Ltd	2.384	0.010	29
OxJ68	CRM	Rocklabs Ltd	2.342	0.0125	14
OxK48	CRM	Rocklabs Ltd	3.557	0.021	12
SE58	CRM	Rocklabs Ltd	0.607	0.003	63
SF30	CRM	Rocklabs Ltd	0.832	0.004	158
SG56	CRM	Rocklabs Ltd	1.027	0.0055	1
SH24	CRM	Rocklabs Ltd	1.326	0.008	146
SH41	CRM	Rocklabs Ltd	1.344	0.0075	65
SH55	CRM	Rocklabs Ltd	1.375	0.007	2
SL34	CRM	Rocklabs Ltd	5.893	0.0285	82
SL46	CRM	Rocklabs Ltd	5.867	0.033	7
SN26	CRM	Rocklabs Ltd	8.543	0.036	6

11.3.3 Tail Fire Assaying

All LeachWELL samples with grades >0.2 g/t or >0.5 g/t Au are selected for additional verification as part of the Orezone tail fire assay (TFA) program. The cut-off grade for TFA since the 2010 RC drill programme is 0.2 g/t Au, whereas until 2009 the TFA were done only on the samples with a soluble gold grade of at least 0.5 g/t Au.

The objective of the TFA is to determine the amount of gold that has not been leached in 10 hours and to calculate the head grade from a given sample.

The TFA samples are first neutralized, then dried and pulverized with LM2 pulverizer at the BIGS laboratory. ICMs, certified blanks and CRMs are added to samples together with pulp duplicates and lab-aware pulp duplicates prior to submission to the analytical laboratory.

About 21% of the samples in the Project database were assayed as part of the TFA programme (including control samples and check assays). The recovery rates derived from the ICM demonstrate >90% recovery, with a drop in the average recovery of about 5% between 1.0 and 0.2 g/t Au, and a drop of about 3% in the average recovery from the oxide to the transition zone samples.

CRMs and blanks sourced from Rocklabs Ltd., Analytical Solutions Ltd., and AMISO were used on all LeachWell tails sampling (refer to Table 11.3.7.)

Table 11.3.5 Specifications fo ICM Used for LeachWELL Analysis of Primary Samples from October 2007 to May 2023

Reference Material	Type	Source	Certified Grade Au g/t		Number of Samples Used
			Mean	Standard Deviation	
AA23-215_2008_1	ICM	Orezone	0.268	0.015	2
AA23-230*	ICM	Orezone	0.536	0.023	78
AA43-210_2011_1	ICM	Orezone	0.083	0.006	47
AA43-220_2011_1	ICM	Orezone	0.165	0.009	14
AA43-230_2011_1	ICM	Orezone	0.248	0.015	123
AO15h-210_2021_1	ICM	Orezone	0.051	0.002	16
AO15Pb-210_2021_1	ICM	Orezone	0.053	0.002	15
AO60b-210_2021_1	ICM	Orezone	0.129	0.009	15
AO62c-206_2011_1	ICM	Orezone	0.264	0.015	299
AO62c-218_2011_1	ICM	Orezone	0.791	0.040	64
AO67a-210_2012_1	ICM	Orezone	0.112	0.007	15
AR101-205_2017_1	ICM	Orezone	0.015	0.005	3
AR106-205_2017_1	ICM	Orezone	0.015	0.004	4
AR106-210_2017_1	ICM	Orezone	0.030	0.008	5
AR108-210_2020_1	ICM	Orezone	0.287	0.015	112
AR108-215_2020_1	ICM	Orezone	0.431	0.020	28
AR109-205_2017_1	ICM	Orezone	0.005	0.001	6
AR109-210_2017_1	ICM	Orezone	0.010	0.005	13
AR117-205_2021_1	ICM	Orezone	0.211	0.003	7
AR172-210_2022_1	ICM	Orezone	0.422	0.003	7
AR26-210_2008_1	ICM	Orezone	0.427	0.020	91
AR26-220_2008_1	ICM	Orezone	0.854	0.042	54
AR34-210_2010_1	ICM	Orezone	0.295	0.015	1
AR36-205*	ICM	Orezone	0.751	0.046	1,970
AR36-207_2008_1	ICM	Orezone	1.051	0.050	3
AR36-210*	ICM	Orezone	1.502	0.083	896
AR36-220_2008_1	ICM	Orezone	3.004	0.120	5
AR38-205*	ICM	Orezone	0.214	0.011	306
AR38-210_*	ICM	Orezone	0.429	0.020	99
AR46-210_2010_1	ICM	Orezone	0.293	0.015	171
AR47-220_2010_1	ICM	Orezone	0.238	0.015	150

Reference Material	Type	Source	Certified Grade Au g/t		Number of Samples Used
			Mean	Standard Deviation	
AR48-220_2010_1	ICM	Orezone	0.356	0.018	22
AR50-205_2010_1	ICM	Orezone	0.217	0.013	463
AR50-210_2010_1	ICM	Orezone	0.434	0.022	164
AR51-210_2011_1	ICM	Orezone	0.295	0.015	352
AR55-210_2012_1	ICM	Orezone	0.069	0.005	97
AR55-220_2012_1	ICM	Orezone	0.138	0.008	54
AR56-210_2012_1	ICM	Orezone	0.051	0.004	61
AR56-220_2012_1	ICM	Orezone	0.103	0.006	14
AR61-105_2012_1	ICM	Orezone	0.297	0.015	27
AR61-10510_2012_1	ICM	Orezone	0.565	0.025	2
AR61-205_2014_1	ICM	Orezone	0.148	0.009	94
AR61-210_2012_1	ICM	Orezone	0.297	0.015	1,209
AR66-210_2021_1	ICM	Orezone	0.054	0.001	46
AR75-201_2012_1	ICM	Orezone	0.250	0.015	106
AR75-202_2012_1	ICM	Orezone	0.500	0.025	96
AR75-205_2012_1	ICM	Orezone	1.251	0.060	28
AR76-105_2018_1	ICM	Orezone	0.298	0.015	1
AR76-210_2018_1	ICM	Orezone	0.298	0.015	91
AR89_110_2017_1	ICM	Orezone	0.008	0.006	1
AR91-205_2020_1	ICM	Orezone	0.217	0.011	89
ARK2-105_2014_1	ICM	Orezone	0.174	0.010	13
ARK2-210_2012_1	ICM	Orezone	0.174	0.010	402
ARK2-215_2011_1	ICM	Orezone	0.261	0.015	503
ARK4-205_2021_1	ICM	Orezone	0.087	0.001	19
ARK4-210_2018_1	ICM	Orezone	0.173	0.010	48
ARK4-215_2020_1	ICM	Orezone	0.260	0.015	63
ARP1-1025_2012_1	ICM	Orezone	0.301	0.015	5
ARP1-105_2012_1	ICM	Orezone	0.603	0.030	45
ARP1-205_2011_1	ICM	Orezone	0.301	0.015	968
ARP1-210_2011_1	ICM	Orezone	0.603	0.030	2,268
ARP1-215_2012_1	ICM	Orezone	0.904	0.045	178
ARP1-220_2011_1	ICM	Orezone	1.205	0.060	146
ARP3-203_2021_1	ICM	Orezone	0.184	0.004	12
ARP3-205_2018_1	ICM	Orezone	0.306	0.015	292

Reference Material	Type	Source	Certified Grade Au g/t		Number of Samples Used
			Mean	Standard Deviation	
ARP3-210_2018_1	ICM	Orezone	0.612	0.030	69
ARP5-205_2022_1	ICM	Orezone	0.301	0.004	2
ARP5-210_2022_1	ICM	Orezone	0.603	0.005	2
ARS48_110_2017_1	ICM	Orezone	3.025	0.170	1
ARS48-1025_2012_1	ICM	Orezone	0.756	0.046	7
ARS48-105_2012_1	ICM	Orezone	1.513	0.085	63
ARS48-10510_2012_1	ICM	Orezone	2.881	0.120	2
ARS48-205_2011_1	ICM	Orezone	0.756	0.010	1,080
ARS48-210_2011_1	ICM	Orezone	1.513	0.085	2,856
ARS87-203_2021_1	ICM	Orezone	0.463	0.009	15
ARS87-205_2018_1	ICM	Orezone	0.772	0.046	176
ARS87-210_2018_1	ICM	Orezone	1.544	0.085	14
ARS88-210_2021_1	ICM	Orezone	1.986	0.017	5
BLKORZ	In-House Blank	Orezone	<0.001	--	13,601
HISILK6_50_2022	ICM	Orezone	0.086	0.001	593
HISILK6_75_2022	ICM	Orezone	0.129	0.002	1
HISILP5_30_2022	ICM	Orezone	0.181	0.004	226
HISILP5_50_2022	ICM	Orezone	0.301	0.004	54
HiSilP6_50_2022*	ICM	Orezone			1
OxL159_30_2023	ICM	Orezone	0.088	0.002	203
OXN173_30_2022	ICM	Orezone	0.115	0.002	281
OxP172_100_2023	ICM	Orezone	0.753	0.007	13
OXP172_30_2022_1	ICM	Orezone	0.226	0.005	122
OXP172_50_2022_1	ICM	Orezone	0.376	0.005	5
OxQ170_100_2023	ICM	Orezone	1.247	0.011	7
OXQ170_30_2022	ICM	Orezone	0.374	0.008	267
OXQ170_50_2022	ICM	Orezone	0.623	0.009	46
OxQ170_50_2023	ICM	Orezone	0.147	0.002	12
SL123_30_2022_1	ICM	Orezone	0.088	0.002	36
SN117_30_2022_1	ICM	Orezone	0.127	0.002	94
SP116_30_2022_1	ICM	Orezone	0.271	0.005	40
SP116_50_2022_1	ICM	Orezone	0.452	0.006	73
SP122_30_2023	ICM	Orezone	0.271	0.005	48
SQ87_100_2022	ICM	Orezone	1.544	0.013	78

Reference Material	Type	Source	Certified Grade Au g/t		Number of Samples Used
			Mean	Standard Deviation	
SQ87_30_2022	ICM	Orezone	0.463	0.009	18
SQ87_50_2022	ICM	Orezone	0.772	0.011	162
SQ88_100_2022_1	ICM	Orezone	1.986	0.017	3
SQ88_30_2022_1	ICM	Orezone	0.596	0.012	45
SQ88_50_2022_1	ICM	Orezone	0.993	0.014	14

Note: LeachWELL

* Labelling error: HISILK6 or HISILP5

Table 11.3.6 Specifications of Control Samples Used for Fire Assay Analysis of Umpire Samples with Gravimetric Finish from October 2007 to May 2023

Reference Material	Type	Source	Certified Grade Au g/t		Number of Samples Used
			Mean	Standard Deviation	
BLK44	Certified Blank	Rocklabs Ltd	<0.002	--	48
BLK50	Certified Blank	Rocklabs Ltd	<0.002	--	23
BLK51	Certified Blank	Rocklabs Ltd	<0.002	--	7
BLK90	Certified Blank	Rocklabs Ltd	<0.002	--	13
HiSilK2	CRM	Rocklabs Ltd	3.474	0.017	6
HiSilK6	CRM	Rocklabs Ltd	3.446	0.0165	3
HiSilP1	CRM	Rocklabs Ltd	12.05	0.065	46
HiSilP3	CRM	Rocklabs Ltd	12.24	0.0355	7
Oreas 151a	CRM	Analytical Solutions Ltd	0.043	0.002	2
OxC109	CRM	Rocklabs Ltd	0.201	0.001	1
OxE106	CRM	Rocklabs Ltd	0.606	0.002	2
OXN173	CRM	Rocklabs Ltd	7.668	0.023	1
OXP172	CRM	Rocklabs Ltd	15.057	0.0575	5
SL123	CRM	Rocklabs Ltd	5.899	0.0195	2
SL61	CRM	Rocklabs Ltd	5.931	0.0285	27
SL76	CRM	Rocklabs Ltd	5.96	0.026	3
SN117	CRM	Rocklabs Ltd	8.443	0.018	3
SN91	CRM	Rocklabs Ltd	8.679	0.028	5
SP116	CRM	Rocklabs Ltd	18.091	0.0515	2

Reference Material	Type	Source	Certified Grade Au g/t		Number of Samples Used
			Mean	Standard Deviation	
SQ48	CRM	Rocklabs Ltd	30.25	0.085	22
SQ87	CRM	Rocklabs Ltd	30.87	0.105	1
SQ88	CRM	Rocklabs Ltd	39.72	0.14	4

Table 11.3.7 Specifications of Control Samples Used for Fire Assay on LeachWELL Residues from October 2007 to May 2023

Reference Material	Type	Source	Certified Grade Au g/t		Number of Samples Used
			Mean	Standard Deviation	
Amis43	CRM	Amiso	3.57	0.13	1
AR61-210_2012_1	ICM	Orezone	0.297	0.015	1
BLK12	Certified Blank	Rocklabs Ltd	<0.002	--	47
BLK13	Certified Blank	Rocklabs Ltd	<0.002	--	277
BLK24	Certified Blank	Rocklabs Ltd	<0.002	--	205
BLK31	Certified Blank	Rocklabs Ltd	<0.002	--	168
BLK35	Certified Blank	Rocklabs Ltd	<0.002	--	171
BLK44	Certified Blank	Rocklabs Ltd	<0.002	--	860
BLK50	Certified Blank	Rocklabs Ltd	<0.002	--	675
BLK51	Certified Blank	Rocklabs Ltd	<0.002	--	215
BLK90	Certified Blank	Rocklabs Ltd	<0.002	--	69
BLKORZ	In-House Blank	Orezone	<0.001	--	75
Oreas 151a	CRM	Analytical Solutions Ltd	0.043	0.002	55
OREAS 15f	CRM	Analytical Solutions Ltd	0.334	0.016	121
OREAS 15h	CRM	Analytical Solutions Ltd	1.019	0.025	65
OREAS 65a	CRM	Analytical Solutions Ltd	0.52	0.017	98
OREASBLK	Certified Blank	Analytical Solutions Ltd	0.024	--	769
OXA131	CRM	Rocklabs Ltd	0.077	0.001	25
OxA71	CRM	Rocklabs Ltd	0.0849	0.0011	493
OxA89	CRM	Rocklabs Ltd	0.0836	0.00125	431
OxC102	CRM	Rocklabs Ltd	0.207	0.002	41
OxC109	CRM	Rocklabs Ltd	0.201	0.001	370
OxC145	CRM	Rocklabs Ltd	0.212	0.001	45

Reference Material	Type	Source	Certified Grade Au g/t		Number of Samples Used
			Mean	Standard Deviation	
OxC72	CRM	Rocklabs Ltd	0.205	0.0015	199
OxC88	CRM	Rocklabs Ltd	0.203	0.0015	258
OxD127	CRM	Rocklabs Ltd	0.459	0.002	40
OxD73	CRM	Rocklabs Ltd	0.416	0.0025	2
OxD87	CRM	Rocklabs Ltd	0.417	0.002	14
OxE101	CRM	Rocklabs Ltd	0.607	0.0025	209
OxE106	CRM	Rocklabs Ltd	0.606	0.002	348
OxE143	CRM	Rocklabs Ltd	0.621	0.002	38
OxE86	CRM	Rocklabs Ltd	0.613	0.0035	230
OxF65	CRM	Rocklabs Ltd	0.805	0.007	95
OxG83	CRM	Rocklabs Ltd	1.002	0.0045	266
OxJ47	CRM	Rocklabs Ltd	2.384	0.01	72
OxJ68	CRM	Rocklabs Ltd	2.342	0.0125	65
OxK48	CRM	Rocklabs Ltd	3.557	0.0095	76
SE44	CRM	Rocklabs Ltd	0.606	0.003	197
SE58	CRM	Rocklabs Ltd	0.607	0.003	309
SF30	CRM	Rocklabs Ltd	0.832	0.004	174
SF57	CRM	Rocklabs Ltd	0.848	0.003	3
SG56	CRM	Rocklabs Ltd	1.027	0.0055	260
SG66	CRM	Rocklabs Ltd	1.086	0.0045	74
SH41	CRM	Rocklabs Ltd	1.344	0.0075	302
SH55	CRM	Rocklabs Ltd	1.375	0.007	169
SL34	CRM	Rocklabs Ltd	5.893	0.0285	40
SL46	CRM	Rocklabs Ltd	5.867	0.033	2

11.4 Comments

Orezone personnel routinely monitors sample shipments and assay deliveries by the preparation and assaying laboratories and promptly addresses any issues that arise. Assay results and QA/QC data produced by the various laboratories are also assessed and analyzed using various bias and precision charts. At the end of each drilling programme, Orezone produces quality control reports summarizing protocol and QA/QC results.

The Author has reviewed QA/QC data and reports provided by Orezone for drilling completed at Bomboré from 2007 to 2023 and concludes that sample preparation, security, analytical procedures, and QA/QC undertaken by Orezone adequate for the purposes of this Mineral Resource Estimate and that there are no factors that materially impact the reliability or accuracy of the dataset employed in the calculation.

12.0 DATA VERIFICATION

12.1 P&E Database Verification

12.1.1 Assay Verification

2008 to 2009 Borehole Assay Data

The Authors of this Technical Report section conducted verification of the Bomboré Project borehole assay database for gold, by comparison of the database entries with assay certificates, supplied to the Authors by SGS Burkina Faso SA, ALS Burkina SARL, and BIGS laboratory in Ouagadougou, in comma-separated values (csv) format and Portable Document Format (pdf) format.

A selection was made to identify several laboratory certificates with significant assay results that would cover a large range of values, assay methods and at least 10% of the total assay database. Assay certificates were selected by extracting the average grade of all samples within each certificate. Assays from the selected certificates were compiled, giving a total of 49,248 assay results for primary samples and 22,032 for the tails samples assayed by Bottle roll method. Verification was first conducted prior to 2017, and then repeated in early 2020 to ensure coverage of the additional drilling results following November 2017. Assay data ranging from 2008 through 2019 were verified for the Bomboré Project. Exactly 11.45% of the database was checked for gold and no errors or discrepancies were encountered. More detailed verification statistics for the South and North Deposits are given in Table 12.1.1.

Table 12.1.1 April 2020 Database Verification Statistics

29 April 2020		South	North
Percentage Verification	Bottle roll	8.5%	10.3%
	Tails results	20.1%	19.2%

2021 to 2023 Borehole Assay Data

In July 2023, the Authors conducted verification of the additional borehole assays for the B1 and B2 (North), SIGA and P11 (South) and P17 models. Verification was undertaken by comparison of database entries with assay certificates supplied directly by the BIGS laboratory in Ouagadougou and SGS Burkina Faso SA, in Excel Binary File (xls) format. Assay data ranging from 2021 to 2023 were verified, with approximately 63% (71,628 out of 113,938) of all primary samples and approximately 76% (23,959 out of 31,646) of the constrained primary samples checked for gold. In addition, 72% (14,963 out of 20,782) of the tailings samples and about 73% (12,949 out of 17,823) of the constrained tailings samples were checked for gold. No errors were observed in the data.

12.2 P&E Site Visit and Independent Sampling

The Bomboré Project was visited by Mr. Antoine Yassa, P.Geo., from 9 October 2017 to 14 October 2017 for the purposes of completing a site visit and due diligence sampling. General data acquisition procedures, core logging procedures and QA/QC were discussed during the visit.

Mr. Yassa collected 50 samples from 15 RC boreholes and 35 cored boreholes during the site visit. The RC chip samples were weighted between 1.8 kg and 2.6 kg collected from the remaining 10 to 15 kg bags. The ¼-drill core samples weighed between 0.85 kg and 2.15 kg. Effort was made to sample three ranges of grades from the lower grade around 0.3 g/t Au to the high-grade around 3.5 g/t Au. Mid-range was set at an average of 0.6 g/t Au. Ten different deposits were sampled to cover a large selection across the Bomboré Property. At no time, were any employees of the Company advised as to the identification of the samples to be chosen during the visit.

The samples were brought by Mr. Yassa to SGS in Ouagadougou, Burkino Faso, for preparation and analysis. Sample processing services at SGS are ISO 17025 accredited by the Standards Council of Canada. Quality Assurance procedures include standard operating procedures for all aspects of the processing and include protocols for training and monitoring of staff. ONLINE LIMS is used for detailed worksheets, batch and sample tracking including weights and labelling for all the products from each sample. SGS Ouagadougou is accredited by the South African National Accreditation System (SANAS) for meeting the requirements of the ISO/IEC 17025 standard, for specific registered tests for the minerals industry.

Samples were analyzed for gold only using SGS LWL69M fire assay method, which is an Accelerated Cyanide Leach, 1,000 g/2,000 ml, Solvent Extraction, Au/AAS. Results of the site visit due diligence samples are presented in Figure 12.3.1.

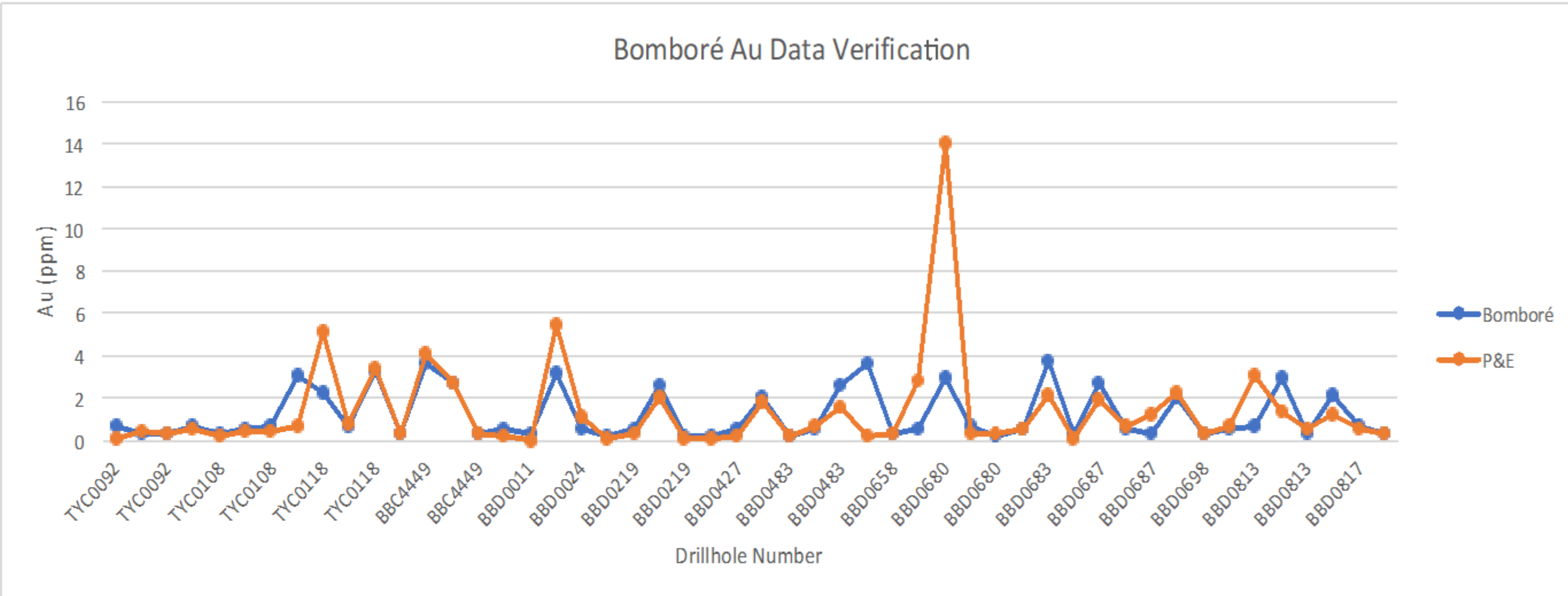
12.3 Adequacy of Data

Verification of the Bomboré Project data, used for the current Mineral Resource Estimate, has been undertaken by the Authors, including evaluation of the QA/QC programme undertaken by Orezone, verification of drilling assay data and via a site visit and due diligence sampling.

The Authors consider that there is good correlation between the independently collected verification samples analyzed at SGS and the Company data for gold.

The Authors are satisfied that sufficient verification of the borehole data has been undertaken and that the supplied data are of good quality and suitable for use in the current Mineral Resource Estimate of the Bomboré Project.

Figure 12.3.1 Site Visit Sample Comparison for Gold



13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Summary

Extensive metallurgical testwork has been completed on Bomboré ore to confirm plant design parameters. The regolith, oxide and upper transition ore types will be processed in the existing oxide plant consisting of a single stage ball mill and CIL circuit. The lower transition and fresh ore types will be processed in the new hard rock plant consisting of a primary jaw crusher, single stage SAG mill and CIL circuit. The gold recoveries anticipated from the ore types are presented in Table 13.1.1.

Table 13.1.1 Recommended Gold Recovery % (including losses)

Pit Zone	Oxide and Regolith	Upper Transition	Lower Transition	Fresh
Maga	91.8	89.0	86.0	81.7
P8P9	91.8	89.0	86.0	84.0
Siga	91.8	89.0	86.0	81.7
P11	91.8	89.0	86.0	81.7
P16	91.8	89.0	86.0	81.7
P17	95.0	95.0	95.0	95.0

Bomboré Gold Mine is currently processing oxide ore material at a rate of 5.9 Mtpa with a grind size of 90% passing 125 microns (P_{90} 125µm) with a leach residence time of 21 hours. Since plant start up, the head grade has averaged 0.90 g/t Au and the tails grade has averaged 0.07 g/t Au (from September 2022 to the start of April 2023). The average gold recovery has been 91.8% and there is no obvious relationship between head grade and recovery. Plant throughput continues to be increased; however, it is anticipated that continuous improvement initiatives in the plant will maintain oxide ore average recoveries at 91.8%.

Upper transition material will report to the existing oxide plant and lower transition material will report to the new hard rock plant. The thin and unpredictable weathering profile makes the transition ore difficult to schedule in the mine plans and decisions on which circuit this material will be directed to will be made by the mine operations team in the field. Metallurgical testwork has shown that the gold recovery for the transition material is lower than the oxide ore but higher than the fresh ore. A gold recovery of 89% and 86% is recommended for the upper and lower transition ores respectively.

The proposed hard rock plant is designed to process fresh and lower transition material at a rate of 4.4 Mtpa with a grind of P_{80} 75 µm and the CIL circuit is sized for 27.7 hours leach time. This additional residence time above the 24h design time will allow for throughput above the design of 4,4Mtpa should the mill perform above design. Recent metallurgical test work conducted in 2023 has confirmed that gold recovery is independent of head grade, oxygen addition is beneficial to the leach kinetics, a 24 hour leach time is sufficient and the optimal grind size is P_{80} 75 µm. A coarser grind at P_{80} 106 µm with a 24 hour leach exhibits a fixed tail increase of about 0.05 g/t, most likely due to insufficient liberation as a result of the coarse grind. Recent metallurgical testwork on the Siga and Maga fresh ore types has shown that gold recoveries of 81.7% can be expected after allowing for plant losses.Recommended recovery for P11 fresh ore is 81.7% based on similarities with Siga. P8P9 fresh ore recovery is 84% based on recent testwork indicating higher gold recoveries than Siga samples and mineralogical examination showing presence of free pyrrhotite similar to P17.

The metallurgical testwork on the P17 orebodies (including P17S) supports a laboratory recovery of 96% to 97% on all ore types and all feed grades. After allowing for plant losses, the recommended gold recovery for P17 ore is 95%.

Comminution testwork data has been compiled from a total of 42 Axb tests and 43 BWi tests. Orway Mineral Consultants Ltd. (OMC) determined the 85th percentile of Axb and BWi test results, and average of the abrasion index (Ai) and ore relative density for lower transition and fresh ores for each pit. In general, the 85th percentile ore characteristics indicate a very competent and hard ore, low Axb and high BWi. The proposed 4.4 Mtpa comminution circuit will include a primary jaw crusher, a single stage 18,000 kW twin pinon SAG mill, hydrocyclones for product size classification and space for a potential future pebble crusher. The comminution circuit will receive run of mine (ROM) ore predicted to have a ROM ore top size (F_{100}) of 700 mm. The milling circuit will grind to a final product size P_{80} of 75 µm.

13.2 Introduction

Extensive testwork programs have been carried out at different laboratories for the Bomboré Project with the first test program carried out in 1997 and the latest completed in 2023. The test programs were conducted on drill core composites, RC cuttings, and RAB (Rotary Air Blasting) drill samples considered representative of the ore deposit at the time of each test program. A summary list of the programs is included in Table 13.2.1. A map illustrating the metallurgical sampling locations is shown in Figure 13.2.1 and Figure 13.2.2.

Table 13.2.1 Summary of Testwork Programs

Program	LeachWell Recoveries	Head Analysis	Variability	Cyanidation	Gravity	Flotation	Carbon-in-Leach (CIL)	Carbon Adsorption & Equilibrium	Column Leach (HL)	Comminution	Scrubbing	Gold Deportment	Petrography	Thickening / Rheology	Neutralization	Lime Demand	Acid Mine Drainage
SGS / ITS 1997			✓	✓									✓				
Osborne 2008			✓	✓													
AMMTEC 2009		✓	✓	✓	✓	✓	✓		✓	✓							✓
McClelland 2012*		✓	✓	✓	✓	✓	✓			✓			✓	✓	✓		
Phillips 2012										✓							
OREZONE Scrubbing 2012			✓	✓							✓	✓					
Met-Solve 2013											✓	✓					
SGS Lakefield 2013										✓							
COREM 2013				✓						✓			✓				
Met-Solve 2014				✓			✓				✓						
Consolidated Database 2013	✓																
Kappes 2014		✓	✓	✓			✓		✓		✓	✓		✓	✓		
SGS Lakefield 2014										✓							
SGS Lakefield 2016				✓	✓	✓				✓			✓				
SGS Lakefield 2017/2018			✓	✓						✓						✓	
Outotec 2018														✓			
Base Metallurgical Lab 2019		✓	✓	✓						✓			✓	✓			
SGS Lakefield 2019								✓									
Maelgwyn 2023		✓	✓	✓	✓		✓	✓						✓		✓	

*Includes Pocock report in appendix

The following sections summarize selected testwork results that are pertinent to the Project. Additional details of each program are provided in the individual testwork reports.

Figure 13.2.1 Metallurgical Sampling Locations (Oxide Samples)

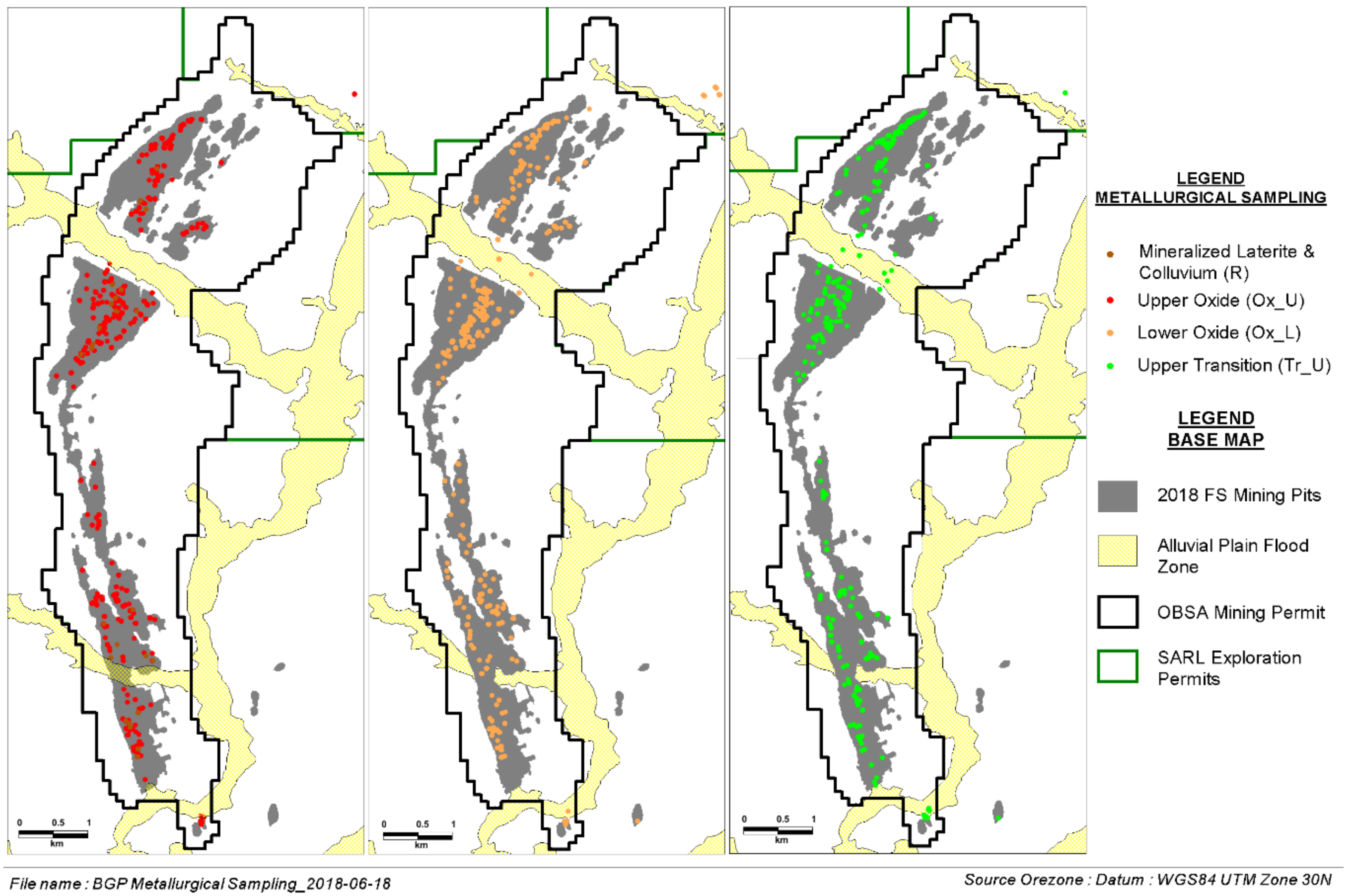
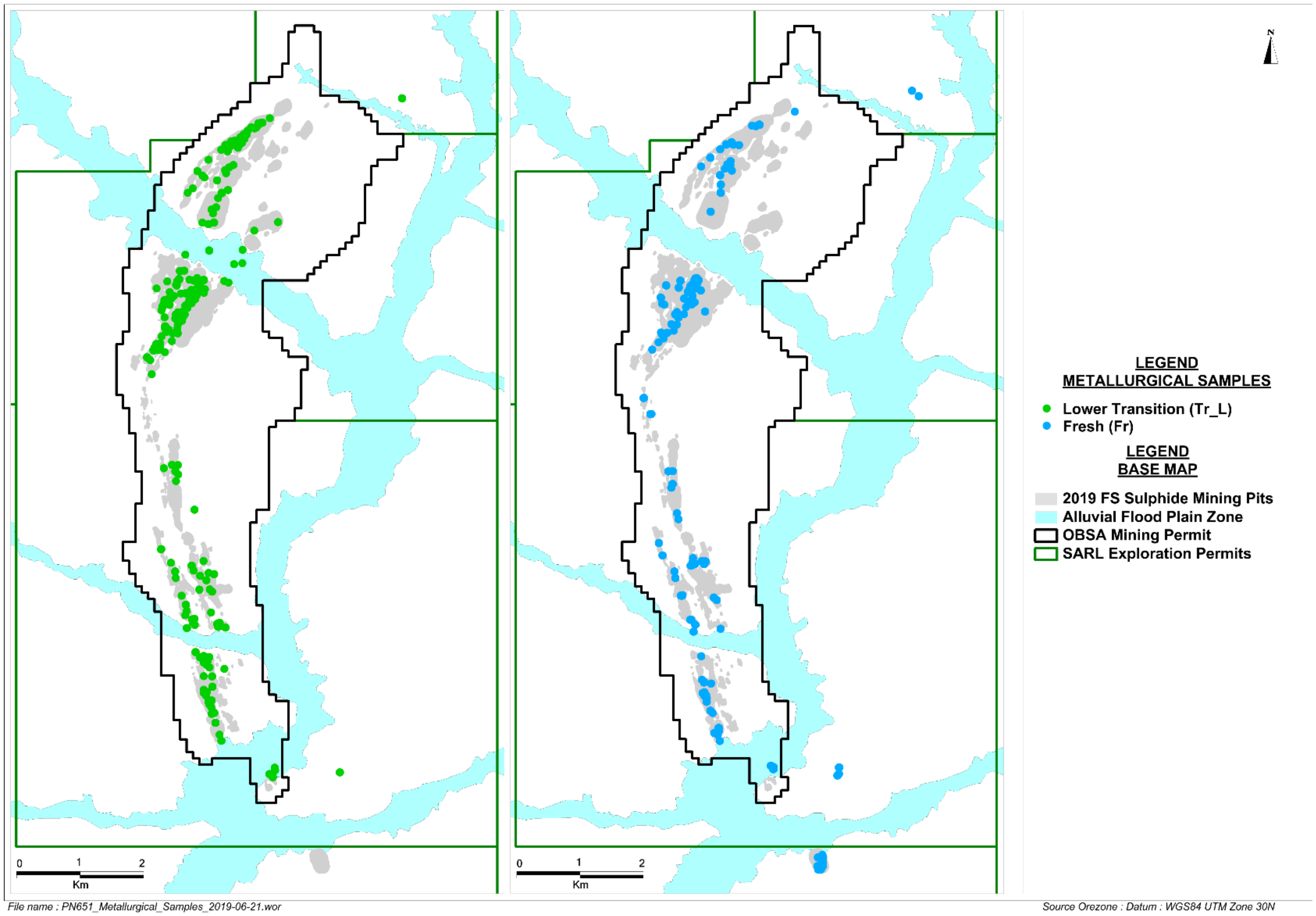


Figure 13.2.2 Metallurgical Sampling Locations (Sulphide Samples)



13.3 Historical Testwork Programs

13.3.1 SGS/ITS 1997

Ninety-one (91) samples from the Siga, Maga and P8/P9 mineralized zones were collected for the 1997 testwork program. Of these, 67 samples were from RC drilling, which were used in bottle roll tests at SGS (Ghana). The remaining 24 samples were from RAB drillings that were used for bottle rolls ?at the ITS Laboratories (Burkina Faso). Mineralogy analyses were also conducted on 10 diamond drill core samples at the SGS Lakefield Research facility in Canada.

Highlights from this program are as follows:

<u>SGS (Ghana)</u>

- Of the 67 RC samples, 57 were oxide and transition ore types, and 10 were sulphides.
- Gold extraction by cyanide leaching ranged from 67% to 99%, with an average of 92% for the RC oxide and transition samples, and 87% for the sulphide samples.

<u>ITS Laboratories (Burkina Faso)</u>

- All 24 RAB drill samples were oxide material.
- Gold extractions by cyanide leaching ranged from 63% to 95%, with an average of 85%.

<u>SGS Lakefield Research (Canada)</u>

- Of the 10 diamond drill samples, eight were oxides and two were sulphides.
- The occurrences of gold in the oxide samples fell under the category of a) liberated gold, b) liberated gold with discontinuous rims and attachment of goethite ± silicates, and c) gold attached to, or encapsulated in, gangue minerals.
- One sulphide sample showed 37% locked gold in pyrite, and the other sulphide sample showed no visible gold.
- The grain size of gold particles ranged from 0.5 µm to 800 µm, with ~85% less than 30 µm in diameter.

13.3.2 Osborne 2008 Testwork Program

One hundred and eighty-four (184) composite RC samples from the Siga, Maga, Maga S, P11, KT, P8/P9 and CFU mineralized zones were used in this program to conduct three streams of testwork. The tests involved bottle roll bulk leach extractable gold (BLEG) tests conducted at Alibab (Ouagadougou) and fire assay (FA) conducted at Alibab (Bamako).

Descriptions of each testwork stream are as follows:

- Stream 1 – Samples were pulverized to 75 µm, subjected to 24-hour bottle roll BLEG, and FA on leach residue at 24 hours.
- Stream 2 – Samples were pulverized to 106 µm, subjected to 24-hour bottle roll BLEG, and FA on leach residue at 24 hours. The +2 mm fractions were subjected to FA for coarse gold detection.
- Stream 3 – Samples were pulverized to 75 µm, subjected to 72-hour bottle roll BLEG, and FA on leach residue at 2, 4, 8, 24, 48 and 72 hours.

The highlights from the leach test results are as follows:

- According to Osborne, after eliminating one sample, gold extraction at 24 hours averaged 93% for oxide, 92% for transition, and 83% for sulphide samples.
- However, based on Lycopodium's observation developed from raw testing data supplied by Orezone, gold extraction at 24 hours averaged 85% for oxide, 80% for transition, and 63% for sulphide samples.
- Leaching was substantially complete after 24 hours for all samples.

13.3.3 AMMTEC 2009 Testwork Program

The 2009 AMMTEC program was a scoping-level metallurgical testwork program supervised by GBM. The program included head analysis, acid mine drainage (AMD) analysis, comminution testwork, heap leach amenability via column leach tests, CIL cyanidation, grind optimization, and flotation testwork. The samples provided to AMMTEC were PQ and HQ drill cores taken from the fresh rock, transition, and oxide ore zones.

The results showed that all samples tested are capable of neutralizing any acidic solutions formed from oxidation of sulphides, therefore, acid mine drainage is not expected to be problematic at Bomboré.

The comminution results indicated that fresh ore is hard, while transition and oxide ores are soft. When the fresh rock A x b parameters are compared to other samples in the JKTech database, only about 20% are harder. Fresh ore is moderately abrasive, while transition ore is mildly abrasive, and oxide ore is non-abrasive.

A high-level summary of the AMMTEC leach and flotation results is as follows:

- Coarse crush cyanidation results indicated that <19 mm crush size would be sufficient for heap leaching transition ore and <25 mm for the oxide ore. Coarse crush cyanidation results for the fresh rock showed low gold extraction even at the finest crush size of <4 mm, therefore, column leach tests were not conducted for the fresh rock samples, and only conducted for the transition and oxide ores.

- Results from the column leach tests predicted that heap leaching could provide a gold recovery of approximately 80% for both transition and oxide ores.

- CIL results (at finer grind sizes) indicated that gold recovery could improve by 10% to 12% when compared to heap leaching.

- Fresh rock samples contained sulphide sulphur in reasonable quantity, therefore, good flotation recoveries were achieved for these, however, the concentrate grades were not high enough to be considered a saleable product and further treatment would be required.

- Ultra fine grinding (P_{80} of 10 µm) of the flotation concentrate followed by cyanidation was conducted to determine the overall combined recovery. The results showed lower recoveries than simply conducting whole ore CIL for both fresh rock samples.

- Results from gravity concentration followed by gravity tails cyanidation could not be properly assessed because in every one of the cases, the overall recovery was poorer than that of the whole ore CIL. It was expected that at the very least, the recovery would match that of the CIL. AMMTEC considered these results anomalous.

- It was recommended at the conclusion of this program that heap leaching should be the approach for transition and oxide ores, and the fresh rock ore should be treated by milling and leaching.

13.3.4 McClelland 2012 Testwork Program

The 2012 McClelland program was a feasibility-level metallurgical testwork program supervised by Mr. J. Woods, a metallurgical consultant from WSP. The testwork included ore variability composite testing, comminution testing, CIL/CIP, residue characterization, and waste rock testing. Gravity concentration and bulk flotation tests were conducted at a scoping-level. A total of 76 drill core samples were submitted for a detailed head analysis. Four composite samples were then formed from 33 oxide samples and 33 sulphide samples, then stage-crushed to <12.5 mm, and categorized by ore type and grade. High and medium grade oxide ores were denoted as HGO and MGO, and high and medium grade hard rock ores were denoted as HGS and MGS. Selected head analysis results for the four composites are presented in Table 13.3.1.

Table 13.3.1 McClelland Head Analysis

Analysis	HGO	MGO	HGS	MGS
Fire Assay (g/t Au)	2.38	0.57	1.81	0.74
Met Screen Assay (g/t Au)	2.62	0.61	1.70	0.85
ICP Scans (g/t Ag)	1.0	2.0	1.0	1.0
Duplicate (g/t Ag)	1.2	1.54	1.09	1.78
Al %	8.05	7.87	6.06	6.39
g/t As	715	251	1,810	1,065
g/t Cu	120.5	174.5	84.5	75.0
Fe %	5.55	6.49	6.23	6.17
g/t Hg	0.02	0.03	0.02	0.03
g/t Ni	50.7	54.9	27.2	55.8
g/t Te	1.22	2.02	1.03	1.10
C (Total) %	0.11	0.07	1.04	1.09
C (Organic) %	0.11	0.02	0.07	0.05
C (Inorganic) %	<0.05	<0.05	0.93	0.98
S (total) %	0.04	0.16	1.88	2.26
S (Sulphate) %	0.01	0.12	0.01	<0.01
S (Sulphide) %	0.04	0.03	1.71	2.16

There were notable levels of aluminium (6% to 8%) in the samples. Mercury content was low, indicating that a mercury removal system need not be considered in the plant design. Copper was also present at low levels, indicating that excess cyanide consumption is not to be expected. There were notable levels of sulphides in the HGS and MGS samples as expected. The total organic carbon was also low (0.2% to 0.11%), indicating preg-robbing should not be expected.

Previous head analysis conducted by Osborne suggested the possibility of preg-robbing, therefore, a preg-robbing analysis was conducted in this program using a spiked (Au-AA31) and cyanide shake (Au-AA31a) test to determine the Preg Rob factor. A factor that approaches zero or negative value indicates low likelihood of preg-robbing. The analysis showed that the Bomboré oxide and hard rock ores are unlikely to encounter preg-robbing issues. Refer to Table 13.3.2 for more details.

Table 13.3.2 McClelland Preg-Robbing Analysis

Sample	TOC %	g/t Au					% Preg Rob Factor
		Au-AA31	Au-AA31a	Difference	Spike Value	Difference	
HGO	0.11	5.55	1.97	3.58	3.43	-0.15	-4.4
MGO	0.02	4.08	0.48	3.60	3.43	-0.17	-5.0
HGS	0.07	4.87	1.16	3.71	3.43	-0.28	-8.2
MGS	0.05	3.94	0.54	3.40	3.43	0.03	0.9

Scoping level gravity concentration tests were conducted at <850 µm feed size. The results showed that oxide ore did not respond well to gravity concentration treatment at this feed size, while the hard rock ore responded better but not particularly well either. Table 13.3.3 provides a summary of the results.

Table 13.3.3 McClelland Gravity Concentration Test Results

Sample	Mass Pull %		Grade, g/t Au				Gold Extraction %	
	Cleaner Conc.	Rougher Conc.	Cleaner Conc.	Rougher Conc.	Rougher Tails	Calc'd Head	Cleaner Conc.	Rougher Conc.
HGO	0.25	1.29	75.42	26.84	1.29	1.62	11.6	21.3
MGO	0.35	1.19	15.71	6.35	0.54	0.54	9.0	12.4
HGS	0.42	1.52	123.66	41.31	1.50	2.11	24.6	29.9
MGS	0.45	1.62	80.25	25.30	0.66	1.06	34.1	38.7

Microscopic examinations of each gravity concentrate showed presence of particulate gold in HGO, MGO and HGS composites, with size ranging from 0.16 mm to 0.71 mm.

Scoping level bulk sulphide flotation tests were conducted at P_{80} of 75 µm feed size. The results showed that the oxide ore did not respond well to conventional bulk sulphide flotation treatment at this feed size, while the hard rock ore responded well under the same conditions with gold extractions of 92.5% and 89.4% for the HGS and MGS, respectively. Table 13.3.4 provides a summary of the results.

Table 13.3.4 McClelland Flotation Concentration Test Results

Sample	Mass Pull %		Grade, g/t Au				Gold Extraction %	
	Cleaner Conc.	Rougher Conc.	Cleaner Conc.	Rougher Conc.	Rougher Tails	Calc'd Head	Cleaner Conc.	Rougher Conc.
HGO	6.27	21.51	15.40	5.77	0.62	1.73	55.9	71.8
MGO	1.35	8.16	12.58	3.66	0.39	0.66	25.9	45.5
HGS	6.07	9.68	26.98	17.38	0.15	1.82	90.1	92.5
MGS	9.92	23.43	7.25	3.32	0.12	0.87	82.7	89.4

Microscopic examinations of each flotation cleaner concentrate showed no presence of particulate gold.

Grind size optimization tests using bottle roll leaching by cyanidation were conducted at three $P_{100's}$ of 212 µm, 106 µm and 53 µm for the oxide ore, and five $P_{80's}$ - 150 µm, 106 µm, 75 µm, 53 µm, and 45 µm for the hard rock ore. A summary of the results is presented in Table 13.3.5.

Table 13.3.5 McClelland Summary of Grind Optimization Tests

Ore Type	Grind Size	72-hr Gold Extraction %	g/t Au			Reagents kg/t	
			Extracted	Tails	Calc'd Head	NaCN	Lime Added
Oxide (avg. of HGO & MGO)	P_{100} -212 µm	93.0	1.29	0.09	1.37	0.12	3.1
	P_{100} -106 µm	93.0	1.19	0.08	1.27	0.08	3.1
	P_{100} -53 µm	91.3	1.13	0.08	1.21	0.28	3.0
Sulphides (avg. of HGS & MGS)	P_{80} -150 µm	75.3	1.06	0.32	1.38	0.47	1.0
	P_{80} -106 µm	78.2	1.05	0.29	1.34	0.39	1.0
	P_{80} -75 µm	80.2	1.12	0.27	1.39	0.52	1.1
	P_{80} -53 µm	84.1	1.22	0.23	1.44	0.76	1.3
	P_{80} -45 µm	84.4	1.28	0.24	1.52	2.36	1.7

The results showed that both oxide and sulphide composites were readily amenable to direct cyanidation at all feed sizes. The oxide composites had consistently good gold extraction for all three feed sizes, while the sulphide composites appeared to have small improvement with decreasing feed size. Cyanide consumptions were low in all cases except for the P_{80} of 45 µm case (HGS) for sulphides where the consumption was significantly higher. The P_{80} of 45 µm case for the HGS composite was also the only test conducted with mechanical agitation. Cyanide consumption did appear to increase slightly with decreasing feed size. Lime consumptions were moderate for the oxide composites and low for the sulphide composites.

The leach kinetics for all the composites were considered rapid and gold extraction showed signs of plateauing after the 24-hour mark. The gold leach profiles are illustrated in Figure 13.3.1 to Figure 13.3.4.

Figure 13.3.1 McClelland Gold Leach Profile for High Grade Oxide Composite (HGO)

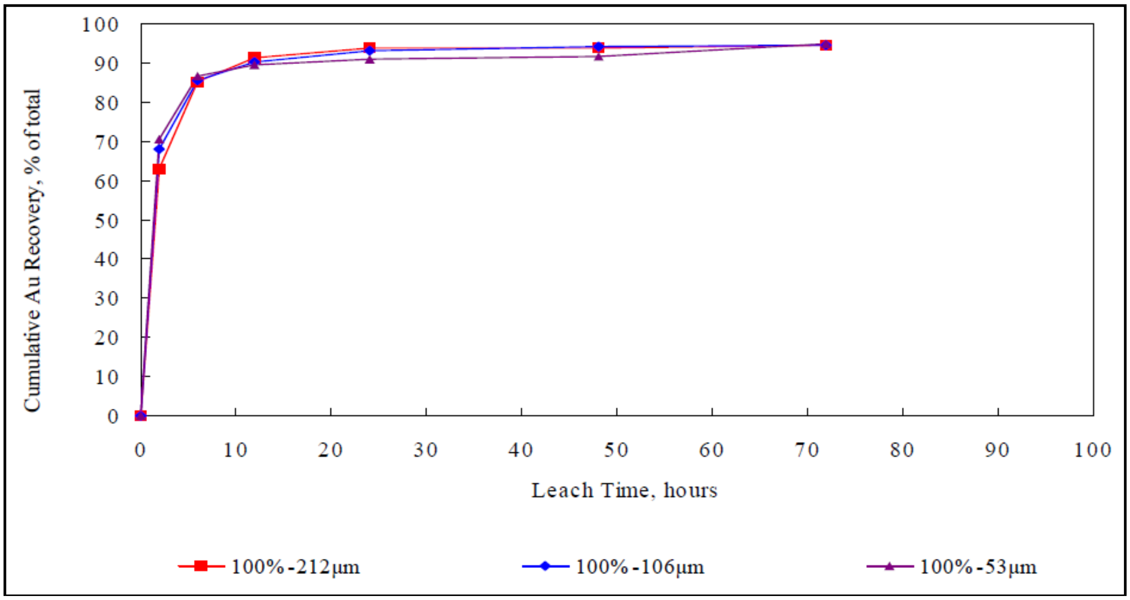


Figure 13.3.2 McClelland Gold Leach Profile for Medium Grade Oxide Composite (MGO)

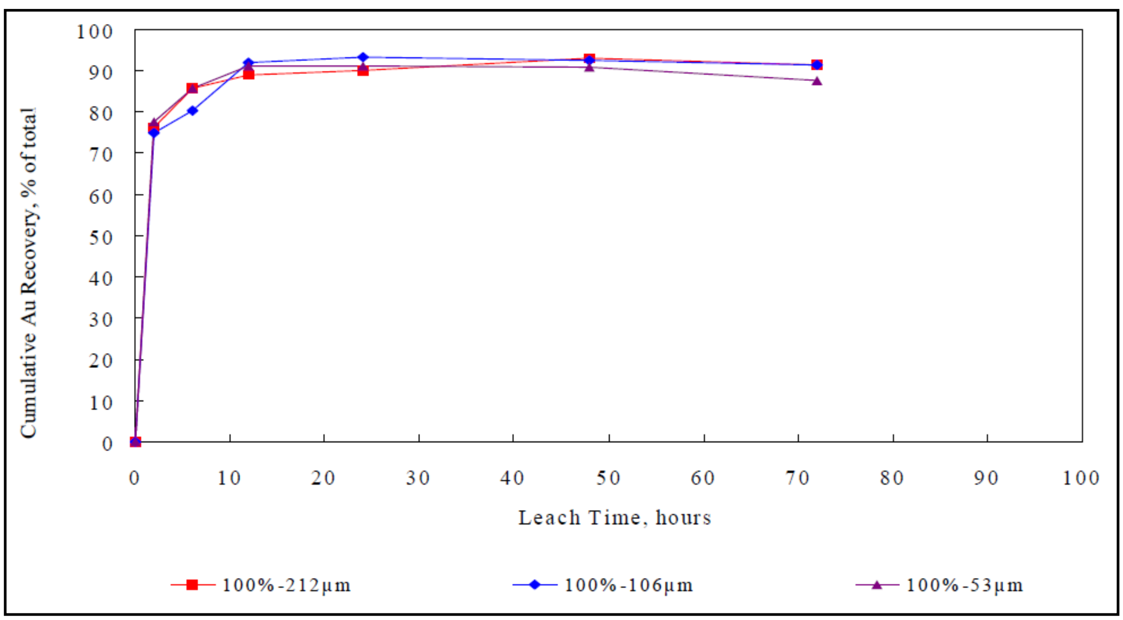


Figure 13.3.3 McClelland Gold Leach Profiles for High Grade Sulphide Composite (HGS)

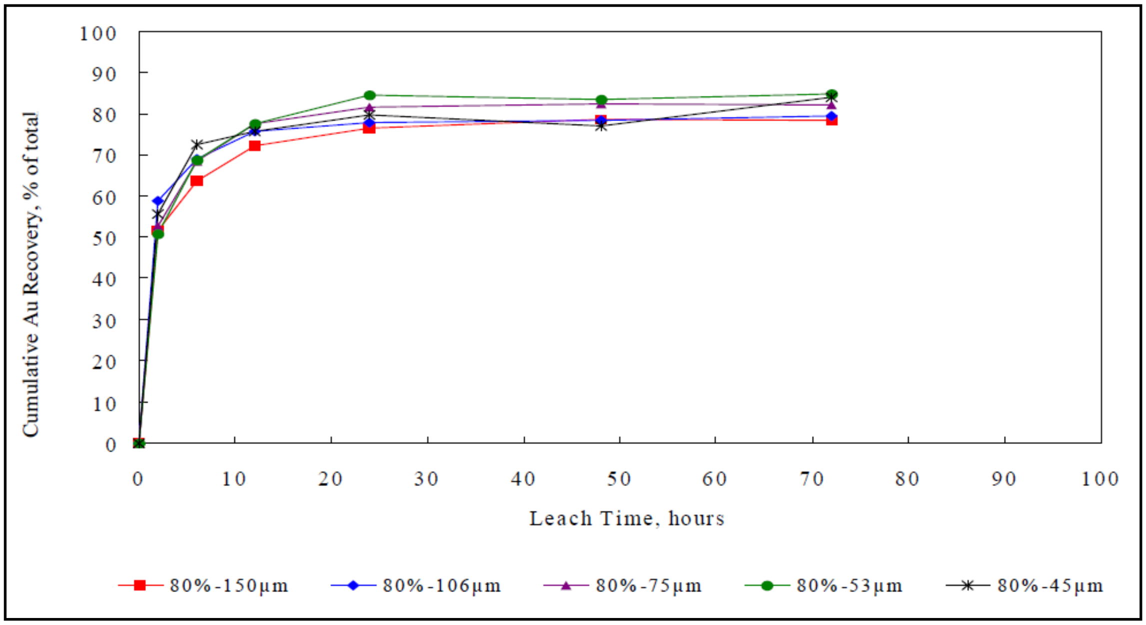
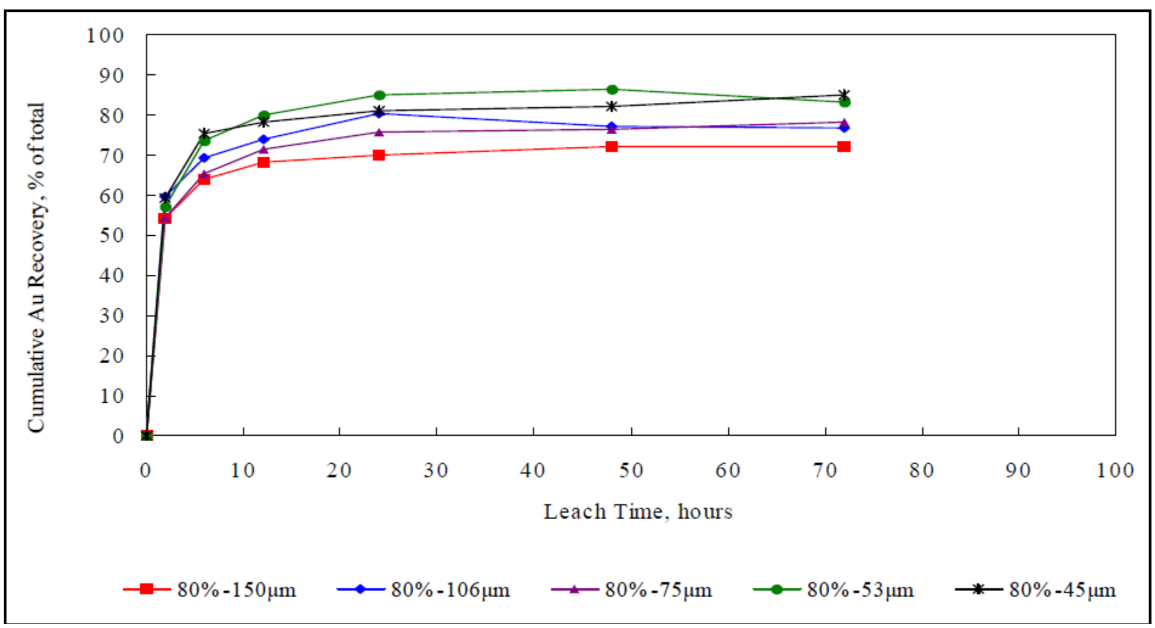


Figure 13.3.4 McClelland Gold Leach Profiles for Medium Grade Sulphide Composite (MGS)



Additional bottle roll tests were conducted on all the composites to optimize the cyanide dosing rates. The oxide composites were tested at three grind sizes (-212 µm, -106 µm, and -53 µm) and four NaCN concentrations (0.5, 1.0, 1.5, and 2.0 g/L). The sulphide composites were tested at two grind sizes (-75 µm and 53 µm) and at the same four NaCN concentrations as the oxides. The results are summarized in Table 13.3.6.

Table 13.3.6 McClelland Grind Size and Cyanide Concentration Optimization Tests

Ore Type	Grind Size	g/L NaCN	No. of Tests	Gold Extraction %	g/t Au			Reagents, kg/t	
					Extracted	Tail	Calc'd Head	NaCN	Lime Added[2]
Oxide	P_{100}-212 µm	0.5	2	93.4	1.18	0.07	1.25	<0.07	3.4
		1.0	4	93.7	1.26	0.07	1.33	0.15	3.2
		1.5	2	94.7	1.24	0.06	1.29	0.17	3.3
		2.0	2	91.8	1.23	0.08	1.31	0.21	3.2
Oxide	P_{100}-150 µm	0.5	2	94.1	1.20	0.06	1.26	0.09	3.5
		1.0	2	89.6	1.10	0.09	1.19	0.16	3.6
		1.5	2	94.7	1.23	0.06	1.29	0.19	3.6
		2.0	2	94.0	1.18	0.07	1.24	0.18	3.4
Oxide	P_{100}-106	1.0	2	93.0	1.19	0.08	1.27	0.08	3.1
Oxide	P_{100}-75 µm	0.5	2	94.1	1.13	0.07	1.20	0.09	4.0
		1.0	2	94.4	1.11	0.06	1.16	0.09	4.0
		1.5	2	95.1	1.09	0.05	1.14	0.17	3.7
		2.0	2	95.5	1.14	0.05	1.18	0.16	3.6
Oxide[1]	P_{100}-53 µm	1.0	2	94.1	1.13	0.07	1.20	0.09	4.0
Sulphide[1]	P_{80}-150µm	1.0	2	75.3	1.06	0.32	1.38	0.47	1.0
Sulphide[1]	P_{80}-106µm	1.0	2	78.2	1.05	0.29	1.34	0.39	1.0
Sulphide	P_{100}-75µm	0.5	2	80.0	1.20	0.28	1.48	0.23	1.4
		1.0	4	79.6	1.10	0.27	1.37	0.5	1.1
		1.5	2	83.4	1.41	0.28	1.68	1.13	0.9
		2.0	2	79.4	1.11	0.27	1.38	1.88	0.8
Sulphide	P_{100}-53µm	0.5	2	82.3	1.16	0.25	1.40	0.47	1.5
		1.0	4	83.3	1.19	0.23	1.42	1.28	1.4
		1.5	2	80.8	1.14	0.25	1.39	1.52	1.0
		2.0	2	81.0	1.13	0.25	1.38	2.62	0.8
Sulphide[1]	P_{80}-45µm	1.0	2	84.4	1.28	0.24	1.52	2.36	1.7

1 Results from Table 13.3.5 were also included for the sake of completeness and comparison.

2 Testwork report does not specify lime purity or whether CaO or Ca(OH)2

The results showed similar gold extraction percentage for oxide composites at the four cyanide concentrations, with an average of 94% gold extraction overall. It is expected that gold recoveries will not increase significantly with decreasing feed size or with increasing cyanide concentration. It was recommended that the near optimum conditions for oxide ore type samples were -150 µm feed size and 0.5 g/L NaCN.

For the sulphide composites, the results showed varying gold extraction percentages at the four cyanide concentrations, but no significant improvement beyond 1.0 g/L NaCN. Average cyanide and lime consumptions for the sulphide samples increased with decreasing feed size. It was recommended that the near optimum conditions for hard rock ore type samples were P_{80} of 53 µm feed size and 1.0 g/L NaCN.

The combined gravity concentration and gravity tails cyanidation results are presented in Table 13.3.7.

Table 13.3.7 McClelland Combined Gravity Concentration/Gravity Tails Cyanidation Tests

Sample	Feed Size	Gold Extraction %					Gold Grade, g/t Au					Reagents	
		Grav. Conc.	CN Leach	Combined	CN Tails	Total	Grav. Conc.	CN Leach	Tail	Calc'd Head	Assayed Head	NaCN	Lime Added
HGS	F_{80} -75 µm	30.9	54.2	85.1	14.9	100	0.61	1.07	0.30	1.98	1.76	0.30	1.1
HGS	F_{80} -53 µm	22.3	63.5	85.8	14.2	100	0.41	1.16	0.26	1.83	1.76	0.52	1.3
MGS	F_{80} -75 µm	14.0	69.5	83.5	16.5	100	0.10	0.52	0.12	0.74	0.80	0.34	1.1
MGS	F_{80} -53 µm	18.2	68.4	86.6	13.4	100	0.14	0.53	0.10	0.77	0.80	0.17	1.1
HGO	F_{100} -212 µm	6.2	89.7	95.9	4.1	100	0.10	1.5	0.07	1.67	2.50	0.04	2.9
HGO	F_{100} -150 µm	12.0	84.9	96.9	3.1	100	0.22	1.54	0.06	1.82	2.50	0.04	2.9
HGO	F_{100} -75 µm	12.8	82.8	95.6	4.4	100	0.21	1.38	0.07	1.66	2.50	0.07	3.1
MGO	F_{100} -212 µm	7.7	84.1	91.8	8.2	100	0.05	0.54	0.05	0.64	0.59	0.18	3.2
MGO	F_{100} -150 µm	10.2	82.6	92.8	7.2	100	0.07	0.55	0.05	0.67	0.59	0.07	4.4
MGO	F_{100} -75 µm	3.2	90.0	93.2	6.8	100	0.02	0.58	0.04	0.64	0.59	0.07	3.6

The oxide composites responded very well to combined gravity concentration treatment followed by gravity tails cyanidation at all three feed sizes evaluated, -212 µm, -150 µm, and -75 µm. Although, when compared to whole ore cyanidation tests at the same feed sizes, no significant improvement to the overall gold extraction was observed for the oxide composites.

The sulphide composites responded moderately well to the combined gravity concentration treatment followed by gravity tails cyanidation at the two feed sizes evaluated, P_{80} of 75 µm and P_{80} of 53 µm. When compared to whole ore cyanidation at the same feed sizes, the sulphide composites had relatively higher overall gold extraction between 2% to 5%. The results suggest a potential benefit to using gravity concentration treatment prior to a leaching circuit for the Bomboré hard rock ore.

Variability testwork for a milling / cyanidation process was conducted on 33 oxide, 5 transition, and 33 sulphide samples under the selected leach conditions deemed as near optimum. A summary of the results is presented in Table 13.3.8.

Table 13.3.8 McClelland Variability Test Summary for Milling / Cyanidation

Test Count	Gold Extraction %			Head Grade g/t Au			NaCN Consumption kg/t			Lime Added kg/t		
	Avg.	Min.	Max.	Avg.	Min.	Max.	Avg.	Min.	Max.	Avg.	Min.	Max.
Oxide Ore Type, P_{80}-150 µm, 0.5 g/L NaCN												
33	90.4	77.1	97.3	1.25	0.38	7.15	0.21	0.07	0.43	2.2	0.7	5.6
Transition Ore Type, P_{80}-150 µm, 0.5 g/L NaCN												
5	91.5	87.5	95.7	1.02	0.66	2.01	0.10	<0.07	0.13	2.1	1	4.1
Transition Ore Type, P_{80}-53 µm, 1 g/L NaCN												
5	92.9	88.9	95.8	1.06	0.69	1.84	0.1	<0.07	0.1	3	1.2	5.2
Hard Rock Ore Type, P_{80}-53 µm, 1 g/L NaCN												
33	82.9	39.9	96.4	1.44	0.39	4.22	1.14	0.3	5.11	0.8	0.6	1.0

The results showed that both oxide and transition variability samples were readily amenable to whole ore milling and cyanidation treatment, with average gold extraction of 90.4% and 92.2%, respectively. Most of the sulphide samples were also amenable to whole ore milling and cyanidation with an average gold extraction of 82.9%. Only one sulphide sample (1027558) performed poorly with a gold extraction of 39.9%. It was also observed that the gold leach rates were rapid with the extraction essentially complete in 24 hours. Only one transition and four sulphide samples out of the 71 samples exhibited slower leach rates. Cyanide consumptions were low for oxide and transition samples (<0.07 kg/t to 0.43 kg/t) but varied from low to high for the sulphide samples (0.3 kg/t to 5.11 kg/t). Lime additions varied substantially for the oxide and transition samples (0.7 to 5.6 kg/t) and were consistently low for the sulphide samples (0.6 kg/t to 1 kg/t).

CIL and CIP carbon adsorption capacity tests were also conducted, and the results are presented in Figure 13.3.5 to Figure 13.3.8. However, no conclusions for carbon adsorption kinetics or equilibrium isotherms were possible from these results as these tests were conducted at too long of a period for determining the kinetics and too short of a period for determining the equilibrium isotherms.

Figure 13.3.5 McClelland Carbon-In-Leach Adsorption Capacity for Oxides

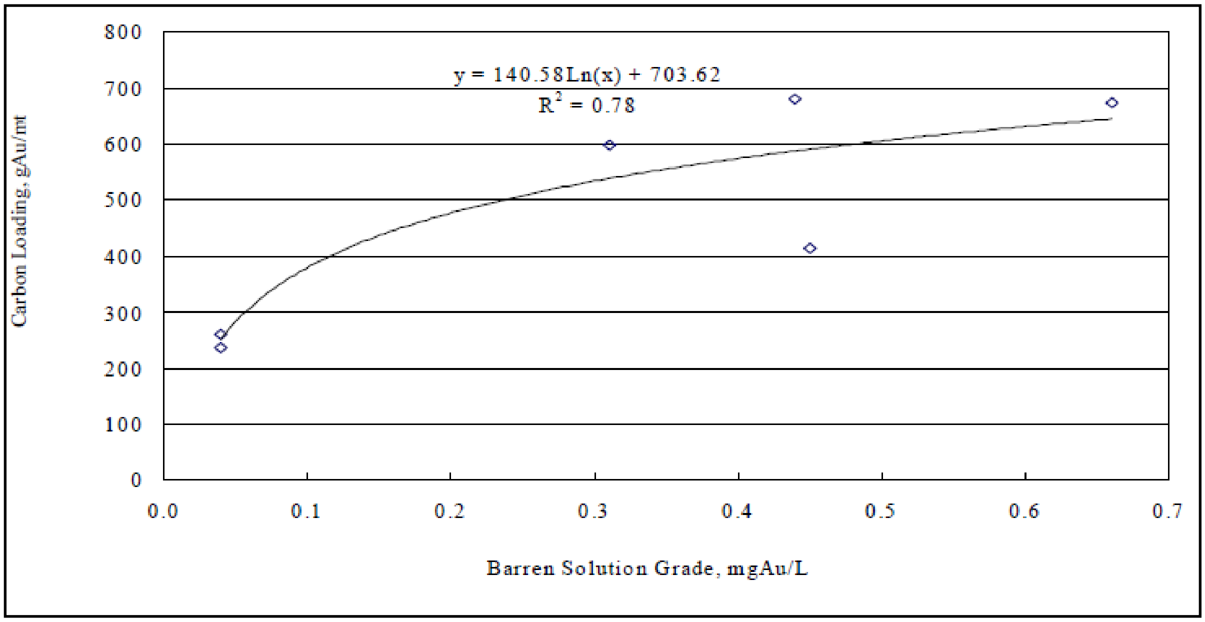


Figure 13.3.6 McClelland Carbon-In-Leach Adsorption Capacity for Sulphides

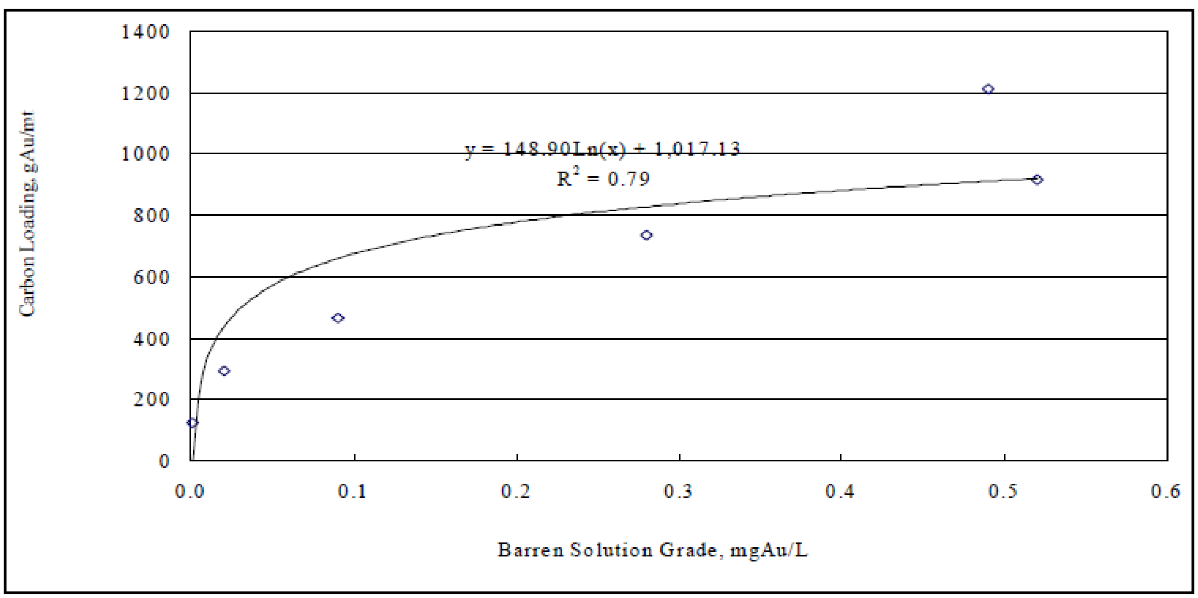


Figure 13.3.7 McClelland Carbon-In-Pulp Adsorption Capacity for Oxides

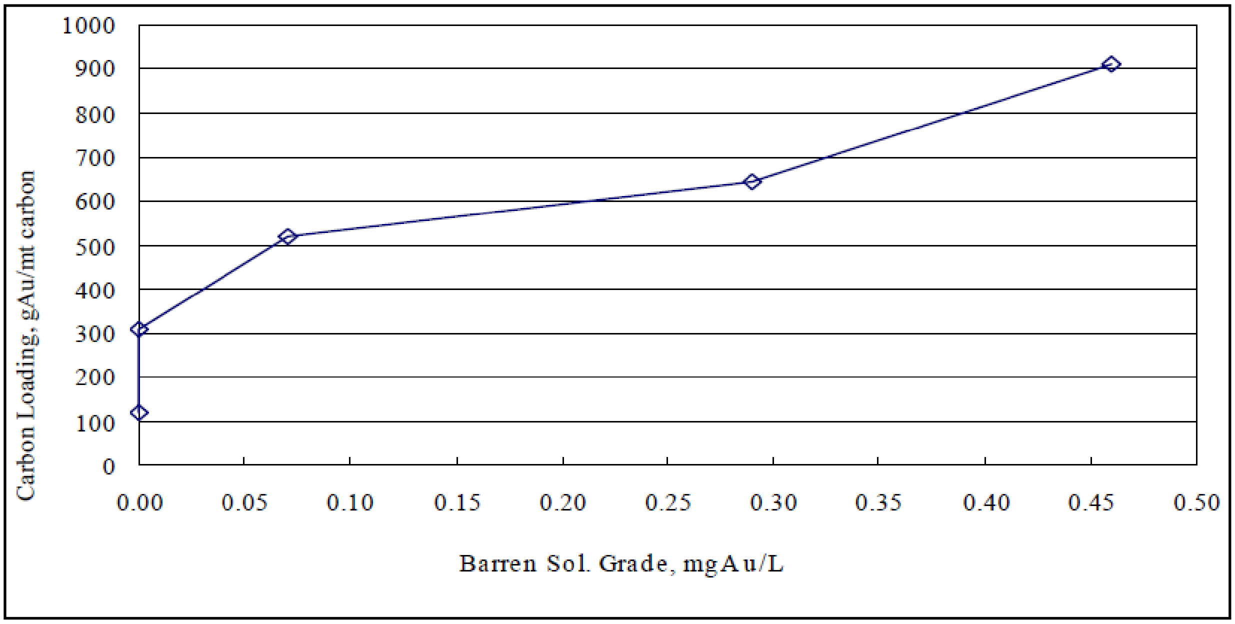
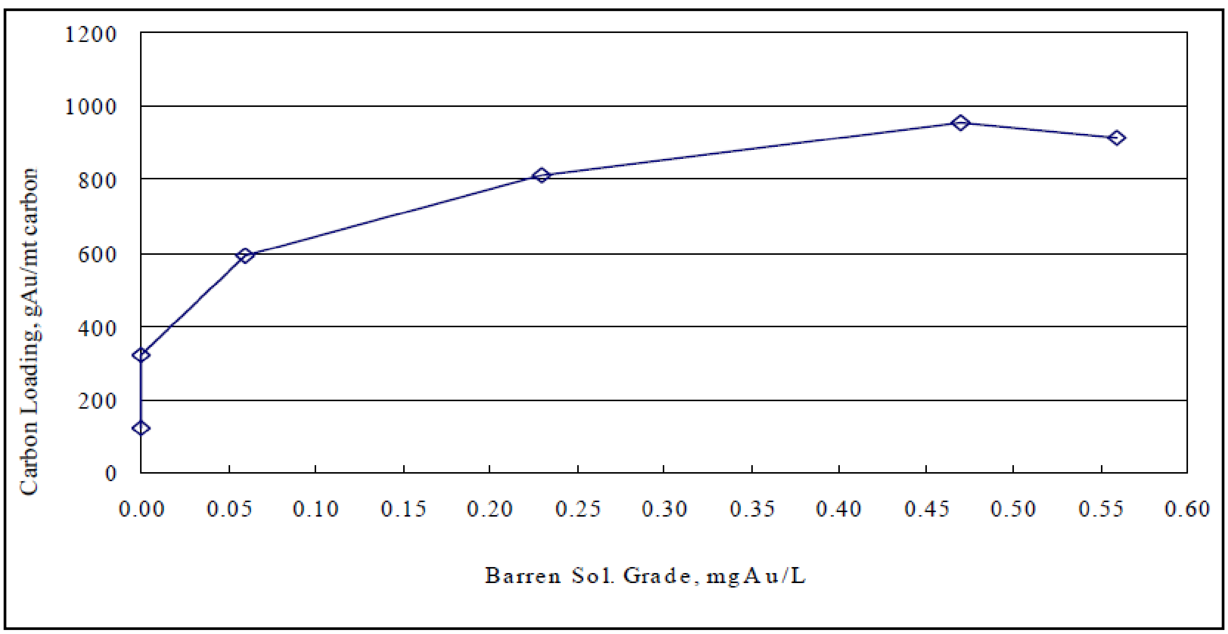


Figure 13.3.8 McClelland Carbon-In-Pulp Adsorption Capacity for Sulphides



Standard cyanide destruction testing was conducted on both oxide and sulphide slurries using SO_2/Air treatment with SMBS used as the source of SO_2. A summary of the results is shown in Table 13.3.9.

Table 13.3.9 McClelland SO2/Air Treatments Results for P_{80} of 150 µm Feeds

Sample	Stream	Retention Minutes	Run #	Sol'n Analyses, mg/L		Molar Ratio, SO_2/CN		g/g CN_{WAD} in Feed			g SO_2/g CN_{WAD} Treated	kg/t		SMBS Utilization Efficiency %
				Slurry pH	CN_{WAD}	Target	Actual	SMBS	SO_2	$CuSO_4$		SMBS	Lime	
Oxide Master	Feed	95.1	1st	10.2	273*									
	Effluent			8.5	0.024	3:1	2.3:1	6.86	5.57	0.58	5.57	3.51	8.93	44.2
	Feed	95.1	2nd	8.5	238									
	Effluent				0.036	3:1	2.7:1	8.12	6.59	0.69	6.60	3.51	5.95	37.3
	Feed	95.1	3rd	8.6	178									
	Effluent				0.034	3:1	4.3:1	13.08	10.63	1.11	10.63	3.51	5.36	23.2
Sulphide Master	Feed	95.1	1st	10.6	200									
	Effluent			8.5	0.28	3:1	4.9:1	14.87	12.08	0.83	12.09	5.31	8.12	20.4
	Feed	95.1	2nd	8.6	158									
	Effluent				0.42	3:1	6.0:1	18.3	14.86	1.03	14.9	5.31	14.88	16.5

*Calculated based on dilution of final preg WAD concentration

Comminution testwork in this program consisted of abrasion index tests on <12.5 mm crushed size and ball mill grindability tests at 150 µm closed screens. The results are presented in Table 13.3.10.

Table 13.3.10 McClelland Abrasion Index and Ball Mill Grindability Tests

Sample	Abrasion Index	BWi (kWh/t)
HGS	0.1035	15.22
MGS	0.0746	15.00
MGO	0.0666	5.71
HGO	Not Tested	4.13

Conclusions from the McClelland program were as follows:

- Gravity concentration prior to cyanidation could benefit gold extraction for sulphide composites.
- Oxide composites did not respond well to flotation treatment, but sulphide composites did.
- The optimum feed size for whole ore cyanidation of oxide composites was -212 µm to 150 µm.
- The optimum feed size for whole ore cyanidation of sulphide composites was a P_{80} of 53 µm.
- The optimum cyanide concentration for leaching of oxide composites was 0.5 g/L NaCN.

- The optimum cyanide concentration for leaching of sulphide composites was 0.5 to 1.0 g/L NaCN.
- Gold extraction rates generally were rapid and essentially completed in 24 hours for most samples.
- Almost all variability samples were amenable to whole ore cyanidation at optimized conditions.
- Conventional SO2/air tails slurry treatment was effective in decreasing WAD CN to acceptable levels.

13.3.5 SGS 2013 Testwork Program

Twenty-six (26) sulphide composite samples of various lithologies were submitted to SGS for comminution testing. A statistical analysis of the results is presented in Table 13.3.11.

Table 13.3.11 SGS 2013 Comminution Test Statistics for Sulphides

Statistics	JK Values		Rel. Density**		Work Indices (kWh/t)		
	A x b (SMC)	DWi kWh/m^3	CWi [50]	SMC [21]	CWi	RWi	BWi
Results Available	26	26	4	26	4	5	26
Average	41.5	7.6	2.72	2.81	10.7	15.5	15.2
Standard Deviation	13.2	2.8	0.26	0.11	2.8	1.0	2.0
Rel. S.D. (%)	32	37	10	4	26	6	13
Min	61.7	4.4	2.48	2.7	6.8	14.0	10.8
10th Percentile	57.0	4.8	-	2.72	-	-	12.8
25th Percentile	51.6	5.4	-	2.74	-	-	14.0
Median	45.7	6.2	2.65	2.77	11.4	15.7	15.3
75th Percentile	27.3	10.1	-	2.82	-	-	16.5
90th Percentile	24.9	11.3	-	2.97	-	-	17.5
Max	24.0	13.0	3.09	3.18	13.3	16.3	19.0

* Min and Max refer to Softest and Hardest for the tests.

** The density values were measured by a water displacement technique. The number in parentheses refers to the geometrical mean (expressed in mm) of the rocks used for the measurement.

The SMC 'A x b' values ranged from moderately soft to very hard. The other comminution indices, namely the CWi, RWi and BWi, respectively, demonstrated less variability and were placed in the soft to moderately hard range of the SGS database. Based on the bimodal ore hardness observed in the coarse size fraction, it was recommended by SGS to evaluate the test results with respect to a mine plan and by depth in order to better understand the ore hardness variability within the deposit.

13.3.6 COREM 2013 Testwork Program

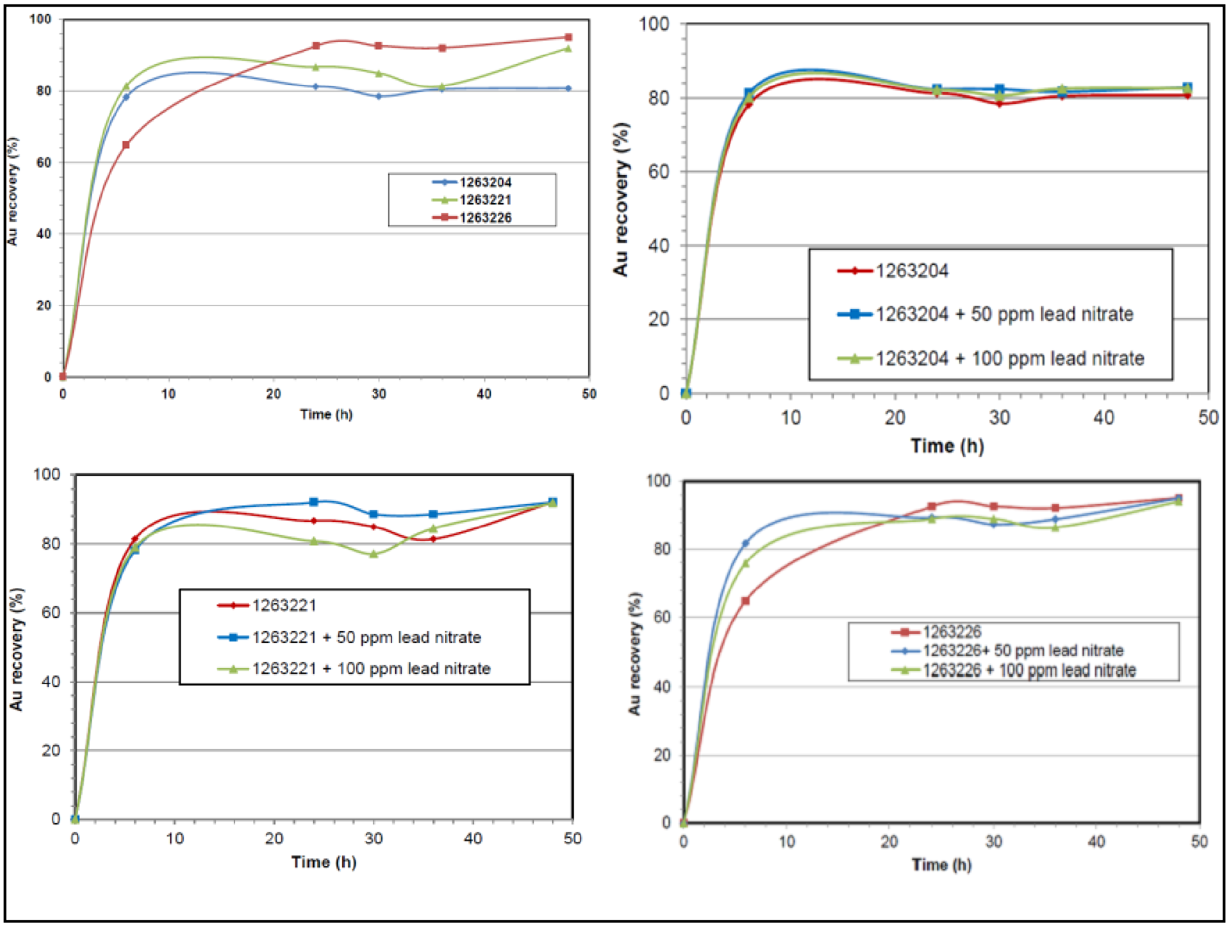
Five of the 26 samples from SGS program were sent to COREM for further testing to assess the impact of lead nitrate on cyanidation of pyrrhotite-rich samples. Three of the five samples were determined to be pyrrhotite-rich, hence, chemical analyses for sulphur and arsenic were performed to estimate the levels of sulphide mineral phases for each sample. The results are shown in Table 13.3.12.

Table 13.3.12 COREM Mineralogical Analysis of High Pyrrhotite Samples

Analysis		Sample #1263204 (49 min. grind)	Sample #1263221 (49 min. grind)	Sample #1263226 (49 min. grind)
Chem.	As (%)	0.17	<0.01	0.26
	S (%)	1.20	2.53	1.17
Modal	Pyrite (%)	0.20	4.60	Not Detected
	Pyrrhotite (%)	2.90	0.20	2.85
	Arsenopyrite (%)	0.40	<0.02	0.57

The leach curves for the pyrrhotite-rich samples are presented in Figure 13.3.9.

Figure 13.3.9 COREM Leach Curves for Pyrrhotite-Rich Samples and Impact of Lead Nitrate

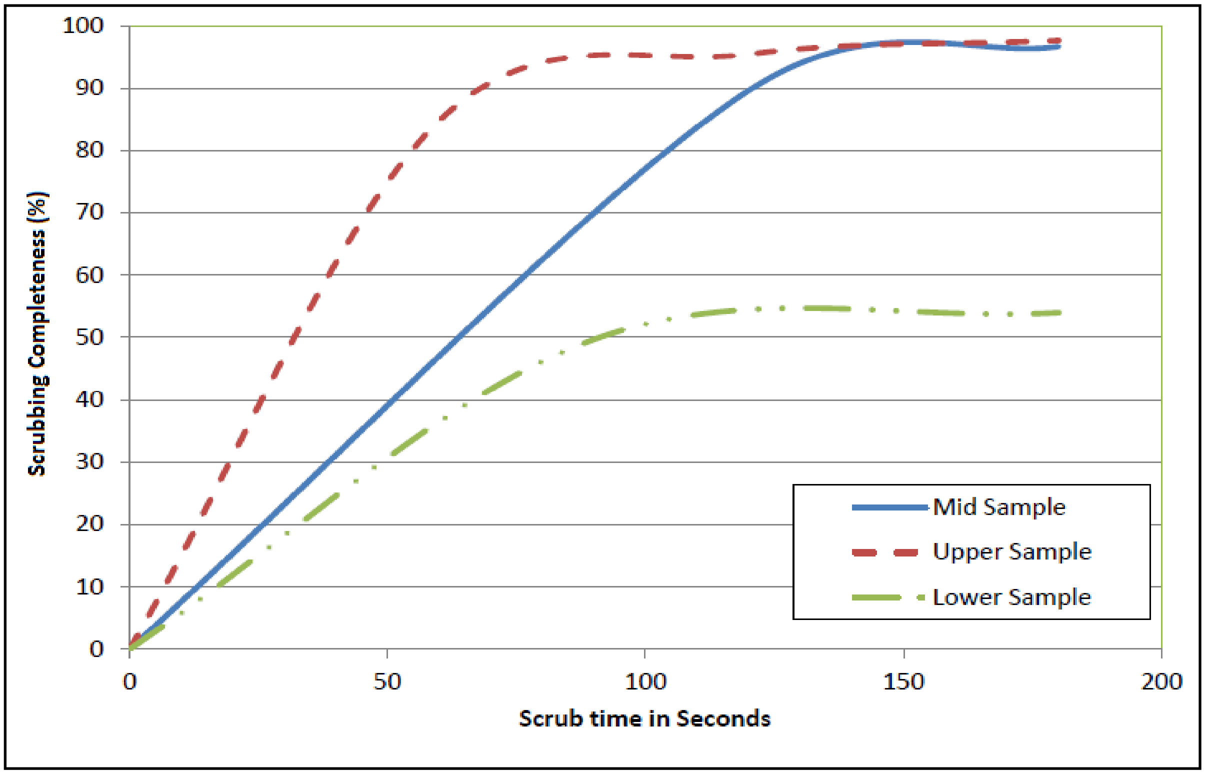


A comparative analysis of gold dissolution profiles for samples containing pyrrhotite showed that the pyrrhotite content of the tested samples was not the only factor controlling the gold recovery. Sample #1263226 showed the highest gold recovery (92.6%) despite its high content of pyrrhotite (2.85%). The addition of 50 ppm lead nitrate enhanced the overall gold extraction for sample #1263204 by about 2%. The use of lead nitrate significantly improved the gold extraction for sample #1263221 from 88.6% to 92% after 24 hours of cyanidation, however, no improvement after 48 hours. The testwork for sample #1263221 may not be as reliable because the results fluctuated significantly throughout the leach period. Some improvements on gold extraction were observed for sample #1263226, but only during the first six hours of cyanidation. Increasing lead nitrate concentration to 100 ppm was counterproductive for all the tested samples.

13.3.7 Met-Solve 2013 / 2014 Testwork Program

Oxide samples were submitted to Met-Solve for scrubbing testwork. The estimated scrubbing completeness for <75 mm size at 50% pulp density as a function of time for different ore zones (upper, middle, and lower) is presented in Figure 13.3.10.

Figure 13.3.10 2013 Met-Solve Scrubbing Test - Scrubbing Completeness % Versus Time



For scrubbing to be complete, a minimum of 95% completeness is required. As seen in Figure 13.3.10, the upper and middle oxide samples met this requirement, however, the lower oxide sample failed to scrub satisfactorily.

Note that material from the middle zone initially did not scrub successfully at 50% and 33% pulp density and it was only after pre-shredding the material to <75 mm that the scrubbing test was successful. Cyanide leach tests were also conducted on the screened undersize at -150 µm from the upper, middle and lower ore zones. The leach curves at different cyanide concentrations are shown in Figure 13.3.11. The cyanide and lime consumptions are shown in Table 13.3.13.

Figure 13.3.11 Met-Solve Leach Curves at Different Cyanide Concentrations

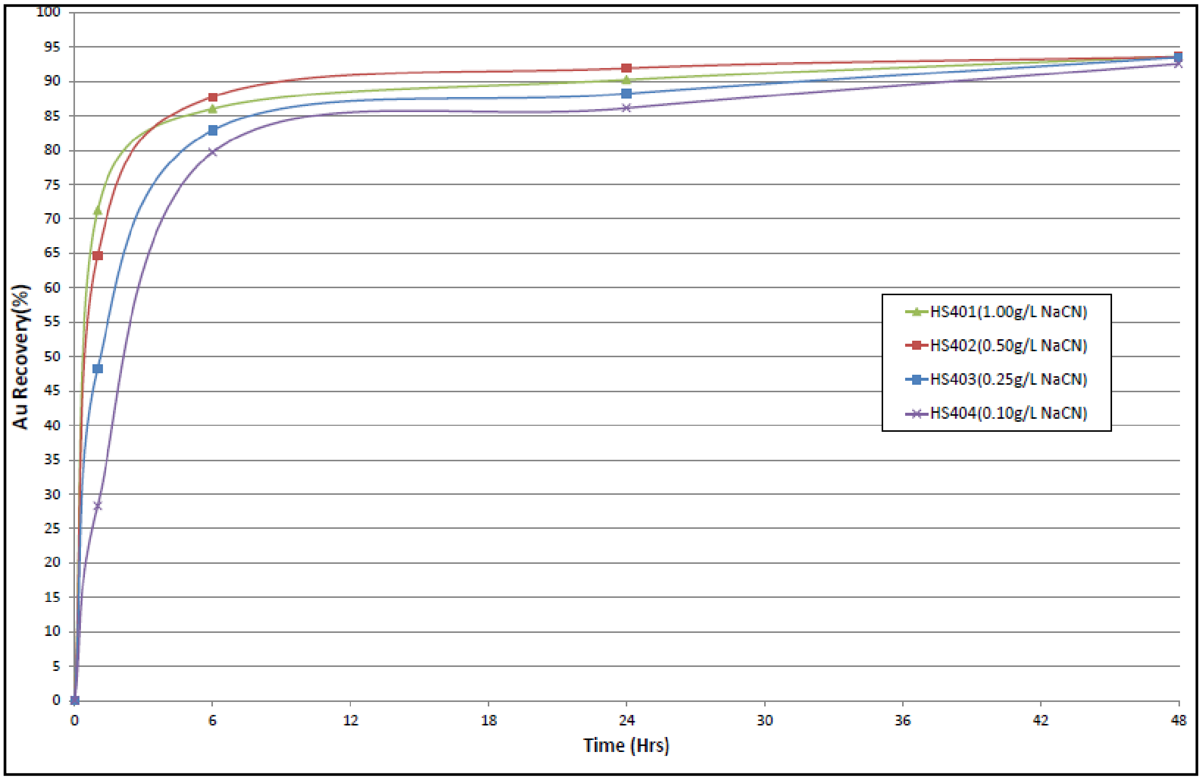


Table 13.3.13 Met-Solve Cyanide Leach Results on Screened U/S (-150 µm)

Test No.	Pulp Density %	g/L NaCN Conc.	Au Extraction %				Residue Assay g/t Au	Head Grade		Reagents Req'd (kg/t)	
			1 hr	6 hrs	24 hrs	48 hrs		Calc'd	Direct	NaCN Consumed	CaO Added
HS401	33	1	71	86	90	94	0.05	0.83	0.76	0.4	1.31
HS402	33	0.5	65	88	92	94	0.06	0.85		0.33	1.74
HS403	33	0.25	48	83	88	93	0.05	0.77		0.08	1.76
HS404	33	0.1	28	80	86	93	0.06	0.8		0.04	1.59

The cyanidation results showed that it was possible to achieve 92.6% gold extraction at cyanide concentration of 0.1 g/L NaCN, hence yielding NaCN consumption of only 0.04 kg/t for this sample. Increasing the cyanide concentration improved leach kinetics within the first six hours, but had only minimal overall gold extraction improvement.

13.3.8 SGS 2016 Testwork Program

The SGS 2016 testwork program was conducted on one fresh rock sample identified as P17S. The program included Bond ball mill work index, gravity separation, cyanidation, and flotation testwork.

The P17S sample had a head grade of 3.13 g/t Au, 0.81% $S^{=}$, and 0.10% total carbonaceous matter. The main sulphide minerals were identified as pyrrhotite (1.8%) and arsenopyrite (0.7%). The material was identified as having medium hardness with a Bond work index of 14.2 kWh/t.

Different flowsheet options were tested at a grind size P_{80} of 74 µm. In all options with flotation, the concentrate was reground before leaching. One option included a regrind of the flotation concentrate to P_{80} of 26 µm. The overall metallurgical results for the different flowsheet options are presented in Table 13.3.14.

Table 13.3.14 SGS 2016 Overall Metallurgical Results for Flowsheet Options

Flowsheet Options	Overall Au Extraction %						Final Tail g/t Au	Head g/t Au
	Gravity	WO CN	Tails CN	Tails Flot.	Conc. CN	Combined		
Whole Ore Cyanidation		94.6				94.6	0.17	3.13
Gravity Separation	62.1					62.1	1.18	3.13
Gravity Separation + Grav Tails CN (with pre-aeration)	62.1		33.9			96.0	0.13	3.13
Gravity Separation + Flotation	62.1			34.2		96.3	0.13	3.13
Gravity Sep'n + Flotation + Conc CN (no regrind)	62.1				29.8	91.9	0.23	3.13
Gravity Sep'n + Flotation + Conc CN (with regrind)	62.1				32.2	94.3	0.18	3.13

13.4 Oxide Ore Testwork 2017 to 2019

In 2017, Orezone approached Lycopodium to conduct a feasibility study for a CIL process on the oxide and upper transition ores at Bomboré. Orezone also contracted Soutex to carry out a gap analysis on the metallurgical testwork programs and results. As a result of the gap analysis, Orezone selected a number of oxide and transition samples to represent different lithologies and grades and submitted these to SGS Quebec for metallurgical testwork. Testwork was then carried out over 2017 to 2019, and included comminution, thickening, and leach testwork as described in the following sections.

13.4.1 SGS 2017 / 2018 Testwork Program

A total of 28 low grade and medium grade samples were provided to SGS for characterization of grindability and gold recovery by cyanidation. A summary of the results is presented in Table 13.4.1.

Table 13.4.1 SGS 2017/2018 Grindability and Gold Recovery Characterization Results

Sample Name	Au - Head g/t	LeachWell Au ppm	Calc Au Head g/t	AS Received (%)		BWI (kWh/t)		Au CN Test	
				Moisture	+3.35mm	Direct	O'All	P_{80} µm	Au Rec, %
Overall Avg.	**0.54**	**0.48**	**0.51**	**0.9**	**85.6**	**8.3**	**7.7**	**87**	**82.1**
MG Avg.	0.70	0.60	0.65	0.9	84.7	8.3	7.7	91	83.9
LG Avg.	0.34	0.31	0.33	1.0	86.7	8.4	7.7	82	79.8
MG Tr U	0.74	0.62	0.69	0.9	87.7	7.4	7.4	92	84.8
LG Tr U	0.37	0.33	0.34	1.0	91.4	8.3	8.3	81	81.6
MG Tr L	0.65	0.58	0.60	0.6	90.9	9.2	9.2	88	82.6
LG Tr L	0.29	0.29	0.35	0.8	92.6	9.1	9.1	84	75.9
MG Ox	0.71	0.61	0.65	1.4	52.8	7.9	3.3	96	85.5
LG Ox	0.36	0.32	0.29	1.3	60.4	6.9	3.0	78	84.8
Oxide	0.54	0.46	0.47	1.3	56.6	7.4	3.2	87	85.2
I1C	0.54	0.45	0.49	0.3	94.0	13.0	13.0	103	73.8
I2	0.53	0.49	0.53	0.7	90.1	7.8	7.8	80	86.1
MI3	0.54	0.47	0.54	1.2	86.7	7.9	7.9	90	80.5
S3	0.54	0.50	0.51	0.8	91.3	6.8	6.8	82	85.9
S4	0.58	0.48	0.52	1.0	91.3	8.2	8.2	82	80.4

Oxide, I2, and S3 lithologies showed the higher gold cyanidation recoveries in the range of 85% Au, while the other three lithologies (I1C, MI3, and S4) showed gold recoveries ranging from 73.8 to 80.5% Au. The low-grade samples generally generated lower gold recoveries.

13.4.2 Outotec 2018 Testwork Program

A composite sample of the oxide samples, from the SGS program above, was screened to remove the plus 125 µm fraction and tested by Outotec to examine the flocculant screening and dynamic thickening. Results indicated as the solids density in the thickener underflow increases, the thickener flux decreases, resulting in increasing thickener diameter size. Due to the saprolitic nature of the Bomboré ore, the maximum density of the thickener underflow is 52.1% at a low flux of 0.2 t/m^{2}h. For thickener underflow density of 48.8% solids, a flux of 0.60 t/m^{2}h can be used for sizing the thickener. For other results provided by Outotec under this program, refer to Outotec Memorandum No. 11282017-TQ1-TM-001-R0.

13.4.3 SGS 2019 Testwork Program – Carbon Kinetics Results

In 2019, SGS Lakefield conducted carbon adsorption kinetics and equilibrium isotherm testwork. The results were analyzed to determine the kinetic and equilibrium constants which were used in SGS CIL modelling for validation of the Project CIL design and elution plant selection.

A composite sample was formed using the remaining oxide and upper transition samples from the SGS 2017 / 2018 program. The make-up for this sample is shown in Table 13.4.2. The sample's oxide and upper transition ratio is similar to the oxide plant design ore blend of 85% oxide and 15% upper transition.

Table 13.4.2 SGS 2019 Testwork Composite Sample

Sample	Lithology	Oxidation	Mass (kg)
N-MG-Ox	Mix	Ox	22.0
N-MG-TrU-I1C	I1C	Tr_U	1.0
N-MG-TrU-I2	I2	Tr_U	2.0
N-MG-TrU-MI3	MI3	Tr_U	1.0
N-MG-TrU-S3	S3	Tr_U	0.5
N-MG-TrU-S4	S4	Tr_U	0.5
S-MG-Ox	Mix	Ox	11.0
S-MG-TrU-MI3	MI3	Tr_U	2.0
Total			**40.0**

Table 13.4.3 presents the various testwork conducted on the composite sample in order to study the leach and carbon adsorption kinetics, and equilibrium isotherm.

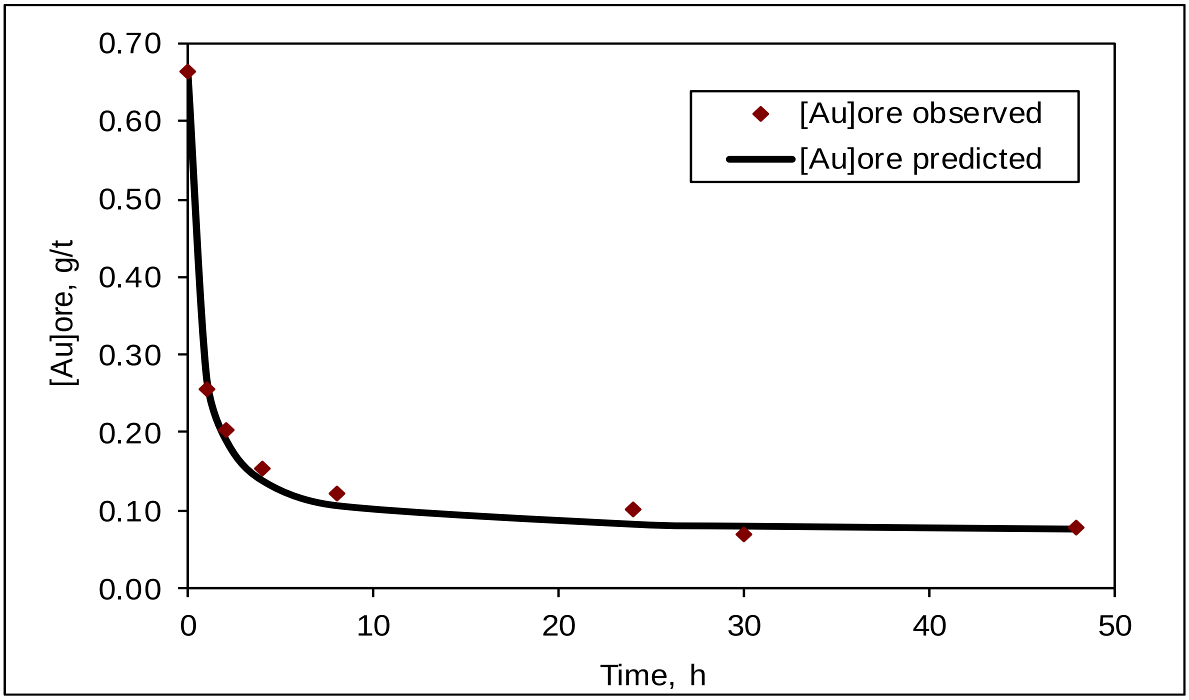
Table 13.4.3 SGS 2019 Testwork Outline

Test No.	Description
CN-1	Initial leach kinetic test (coarse grind)
CN-2	Bulk leach test to create sample for additional testwork
CN-3	Additional leach test using fresh sample and PLS from CN-2
CN-3 AK	Adsorption Kinetic test (using CN-3 pulp)
CN-4	Second leach kinetic test (finer grind)
CN-5 A to G	Equilibrium isotherm testwork using CN-2 pulp (spiked solution)
CN-6	Third leach kinetic test (finer gring)

Three leach kinetic tests were performed at grind sizes coarser than the design grind P_{80} of 125 µm (P_{80} of 278 µm, 196 µm, and 177 µm) due to difficulty calibrating the batch grinding mill. The available remaining sample from the 2017 / 2018 SGS testwork program had a lower head grade than the design head grade of 1 g/t Au. In order to achieve the 0.6 mg/L Au solution tenor required for the tests a double leaching technique was used where the PLS from Test No. CN-2 was used to leach the sample in Test No. CN-3.

The leach kinetic constant (ks) was calculated by SGS to be 3.19 for the composite sample. Figure 13.4.1 presents the gold leach kinetic curve.

Figure 13.4.1 SGS 2019 Gold Leach Kinetics



The leach pulp from Test No. CN-3 was then contacted with activated carbon to establish an adsorption profile and for generating the kinetic and equilibrium constants for CIL modelling. The SGS carbon loading kinetic and equilibrium loading models are presented in Figure 13.4.2 and Figure 13.4.3, respectively.

Figure 13.4.2 SGS 2019 Kinetics of Gold Cyanide Extraction by Carbon

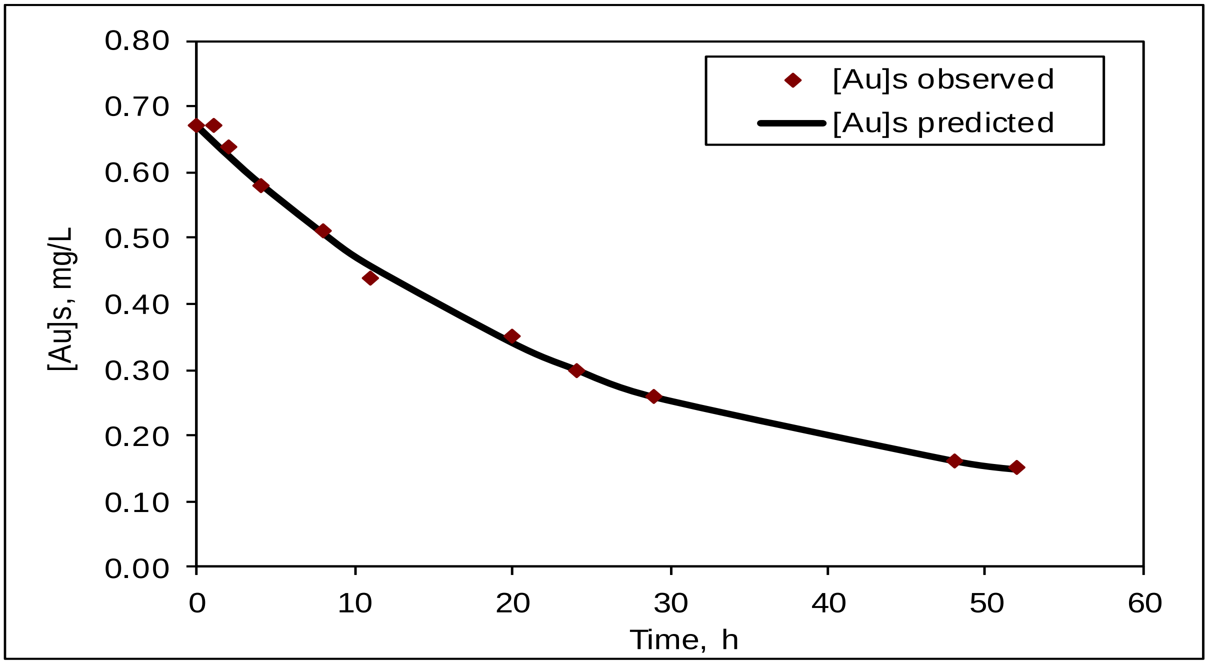


Figure 13.4.3 SGS 2019 Equilibrium Loading of Gold on Carbon

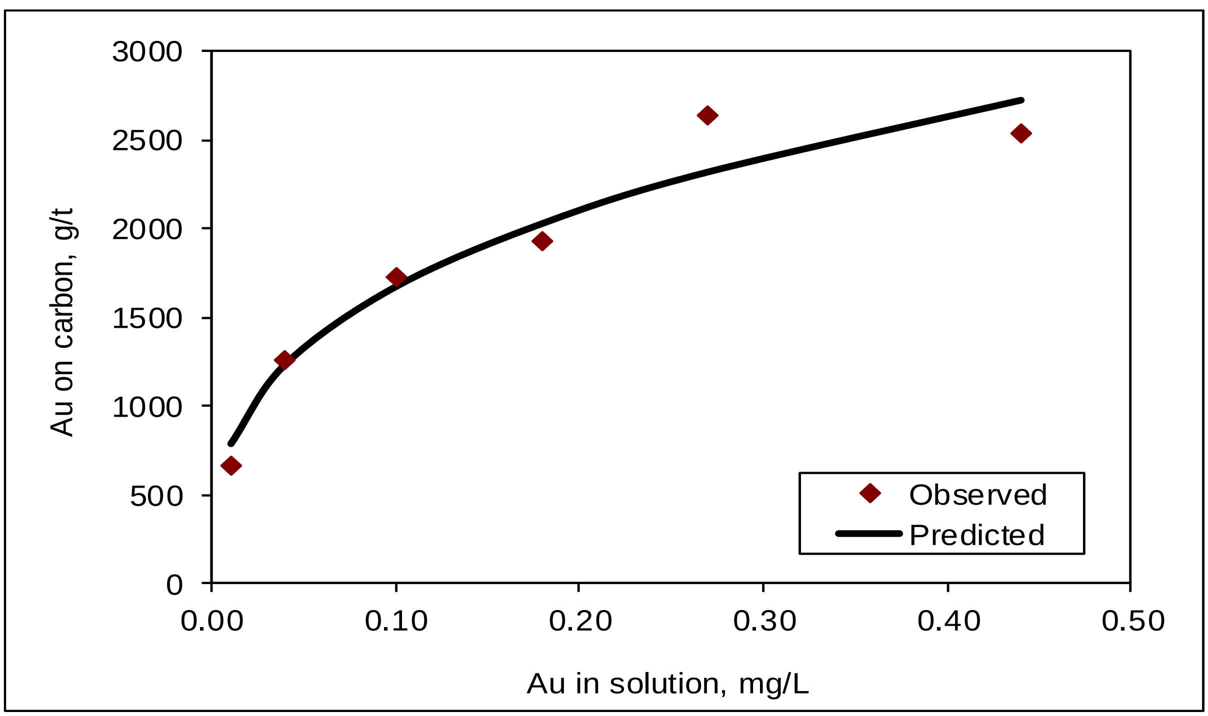


Table 13.4.4 presents the constants used in the SGS CIL modelling exercise.

Table 13.4.4 SGS 2019 Modelling Constants

Kinetic Constant (k), h-1	0.005
Equilibrium Constant (K), g/t	15592
Product of Equilibrium and Kinetic Constants (kK)	78

The product of the two constants (kK) provides a useful indication of how well the pulp will perform under CIL or CIP leach conditions. Typically, a kK value of <50 indicates a slow carbon adsorption process is expected, favoring CIP. The kK value for this composite sample is 78, which indicates satisfactory gold adsorption properties for CIL.

The SGS CIL modelling results indicated that the Project oxide plant will respond well to CIL processing. Table 13.4.5 presents all the modelling inputs for different CIL operating scenarios. The bold red highlighted values indicate the parameter that has been changed in each scenario. Table 13.4.6 presents all the modelling outputs.

Table 13.4.5 SGS 2019 Design Parameters for Multi-stage CIL Adsorption Circuit

Different Scenarios Inputs	1	2	3	4	5	6	7	8
Slurry feed rate (m^3/h)	1207	1207	1207	1207	1207	1207	1207	1207
Solids (t/h)	650	650	650	650	650	650	650	650
Solution (m^3/h)	975	975	975	975	975	975	975	975
Consider Leach after Carbon addition	Y	Y	Y	Y	Y	Y	Y	N
Gold on stripped carbon, g/t	50	50	50	50	50	**100**	**50**	50
Adsorption tank(s) size, m^3	3621	3621	3621	3621	3621	3621	3621	1207
Carbon frequency advance (% in 24 hours)	28%	33%	**18%**	**14%**	**55%**	**28%**	28%	33%
Leaching								
Au leached before Carbon addition	83.6%	83.6%	83.6%	83.6%	83.6%	83.6%	83.6%	91.7%
Leach time before Carbon addition (h)	3.0	3.0	3.0	3.0	3.0	3.0	3.0	**24**
Leach only total tankage (m^3)	3621	3621	3621	3621	3621	3621	3621	**28971**
Number of Leaching tanks	1	1	1	1	1	1	1	**1**
Volume of Leaching tanks (m^3)	3621	3621	3621	3621	3621	3621	3621	28971
CIP/CIL								
Model output kinetic constant (k)	0.005	0.005	0.005	0.005	0.005	0.005	**0.004**	**0.005**
Model output equilibrium constant (K)	15592	15592	15592	15592	15592	15592	**12474**	**15592**
Product of equilibrium and kinetic constants (	78	78	78	78	78	78	**50**	**78**
Number of stages	7	7	7	7	7	7	7	**7**
Total CIP/CIL volume (m^3)	25350	25350	25350	25350	25350	25350	25350	8450
Slurry residence time in each adsorption tank	3.0	3.0	3.0	3.0	3.0	3.0	3.0	**1.0**
Gold grade in residue (g/t)	0.083	0.083	0.083	0.083	0.083	0.083	0.083	0.083
Gold in final barren solution (mg/L)	0.006	0.006	0.005	0.005	0.011	0.009	0.010	0.004
Gold in loaded carbon (g/t)	1466	1230	1467	1468	1455	1508	1457	1471
Carbon residence time/stage (h)	87	72	130	174	43	87	87	72
Carbon Concentration (g/L pulp)	10	10	**15**	**20**	**5**	**10**	10	**25**
Equivalent transferred carbon unit flowrate (kg	417	500	417	417	417	417	417	417
Daily carbon transfer / batch elution capacity	10000	**12000**	10000	10000	10000	10000	10000	10000
Carbon Inventory per stage (kg)	36214	36214	54321	72429	18107	36214	36214	30179
Carbon inventory all stages (tons)	254	254	380	507	127	254	254	211
Gold Lock-Up on Carbon (kg)	102.8	85.8	144.6	186.2	59.9	115.1	116.5	81.5
CIP/CIL Gold recovery per day (g/day)	14158	14164	14174	14180	14048	14083	14067	14207
Overall Gold Leaching Efficiency	91.7%	91.7%	91.7%	91.7%	91.7%	91.7%	91.7%	91.7%
Overall Gold Adsorption Efficiency	98.9%	98.9%	99.0%	99.1%	98.1%	98.3%	98.2%	99.3%
Overall Gold Recovery	90.7%	90.7%	90.8%	90.8%	89.9%	90.1%	90.0%	91.1%
Au in loaded carbon / Au in feed	1466	1230	1467	1468	1455	1508	1457	1471
Upgrading ratio	2631	2208	2634	2635	2611	2707	2615	2406
Circuit filling time - slurry (days)	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.3
Ramp-up time (days) *	7.3	6.1	10.2	13.1	4.3	8.2	8.3	5.7

* Ramp-up time (days) = Gold lock-up (kg) / Gold Produced (kg/day)

Table 13.4.6 SGS 2019 CIL Modelling Circuit Profile Data

Interstage data Scenario	1	2	3	4	5	6	7	8
Gold in ore/stage residues (g/t)								
Feed head grade	1.000	1.000	1.000	1.000	1.000	1.000	1.000	1.000
Leach tank discharge	0.164	0.164	0.164	0.164	0.164	0.164	0.164	0.083
Adsorption stage 1 discharge	0.120	0.120	0.120	0.120	0.120	0.120	0.120	0.083
Adsorption stage 2 discharge	0.104	0.104	0.104	0.104	0.104	0.104	0.104	0.083
Adsorption stage 3 discharge	0.096	0.096	0.096	0.096	0.096	0.096	0.096	0.083
Adsorption stage 4 discharge	0.091	0.091	0.091	0.091	0.091	0.091	0.091	0.083
Adsorption stage 5 discharge	0.087	0.087	0.087	0.087	0.087	0.087	0.087	0.083
Adsorption stage 6 discharge	0.085	0.085	0.085	0.085	0.085	0.085	0.085	0.083
Adsorption stage 7 discharge	0.083	0.083	0.083	0.083	0.083	0.083	0.083	0.083
Gold on carbon (g/t)								
Adsorption stage 1 discharge	1466	1230	1467	1468	1455	1508	1457	1471
Adsorption stage 2 discharge	661	538	588	545	814	709	789	617
Adsorption stage 3 discharge	313	253	259	231	454	362	428	275
Adsorption stage 4 discharge	166	137	137	123	260	216	241	139
Adsorption stage 5 discharge	102	89	88	83	156	152	144	84
Adsorption stage 6 discharge	73	67	67	65	99	123	93	62
Adsorption stage 7 discharge	58	56	56	56	68	108	65	54
Stripped carbon feed to last stage	50	50	50	50	50	100	50	50
Gold in solution (mg/L)								
Leach tank discharge	0.557	0.557	0.557	0.557	0.557	0.557	0.557	0.611
Adsorption stage 1 discharge	0.243	0.232	0.211	0.192	0.313	0.245	0.301	0.246
Adsorption stage 2 discharge	0.105	0.096	0.081	0.069	0.170	0.107	0.158	0.100
Adsorption stage 3 discharge	0.047	0.042	0.034	0.028	0.092	0.050	0.083	0.042
Adsorption stage 4 discharge	0.023	0.021	0.017	0.014	0.051	0.027	0.045	0.019
Adsorption stage 5 discharge	0.013	0.012	0.010	0.009	0.029	0.016	0.026	0.009
Adsorption stage 6 discharge	0.008	0.008	0.007	0.006	0.017	0.012	0.015	0.006
Adsorption stage 7 discharge	**0.006**	**0.006**	**0.005**	**0.005**	**0.011**	**0.009**	**0.010**	**0.004**

This test program confirmed that the existing CIL design, one leach and seven CIL stages and a 12 t elution plant will achieve satisfactory results and low solution losses to the CIL tails.

The model outputs showed that six CIL stages will be sufficient, although having a seventh stage will provide more flexibility for mill feed and head grade fluctuation.

The current plant design calls for a 12 t elution plant with seven strips per week, however, the model outputs predicted that a 10 t elution plant with one elution cycle per day (or seven strips / week) can be adequate.

The maximum gold carbon loading is estimated to be at 2,200 g/t. The targeted gold carbon loading will range between ~1,200 g/t to ~1,500 g/t.

It is recommended that after start-up, the CIL circuit profiles be compared to the modelling results. A sample of the plant CIL feed can then be submitted for confirmatory testing and can be used for further optimization with a focus on lowering operating costs (i.e., less carbon, fewer elution cycles).

13.5 Fresh Ore Testwork Program 2019

At the end of 2018, Orezone approached Lycopodium to expand the 2018 feasibility study to include the treatment of lower transition and fresh ores (sulphides). In 2019, Base Metallurgical Laboratories Ltd. (Base Met) conducted a metallurgical testing program in support of this feasibility study. The objective of the program was to expand the understanding of the response of the lower transition and sulphides to conventional cyanidation, comminution, and sedimentation. A blend composite of sulphide and oxide as per the mine plan was also tested to confirm the performance of combined material in the CIL circuit and to provide composite tailings for other testwork.

Four main domain composite samples and 11 variability samples were tested. The cyanidation program explored the effect of grind, cyanide dosage, slurry density and dissolved oxygen levels on the kinetics and final extraction of gold from these samples.

The following sections present key results from this test program. Additional details are provided in the report Base Met Labs report (BL0402, May 2019).

13.5.1 Head Analysis

Table 13.5.1 below lists the composite details and measured head assays, determined through various methods.

Table 13.5.1 Base Met 2019 Head Analysis Results for Composite and Variability Samples

Sample ID	Domain / Mine	Weathering	Au g/t	Cu ppm	Fe %	Ag g/t	S %	C %	Cg %
Method	**-**	**-**	**SM**	**AR_AA**	**AR_AA**	**AR_AA**	**Leco**	**Leco**	**Leco**
Comp 1	Siga_S	Fresh	1.22	90	7.45	2.5	3.080	1.03	<0.01
Comp 2	P8P9	Fresh	1.30	275	8.85	<1	3.285	0.85	<0.01
Comp 3	P8P9	Lower Trans.	0.96	170	8.25	<1	1.375	0.08	<0.01
Comp 4	P17S_W	Fresh	2.36	20	4.48	2	0.925	0.61	<0.01
Method	**-**	**-**	**SM**	**FAAS**	**FAAS**	**FAAS**	**Leco**	**Leco**	**Leco**
Var 1	Maga_H	Fresh	2.0	0.015	7.94	<1	1.97	0.93	<0.01
Var 2	Maga_M	Lower Trans.	0.8	0.003	5.30	<1	0.21	0.46	0.31
Var 3	Siga_S	Lower Trans.	0.8	0.005	6.42	<1	0.72	0.05	<0.01
Var 4	Siga_S	Fresh	1.6	0.003	4.86	<1	1.39	0.93	<0.01
Var 5	Siga_S	Fresh	1.5	0.018	5.88	<1	3.17	1.49	<0.01

Sample ID	Domain / Mine	Weathering	Au g/t	Cu ppm	Fe %	Ag g/t	S %	C %	Cg %
Var 6	P8P9	Fresh	0.9	0.023	5.02	<1	2.89	0.69	0.01
Var 7	P8P9	Fresh	1.6	0.026	3.74	<1	1.19	1.23	0.01
Var 8	P8P9	Lower Trans.	0.9	0.022	4.90	<1	1.78	0.03	<0.01
Var 9	P8P9	Lower Trans.	0.8	0.011	3.96	<1	0.25	0.08	<0.01
Var 10	P17S_E	Fresh	1.8	<0.001	5.08	<1	0.94	0.56	0.01
Var 11	P17S_MI3	Fresh	0.5	<0.001	5.44	<1	0.43	0.42	<0.01

13.5.2 Comminution

Table 13.5.2 summarizes the comminution test results. The samples exhibited a large degree of variation in terms of both hardness and abrasiveness. For instance, the Bond Ball Mill index ranges from 8.6 kWh/t for Var 3 (Siga_S) to as high as 17.8 kWh/t for Var 6 (P8P9). Within each domain, the transition samples are generally less abrasive and less resistant to grinding. However, there is some overlap between domains with the transition samples for P8P9 being just as hard as the fresh rock from the other domains.

Table 13.5.2 Base Met 2019 Comminution Results

Sample ID		Relative Density	JK Data						BWI parameters				Bond Ai
			SMC					Mesh of Grind	F80	P80	Gram/rev	Work Index	Ai
	Rock Type	SMC	A	b	A x b	t_a	DWI (kWh/m^3)		µm	µm		kWh/t	(g)
Comp 1	Siga_S_Fr	2.90	59.0	0.61	36.0	0.32	8.1	150	1,921	80	1.45	13.9	0.245
Comp 2	P8P9_Fr	2.74	68.5	0.50	34.3	0.32	8.1	150	1,906	80	1.33	14.9	0.293
Comp 3	P8P9_Tr_L	2.42	59.4	1.10	65.3	0.70	3.7	150	1,490	79	1.50	13.9	0.069
Comp 4	P17S_W_Fr	2.72	100	0.29	29.0	0.28	9.5	150	1,938	83	1.29	15.7	0.689
Var-1	Maga_H_Fr	2.66	69.4	0.48	33.3	0.32	8.1	150	1,777	80	1.43	14.2	0.261
Var-2	Maga_M_Tr_L	2.31	43.9	1.49	65.4	0.73	3.5	150	1,685	68	2.11	9.4	0.014
Var-3	Siga_S_Tr_L	2.55	49.7	1.81	90.0	0.91	2.8	150	1,629	78	2.62	8.6	0.049
Var-4	Siga_S_Fr_1	2.47	61.8	0.72	44.5	0.47	5.5	150	1,778	80	1.72	12.2	0.110
Var-5	Siga_S_Fr_2	2.38	54.2	0.72	39.0	0.42	6.1	150	1,807	81	1.61	13.0	0.135
Var-6	P8P9_Fr_I1C	2.37	94.5	0.31	29.3	0.32	8.1	150	1,995	83	1.09	17.8	0.673
Var-7	P8P9_Fr_I2	2.32	94.2	0.29	27.3	0.30	8.5	150	1,915	79	1.14	16.8	0.370
Var-8	P8P9_Tr_L_I1	2.54	74.4	0.63	46.9	0.48	5.4	150	1,902	81	1.21	16.3	0.341
Var-9	P8P9_Tr_L_I2	2.58	64.6	0.81	52.3	0.53	4.9	150	1,791	79	1.54	13.3	0.115
Var-10	P17S_E_Fr	2.77	100	0.27	27.0	0.25	10.2	150	2,011	83	1.39	14.6	0.658
Var-11	P17S_MI3_Fr	2.91	93.4	0.31	29.0	0.26	10.1	150	1,887	81	1.39	14.5	0.457

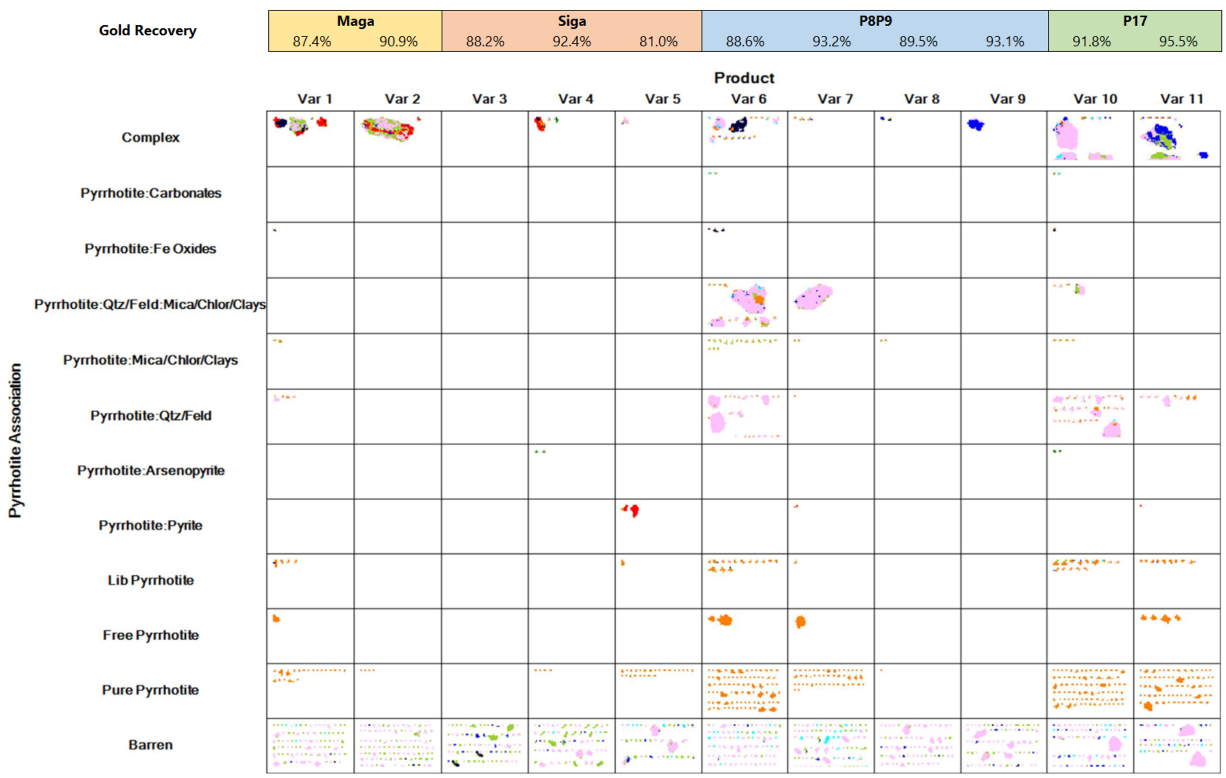
Variability: Overall Statistics	Relative Density SMC	A	b	A x b	t_a	DWI	F80	P80	Gram/rev	Work Index	Ai
Average	**2.576**	**72.5**	**0.69**	**43.2**	**0.44**	**6.8**	**1,829**	**80**	**1.52**	**13.9**	**0.299**
Std. Dev.	0.20	19.2	0.46	18.2	0.20	2.4	143	4	0.39	2.5	0.231
Rel. Std. Dev.	7.79	26.4	66.6	42.1	45.2	35.5	8	4	25.94	17.9	78
Minimum	2.31	43.9	0.27	27.0	0.25	2.8	1,490	68	1.09	8.6	0.014
10th Percentile	2.34	51.5	0.29	28.0	0.27	3.6	1,651	78	1.17	10.5	0.057
25th Percentile	2.40	59.2	0.31	29.1	0.31	5.2	1,778	79	1.31	13.2	0.112
Median	**2.55**	**68.5**	**0.61**	**36.0**	**0.32**	**8.1**	**1,887**	**80**	**1.43**	**14.2**	**0.261**
75th Percentile	2.73	93.8	0.77	49.6	0.51	8.3	1,918	81	1.57	15.3	0.414
90th Percentile	2.85	97.8	1.33	65.4	0.72	9.8	1,972	83	1.96	16.6	0.667
Maximum	2.91	100.0	1.81	90.0	0.91	10.2	2,011	83	2.62	17.8	0.689

Minimum and Maximum refer to softest and hardest for the grindability tests, respectively

13.5.3 Mineralogy

A mineralogical study indicated quartz and plagioclase as the most abundant minerals with biotite also significant for the Siga samples. The dominant sulphide mineral is pyrite at approximately 4% of the total rock mass. This is accompanied by minor quantities (<1%) of pyrrhotite for all except for composite 2 (P8P9) and composite 4 (P17S). The P17S domain differs from the P8P9 in that it contains very little pyrite but significantly higher pyrrhotite (almost 1.8%) as well as some arsenopyrite (1%). The pyrrhotite presence in the P8P9 and P17S domain samples did not affect the ultimate recoveries, as shown in Figure 13.5.1 below.

Figure 13.5.1 Variability Recovery vs Pyrrhotite Association



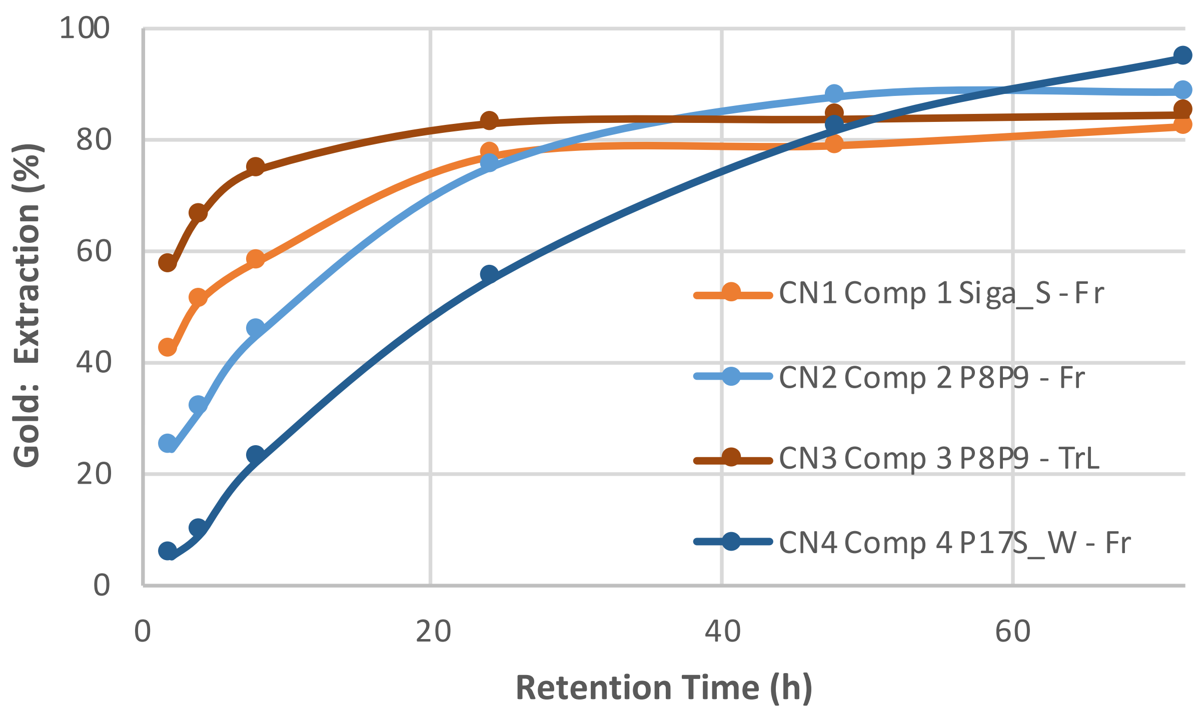
13.5.4 Cyanidation

Cyanidations were performed at the following standard conditions:

- Pulp density of 50% w/w solids.
- Target grind P_{80} of 75 µm.
- Pulp pH of 11.5 (maintained).
- Dissolved oxygen (DO) level maintained through air sparging.
- g/L NaCN (free) of 0.5 (maintained).

Initial optimization testwork varied the grind, NaCN level, pulp density and DO level to determine an optimum suite of operating parameters for the subsequent testing of variability samples. Figure 13.5.2 shows the leach kinetic curves recorded for the four composites at the standard conditions.

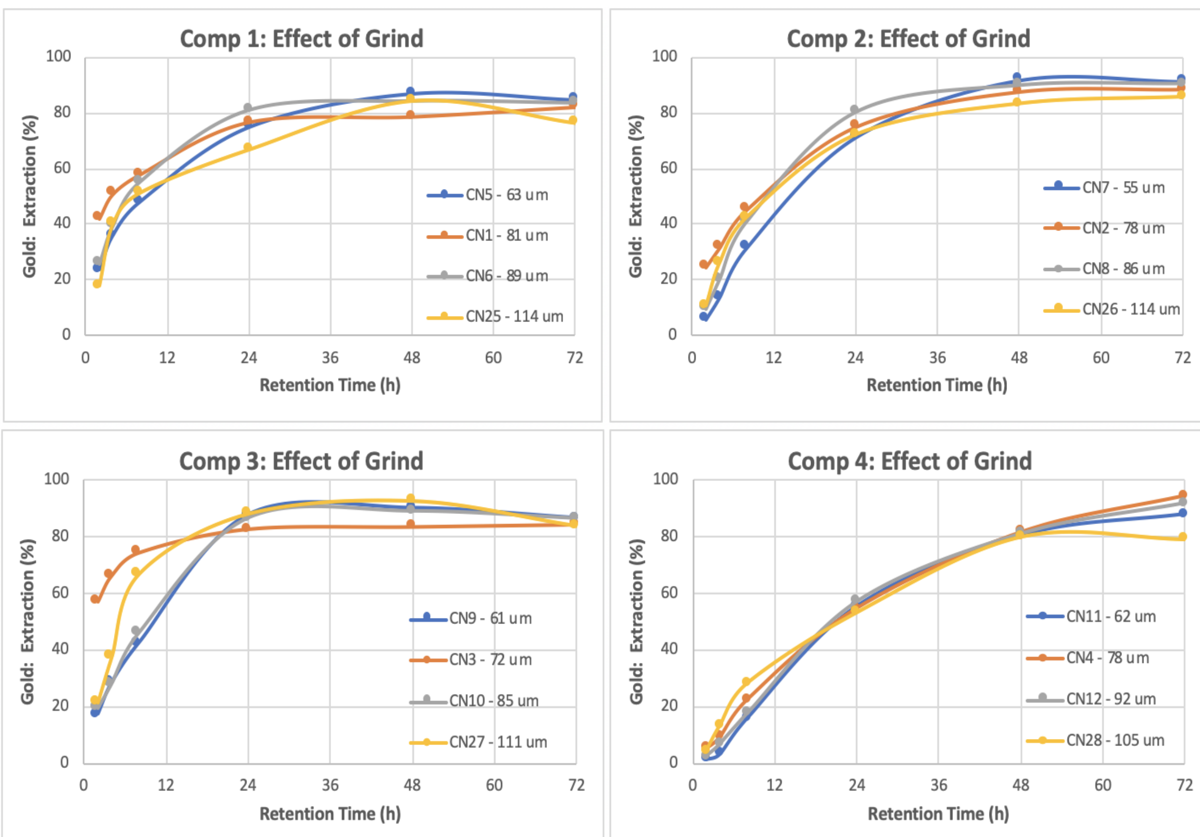
Figure 13.5.2 Base Met 2019 Baseline Leaching Kinetics



Moderate (mid 80%) extractions were achieved for Composites 1 to 3 with Composite 4 yielding a high (95%) gold extraction into solution. The rate of gold leaching is fast for the P8P9 transition sample (Composite 3) and still acceptable for Composite 1. However, it is slow for Composite 2 and exceptionally slow for Composite 4. The rate of dissolution appears to be correlated to the pyrrhotite content of the ore as both Composites 2 and 4 have higher level of pyrrhotite than other composites.

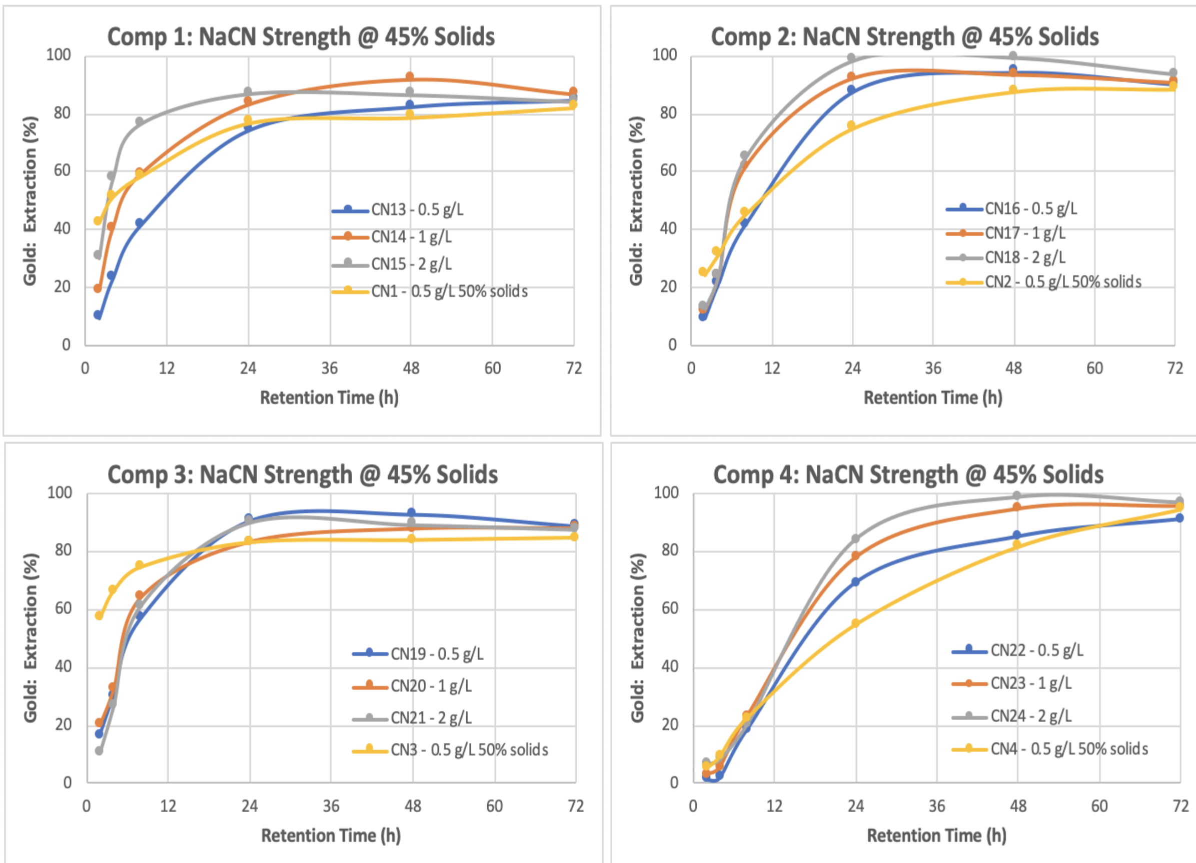
Figure 13.5.3 shows the effect of grind size on leach kinetics. There is no obvious trend indicating that grind size plays a significant role in the rate of dissolution for these samples. When comparing final residue grades the grind sizes below 90 µm did not show consistent nor significant increases in incremental gold extractions at a finer grind size.

Figure 13.5.3 Base Met 2019 - The Effect of Grind Size on Gold Leaching



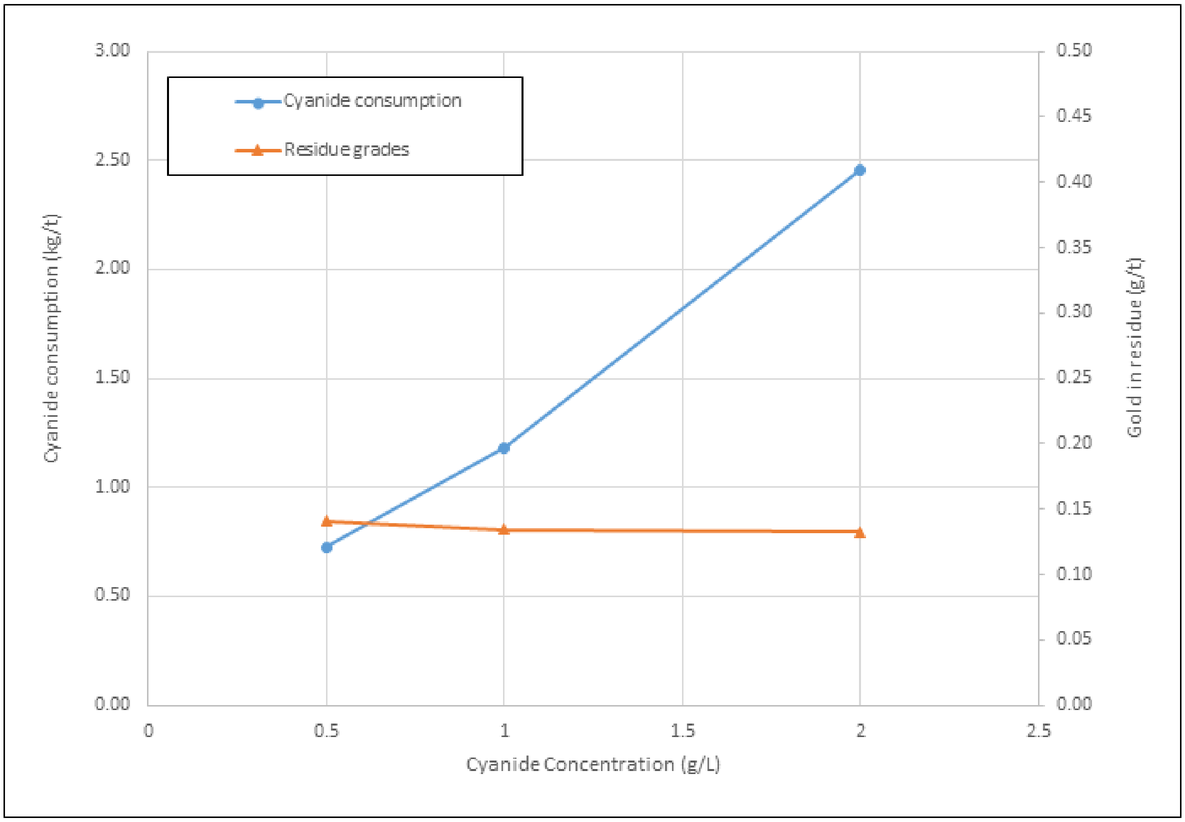
A series of tests at three cyanide levels (0.5, 1.0 and 2.0 g/L) were performed to evaluate the effect of cyanide concentration on leach performance. The kinetics curves presented in Figure 13.5.4 generally suggest an improvement in initial leach kinetics with an increase in cyanide concentration.

Figure 13.5.4 Base Met 2019 - Leach Kinetic Curves at Different Cyanide Concentrations



To further explore the effect of cyanide concentration on leach extraction and consumption, the average residue and consumption rates were calculated for Composites 1, 2 and 3 and are plotted in Figure 13.5.5. Composite 4 (P17S) was excluded due to its very low residue value (high recovery) regardless of test conditions and it represents only a small fraction of the ore body. The maximum value for both the primary and secondary y-axis were chosen to be approximately four times the minimum value of the data in order to provide a scale that allows direct visual comparison of the strength of the correlations.

Figure 13.5.5 Base Met 2019 - Gold Residue and Cyanide Consumption versus Cyanide Concentration

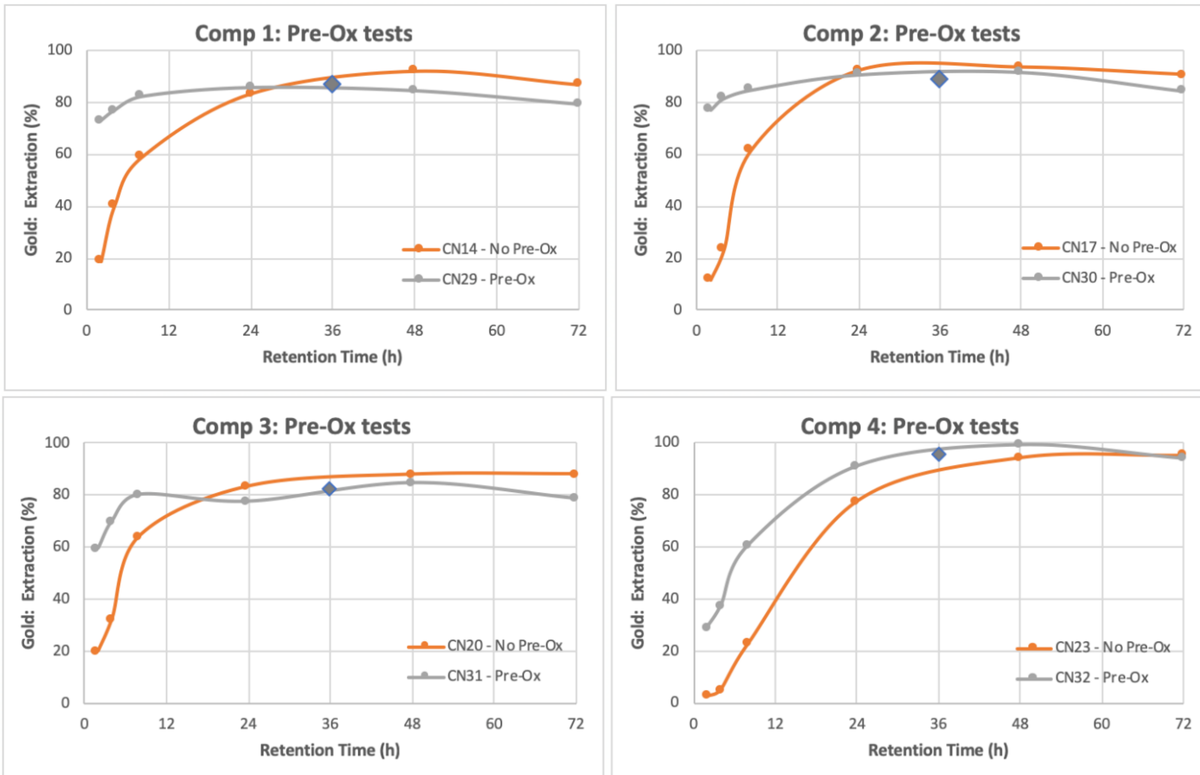


It is evident from Figure 13.5.5 that the average final gold residue (for all composites combined) was largely unaffected by the cyanide concentration used during the test, decreasing from 0.141 g/t at the lowest cyanide concentration to 0.133 g/t at 2 g/L. However, the amount of cyanide consumed during the test was strongly correlated to the cyanide concentration, increasing from 0.72 kg/t at 0.5 g/L to 2.46 kg/t at 2 g/L. As a high cyanide concentration was desirable to improve leach kinetics, pre-oxygenation was investigated to improve the leach kinetics and reduce the impact of cyanide consuming species in the hard rock ores.

Ground samples were pre-oxygenated for eight hours at a controlled dissolved oxygen concentration of 15 ppm. Cyanide was added and the cyanide and oxygen concentrations were maintained at 1 g/L and 15 ppm respectively throughout the duration of the leach.

Figure 13.5.6 demonstrates how the pre-treatment step enhanced leach kinetics for all the composites with much more rapid leaching in the first 12 hours. In addition, the measured cyanide consumption reduced significantly with the use of pre-oxygenation. The average cyanide consumption reduced from 1.2 kg/t with air sparging to less than 0.5 kg/t for tests using pre-oxygenation and oxygen sparging.

Figure 13.5.6 Base Met 2019 - Comparison of Pre-oxygenation and Oxygen Sparging versus Air Only



13.5.5 Variability Testing

Variability testing was performed at the following conditions:

- Grind P_{80} of 75 µm.
- Pulp density of 50% w/w solids.
- Pulp pH of 11.5.
- Pre-oxygenation for eight hours.
- DO of 15 ppm maintained.
- g/L NaCN (free) of 1.0 maintained.
- Leach duration of 48 hours.

The average gold extraction achieved for the variability samples was 91.6%, which is higher than the predicted recoveries from the equations developed from LeachWell database.

The measured cyanide consumption ranged from 0.25 kg/t to 0.9 kg/t with a median value of 0.36 kg/t. Lime consumption ranged from 0.68 kg/t to 2.99 kg/t with a median value of 1.38 kg/t. Generally, the variability testing did not yield an unusually large degree of variability, which augurs well for a smooth operation.

13.5.6 Sedimentation Testwork

Dynamic settling tests were conducted with a bench-scale raked thickener. The three tests were performed at a constant feed rate equivalent to a loading of 0.3 t/h/m^2. One of the tests used a different flocculant (MF10 instead of MF351) while the third test was performed at an elevated pH (11.3 versus 10.5). All three tests yielded similar underflow densities within the range 57 to 58 % w/w solids. The test at a higher pH yielded less suspended solids in the overflow. Further sedimentation tests are required to determine the expected loading rate for the targeted pre-leach thickener underflow density of 50% w/w solids.

13.6 Fresh Ore Testwork Program 2023

At the end of 2022, Orezone approached Lycopodium to undertake a feasibility study to include a sulphide processing stream at their existing oxide ore treatment plant. In 2023, Maelgwyn South Africa (Maelgwyn) conducted a metallurgical testing program in support of this feasibility study to increase the confidence level of the hard rock ore sources and build on historic testwork. Thickening and viscosity testwork was sub-contracted to Mac One Technologies (Mac One).

Eleven P17, twelve P17 South, four Siga West, two Siga South, two Siga East, and six Maga samples were tested as composites and variability samples. The testwork program explored the effect of grind, cyanide dosage, slurry density and varying pre-oxidation methods have on the kinetics and final extraction of gold from these samples.

The following sections present key results from this test program. Additional details are provided in the final testwork report from Maelgwyn South Africa (Pty) Ltd, REP 22-139-3, June 2023.

Sample Selection

Maps indicating the metallurgical sampling locations for this testwork program are shown in Figure 13.6.1, Figure 13.6.2, and Figure 13.6.3 below. The selection of the individual exploration core samples making up each of the composited metallurgical samples was designed to be as representative as possible of the deposit that was targeted in each instance, in terms of lithology, grade bin and spatial distribution.

Figure 13.6.1 P17 Metallurgical Sampling Locations

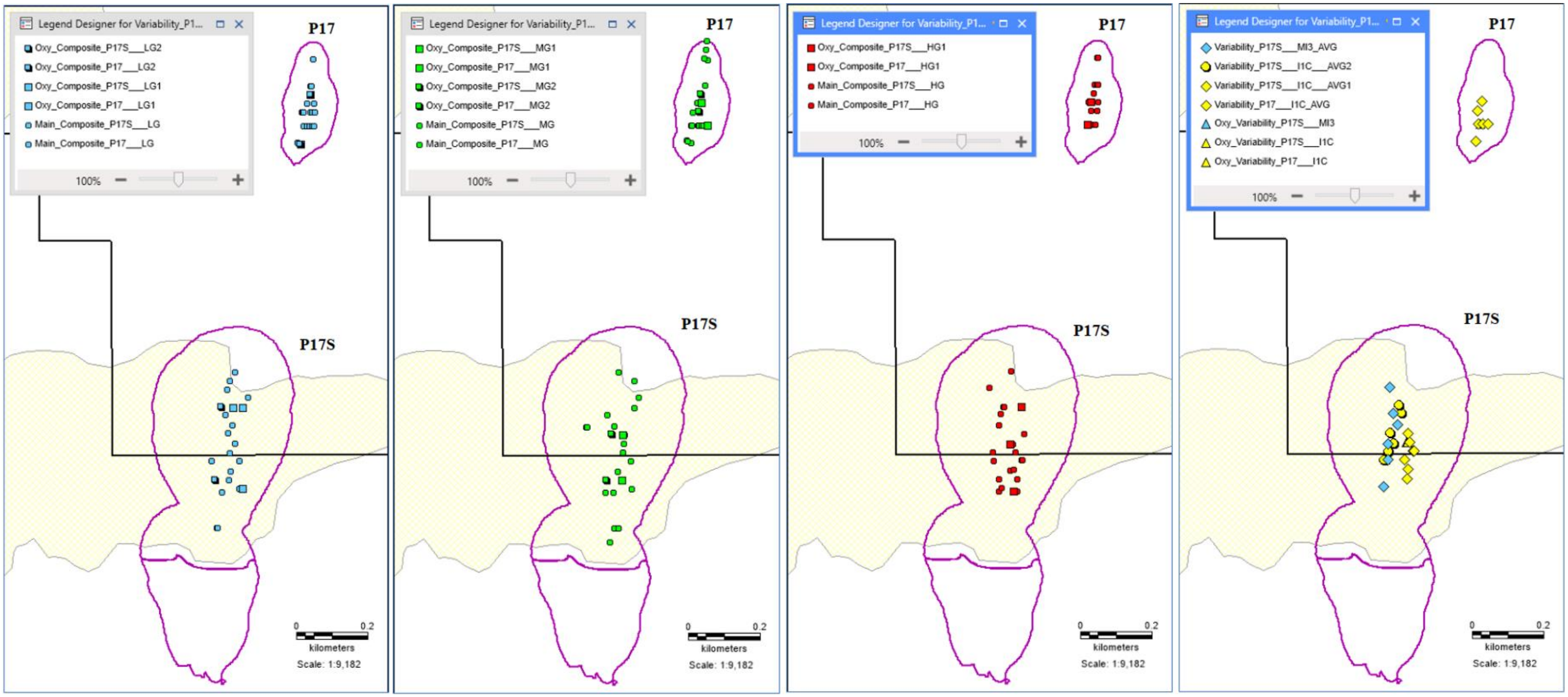


Figure 13.6.2 Siga Metallurgical Sampling Locations

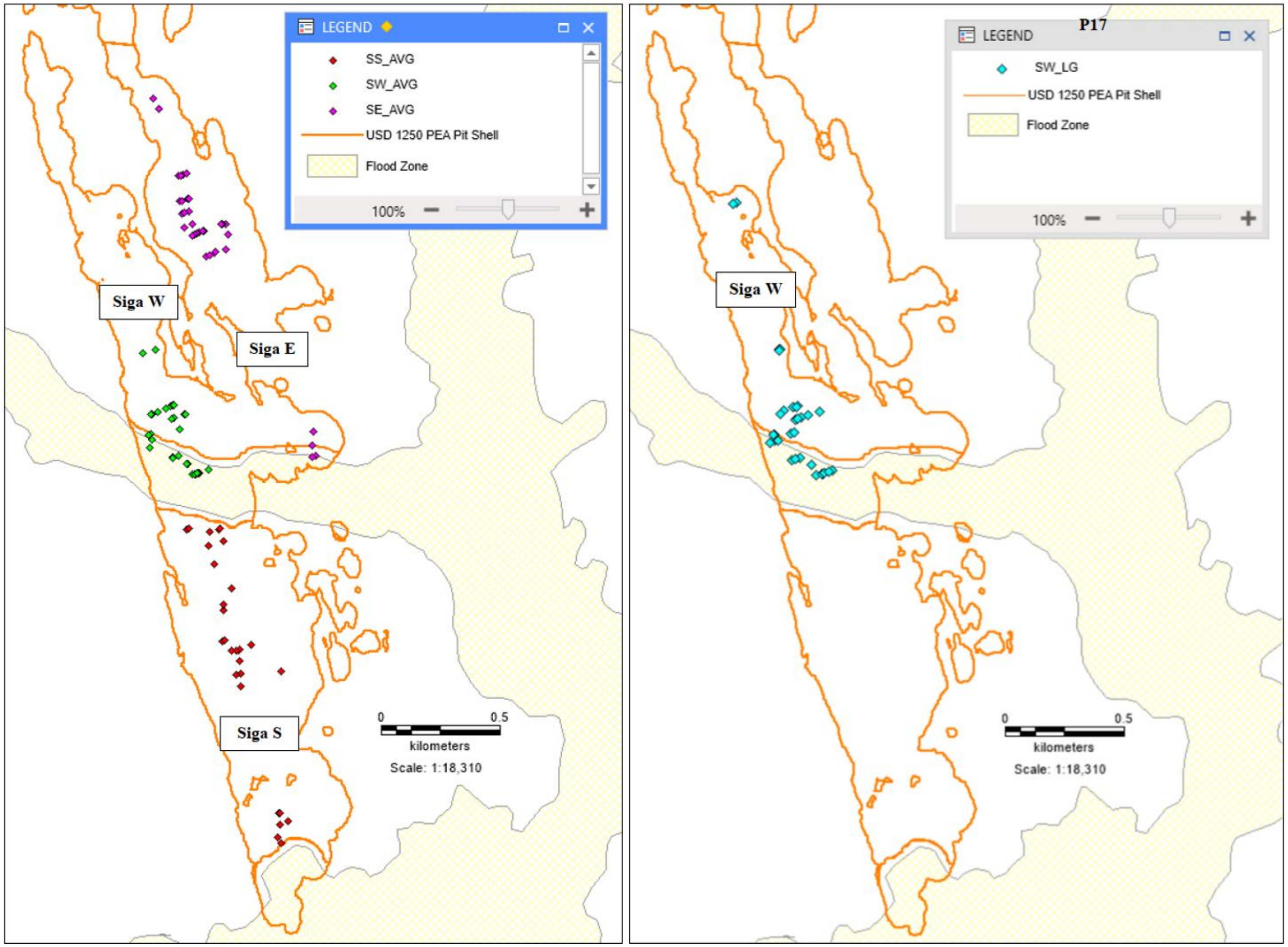
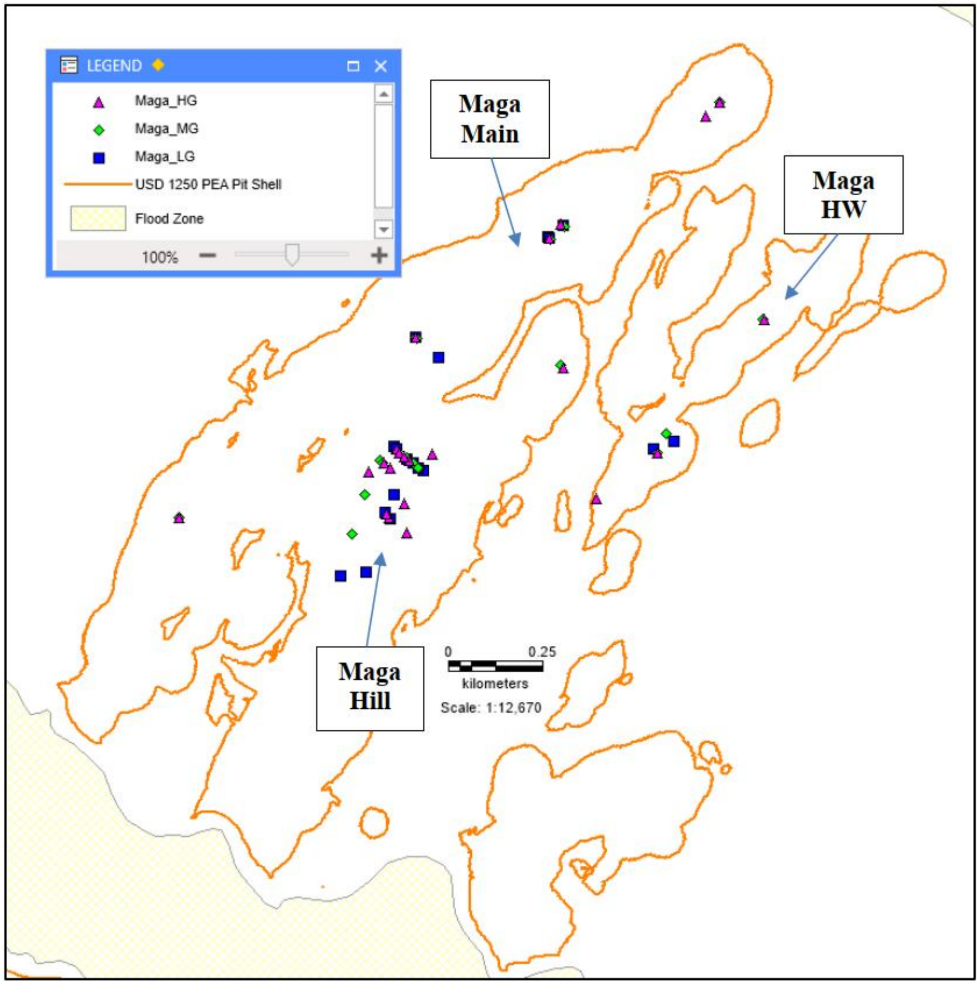


Figure 13.6.3 Maga Metallurgical Sampling Locations



Head Analysis

Table 13.6.1 below lists the composite details and measured head assays, determined through various methods.

Table 13.6.1 Maelgwyn 2023 Head Analysis Results for Composite and Variability Samples

Sample ID	Au - Dupl. Head g/t	Ag Head g/t	Total C %	Total S %	As ppm	SG
P17 Composite	1.91	7	0.51	0.66	2821	
Siga Composite	1.23	2	0.86	2.55	377	
Maga LG Composite	0.73	0.2	1.09	0.98	753	
Maga MG Composite	0.97	0.3	1.25	1.80	906	
Maga HG Composite	1.38	0.3	1.33	1.77	855	
Master Composite	1.43	4.2	0.91	1.77	1647	
P17 HG	2.53	ND	0.40	0.76	2042	2.82
P17 LG	0.36	ND	0.44	0.95	79	2.77
P17 MG	1.02	ND	0.37	0.38	816	2.84
P17 - HG1	1.37	ND	0.53	0.42	1216	2.79
P17 - MG1	0.61	ND	0.52	0.38	101	2.85
P17 - MG2	0.82	ND	0.39	0.03	526	2.96
P17 - LG1	0.10	ND	0.20	0.43	807	3.03
P17 - LG2	0.21	ND	0.95	0.82	4184	2.90
P17-I1C	1.16	ND	0.59	0.82	6123	2.80
P17S-I1C	0.70	ND	0.64	0.90	1069	2.79
P17-I1C Ave	1.19	ND	0.50	0.33	3742	2.80
P17S HG	1.81	ND	0.53	0.63	3239	2.79
P17S MG	0.79	ND	0.47	0.13	1330	2.79
P17S LG	0.49	ND	1.12	0.88	2372	2.81
P17S-HG1	3.66	ND	0.50	0.85	5418	2.79
P17S-MG1	0.89	ND	0.50	0.51	1105	2.79
P17S-MG2	1.98	ND	0.55	0.19	1297	2.81
P17S-MI3	0.94	ND	0.60	0.93	1431	2.87
P17S-LG1	0.32	ND	0.48	0.63	412	2.80
P17S - LG2	0.30	ND	0.48	0.47	313	2.86
P17S-I1C Ave1	1.48	ND	0.79	0.88	2118	2.90
P17S-I1C Ave2	3.03	ND	0.42	0.86	3948	2.81
P17S-MI3 Ave	0.89	ND	0.56	0.81	1188	2.86

Sample ID	Au - Dupl. Head g/t	Ag Head g/t	Total C %	Total S %	As ppm	SG
SW_LG 1	0.68	3	1.09	4.12	91	3.00
SW_LG 2	0.79	4	1.06	3.57	40	2.95
SW_AVG 1	1.77	7	1.21	4.06	44	2.90
SW_ AVG 2	1.01	3	0.75	3.55	49	3.05
SS_AVG 1	1.17	16	1.05	3.56	61	2.91
SS_AVG 2	1.27	6	0.44	1.19	594	2.75
SE_AVG 1	0.99	2	0.76	1.91	1139	2.82
SE_AVG 2	0.83	4	1.04	2.03	877	2.81
Maga_LG1	0.65	0.2	1.23	0.98	1115	2.95
Maga_LG2	0.83	0.2	1.00	0.99	542	2.93
Maga_MG 1	1.13	0.2	1.25	1.88	877	2.87
Maga_MG 2	0.99	0.2	1.23	1.70	1036	2.91
Maga_HG1	1.25	0.3	1.46	1.69	746	3.08
Maga_HG2	1.65	0.2	1.28	1.63	983	3.01

**ND = Not Detected

13.6.2 Gravity Recoverable Gold

Due to limited samples it was not possible to run a full gravity testwork program and Gravity Recoverable Gold (GRG) scoping tests were carried out to test viability of gravity recovery only.

The P17 LG, MG and HG, as well as the P17 South LG, MG and HG samples were individually submitted for the gravity tests. The results showed that for the P17 sample milled to the finest grind of P_{80} of 75 µm ~40% of gravity gold was recovered and for P17 South samples ~30% gravity gold was recovered.

The Siga composite sample was submitted for GRG analysis with 32% gravity gold recovered when milled to P_{80} of 75 µm. No Maga samples were submitted for GRG analysis.

As the tests showed promise, a full gravity recovery and gravity tails leach testwork program should be considered in the future.

13.6.3 Oxygen Uptake Rate

The oxygen uptake rate (OUR) was evaluated on the initial as well as the most abundant fresh ore domains, results below in Table 13.6.2. The oxygen consumption rate generally followed a downward trend over time as the cycles progressed, with cycle one having the highest uptake and cycle six the lowest.

Table 13.6.2 Fresh Ore Oxygen Uptake Rate

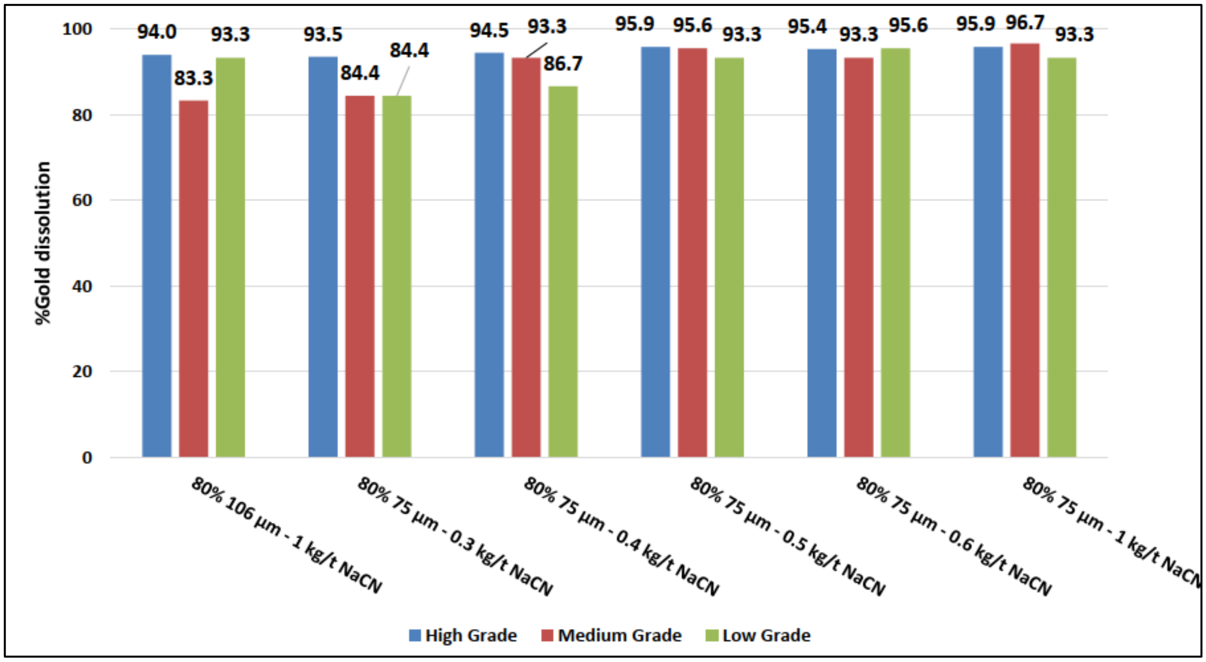
Oxygen consumption rate, $mg/L.min^{-1}$								
Elapsed Time min	**P17 South HG**	**P17 South MG**	**P17 South LG**	**P17 HG**	**P17 MG**	**P17 LG**	**P17 Composite**	**Siga Composite**
60	0.58	0.14	0.22	0.16	0.07	0.10	0.14	0.22
120	0.13	0.11	0.08	0.10	0.11	0.09	0.08	0.12
180	0.16	0.10	0.10	0.11	0.11	0.06	0.09	0.11
240	0.12	0.11	0.09	0.10	0.10	0.09	0.08	0.11
300	0.11	0.10	0.10	0.10	0.07	0.08	0.08	0.10
360	0.11	0.10	0.10	0.08	0.07	0.09	0.08	0.09

13.6.4 Leach Tests

P17 Cyanide consumption and grind size evaluation was done with equal mass composites of P17 and P17 South low, medium and high-grade samples. The leach conditions were as follows:

- Target grind P_{80} of 106 µm and 75 µm (1 kg/t NaCN).
- Target grind P_{80} of 75 µm (0.3, 0.4, 0.5, and 0.6 kg/t NaCN).
- Pulp density of 50% w/w solids.
- Pulp pH of 10.5.
- Leach time: 24 hours without carbon plus 24 hours after carbon addition (48 hours).

Figure 13.6.4 P17 Cyanide Consumption and Grind Evaluation

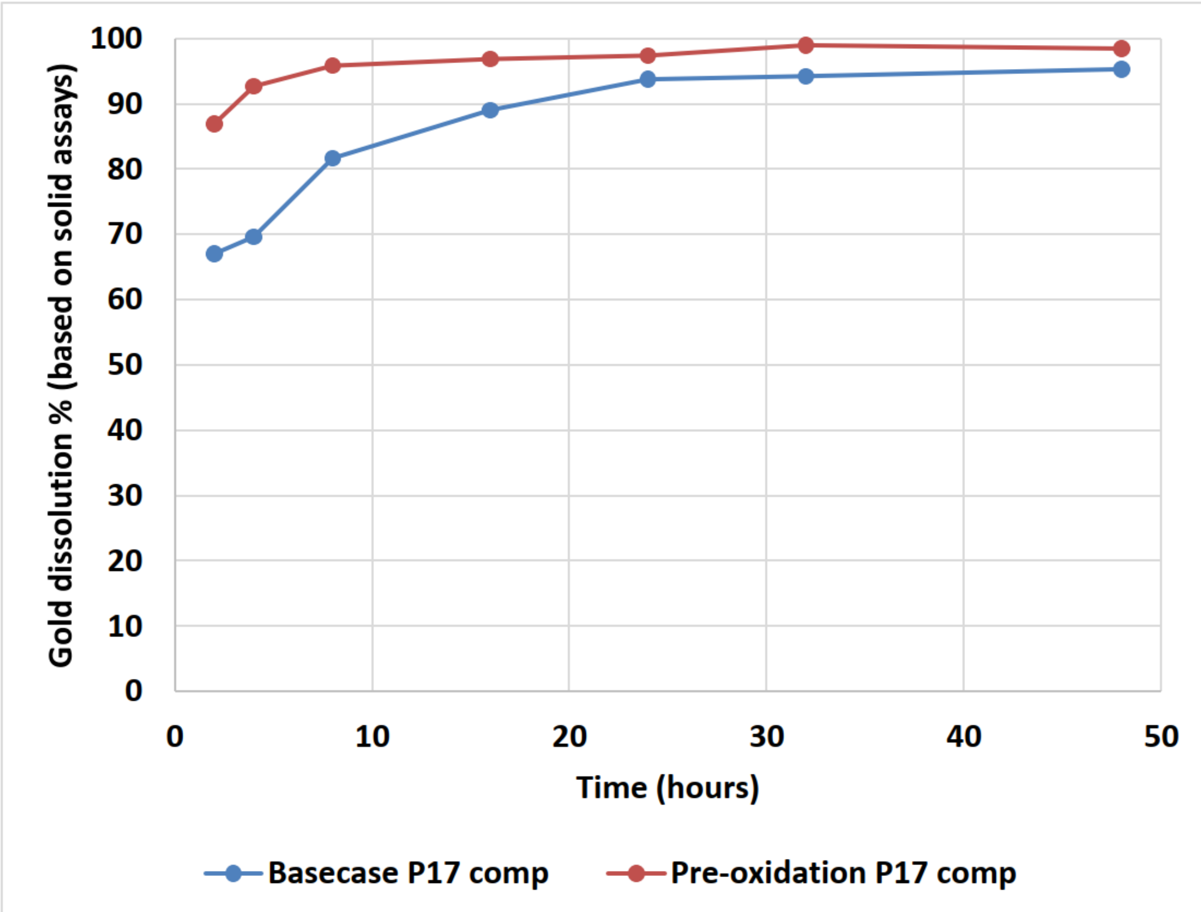


In Figure 13.6.4 cyanide addition below 0.6 kg/t negatively affected the low-grade composite recovery and below 0.5 kg/t for the medium and high grade samples. The coarser grind notably affected the medium grade composite recovery negatively.

Kinetic bottle roll tests were conducted to evaluate the effect of oxygen pre-oxidation with leach conditions as follows:

- Target grind P_{80} of 75 µm.
- Cyanide addition of 0.5 kg/t.
- Pulp density of 50% w/w solids.
- Pulp pH of 10.5.
- Leach time: 2, 4, 8, 16, 24, 32, and 48 hours.

Figure 13.6.5 P17 Basecase Kinetic vs Oxygen Pre-Oxidation Leach Results



In Figure 13.6.5oxygen pre-oxidation resulted in faster leach kinetics and a 3% recovery increase after 48 hours, with pre-oxidation resulting in 98% gold dissolution.

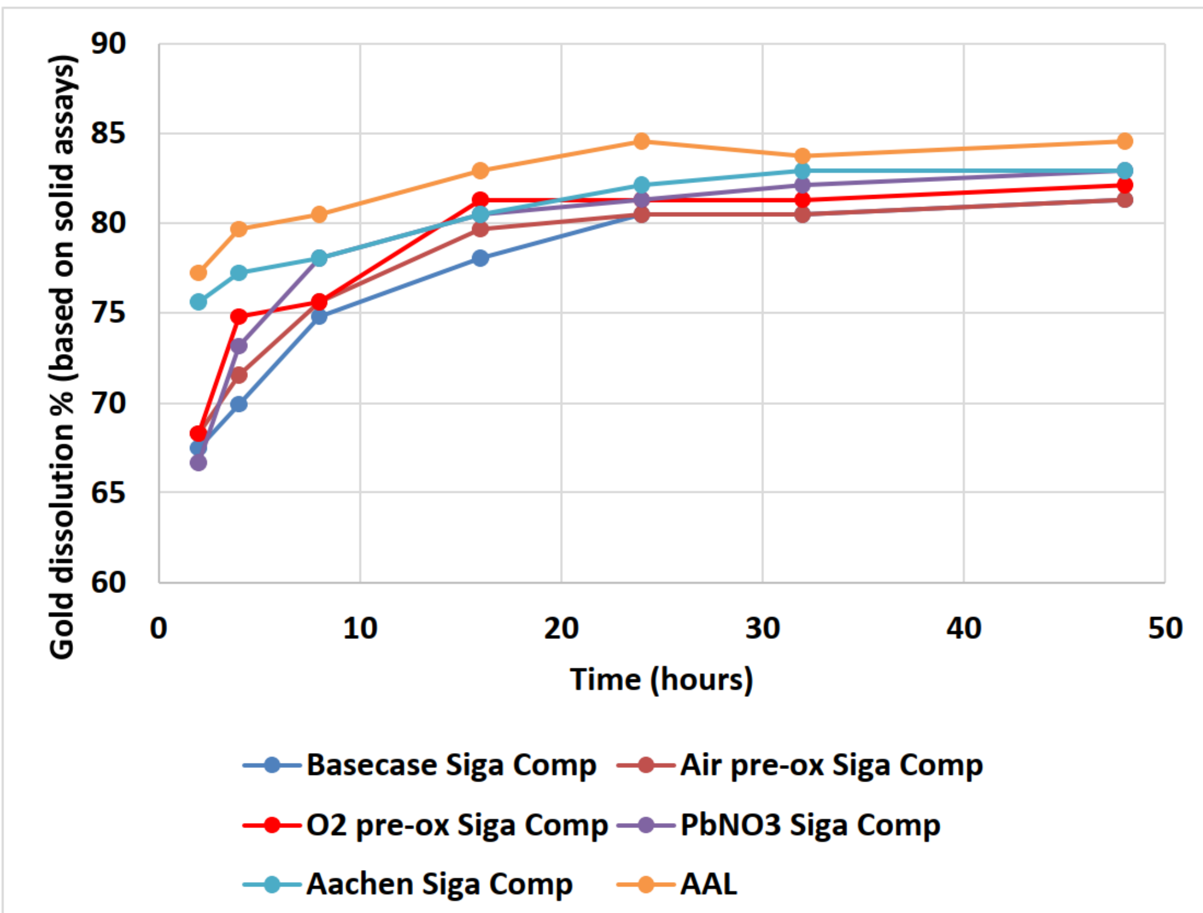
For the Siga composite samples, various pre-oxidation methods were compared with kinetic leaches as follows:

- 4 hour compressed air pre-oxidation.
- 4 hour oxygen pre-oxidation.
- 500 g/t $PbNO_3$ addition and 4 hour compressed air pre-oxidation.
- 2 Pass Aachen shear reactor pre-oxidation.
- 2 Pass Aachen shear reactor pre-oxidation + 2 Pass Aachen Assisted Leach (AAL)

The base case leach conditions were:

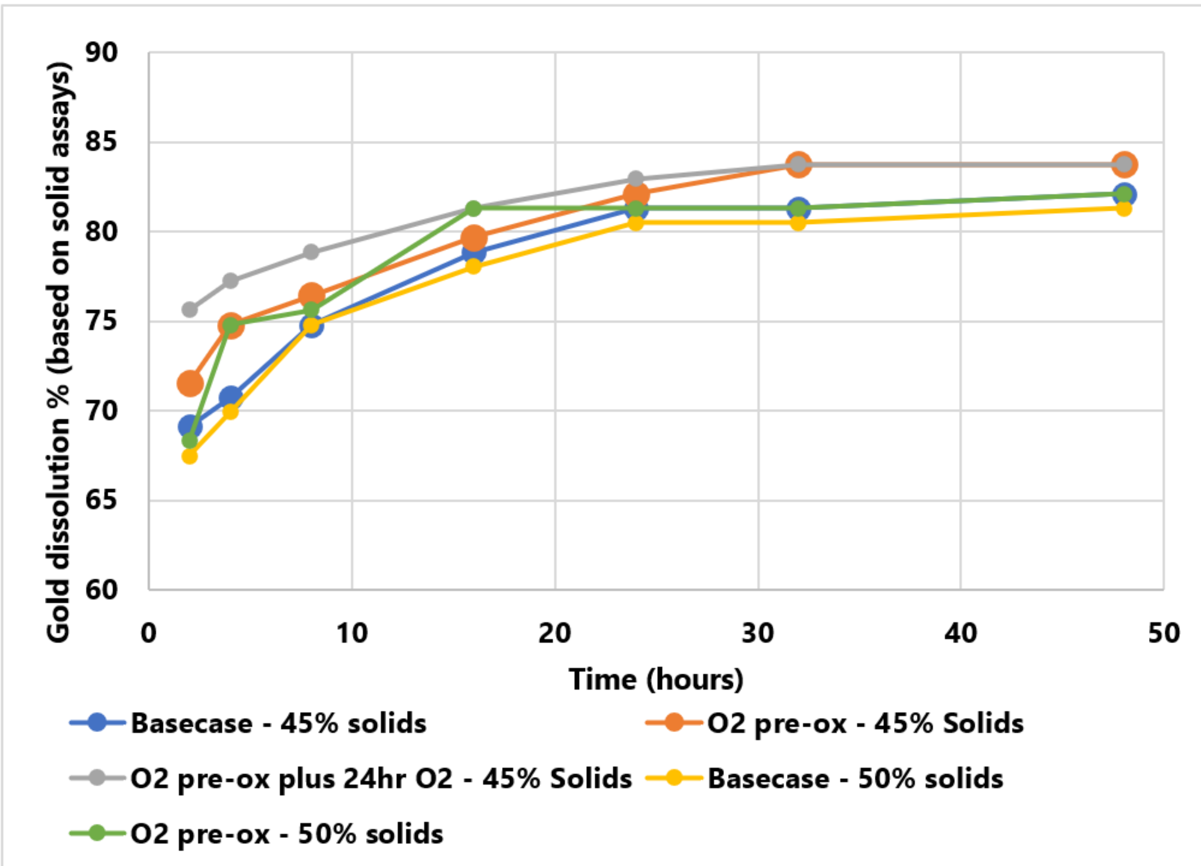
- Target grind P_{80} of 75 µm.
- Cyanide addition of 1 kg/t.
- Pulp density of 50% w/w solids.
- Pulp pH of 10.5.
- 20 g/L Carbon addition (CIL).
- Leach time: 2, 4, 8, 16, 24, 32, and 48 hours.

Figure 13.6.6 Siga Kinetic Pre-Oxidation Comparison Leach Results



With interim viscosity results indicating possible concerns around 50%w/w solids and historic testwork indicating some benefit in leaching at lower solids concentrations, a set of kinetic leaches were performed at 45%w/w solids with the base case conditions and four hour oxygen pre-oxidation, see Figure 13.6.7 below.

Figure 13.6.7 Siga Kinetic Solids% Leach Results



From Figure 13.6.6 and Figure 13.6.7 oxygen pre-oxidation at 45%w/w solids gave comparable results to the best case at 50%w/w solids with AAL, 83.7% and 84.5% recovery respectively. Oxygen sparging throughout the leach further improved leach kinetics and was used in all variability testing.

13.6.5 Variability Tests

Variability testing was done with the following leach conditions:

- Target grind P_{80} of 106 µm and 75 µm.
- Pulp density of 45% w/w solids.

- Pulp pH of 10.5.
- 4 hour oxygen pre-oxidation with oxygen followed by oxygen sparging throughout the leach.
- 0.72 kg/t NaCN addition targeting final 100 ppm NaCN.
- 20 g/L Carbon addition (CIL).
- Leach time: 2, 4, 8, 16, 24, 32, and 48 hours.

Table 13.6.3 Variability Test Summary

Sample ID	Rock Type	Grind µm	Consumption kg/t		Au			Recovery %						
					Hd Assay	Hd Calc	Residue	Leach Kinetics Hours						
			NaCN	CaO	g/t	g/t	g/t	2	4	8	16	24	32	48
SW LG1	Siga	75	0.49	0.71	0.68	0.66	0.12	67.6	70.6	73.5	75.0	77.9	80.9	82.4
SW LG1	Siga	106	0.45	0.58	0.68	0.65	0.15	63.2	66.2	67.6	70.6	73.5	75.0	77.9
SW Ave1	Siga	75	0.54	0.54	1.77	1.84	0.30	67.7	68.8	71.7	76.8	79.6	81.9	83.0
SW Ave1	Siga	106	0.58	0.42	1.77	1.80	0.35	62.6	67.7	70.0	74.5	77.3	79.0	80.2
SS Ave1	Siga	75	0.61	0.36	1.17	1.12	0.21	66.7	70.1	73.5	76.1	80.3	81.2	82.1
SS Ave1	Siga	106	0.50	0.36	1.17	1.15	0.27	61.5	63.2	65.0	68.4	73.5	76.1	76.9
SS Ave2	Siga	75	0.42	0.49	1.27	1.22	0.21	75.6	78.0	81.1	81.3	83.5	83.5	83.5
SS Ave2	Siga	106	0.57	0.46	1.27	1.29	0.28	70.1	71.7	72.4	74.0	74.8	76.4	78.0
SE Ave1	Siga	75	0.61	0.40	0.99	1.01	0.10	77.8	80.8	81.8	84.8	87.9	88.9	89.9
SE Ave1	Siga	106	0.60	0.38	0.99	0.98	0.15	72.7	76.8	79.8	80.8	81.8	82.8	84.8
SE Ave2	Siga	75	0.50	0.53	0.83	0.84	0.18	66.3	67.5	71.1	75.9	77.1	78.3	78.3
SE Ave2	Siga	106	0.47	0.47	0.83	0.80	0.21	56.6	63.9	67.5	71.1	71.1	72.3	74.7
M LG Comp	Maga	75	0.57	1.25	0.76	0.76	0.15	69.7	71.1	73.7	76.3	77.6	78.9	80.3
M LG Comp	Maga	106	0.49	1.28	0.76	0.77	0.18	65.8	67.1	68.4	71.1	73.7	75.0	76.3
M MG Comp	Maga	75	0.58	0.86	0.97	0.95	0.16	69.1	71.1	73.2	76.3	79.4	81.4	83.5
M MG Comp	Maga	106	0.60	0.85	0.97	0.93	0.22	61.9	62.9	66.0	69.1	72.2	74.2	77.3
M HG Comp	Maga	75	0.59	0.81	1.38	1.43	0.28	74.6	75.4	76.8	77.5	78.3	79.0	79.7
M HG Comp	Maga	106	0.58	0.80	1.38	1.45	0.35	65.2	68.8	70.3	72.5	73.2	73.9	74.6

For the Siga samples the recoveries ranged between 78.3% to 89.9% for the P_{80} 75µm samples and 74.7% to 84.9% for the P_{80} 106µm samples respectively.

For the Maga samples the recoveries ranged between 79.7% to 83.5% for the P_{80} 75µm samples and 74.6% to 77.3% for the P_{80} 106µm samples respectively.

On average, grinding to a coarser grind of P_{80} 106µm compared to P_{80} 75µm resulted in a recovery drop of 4.4% for Siga samples and 5.1% for Maga samples respectively.

13.6.6 Diagnostic Leach

The Siga composite leach tails (P_{80} 75µm, 4 hour oxygen pre-oxidation) and the worst leaching tails of the Maga composites (high grade variability leach, P_{80} 75µm) were sent for diagnostic leaching in order to define the mineralogy and gold association of theses samples on a high level.

Both the Siga and Maga tails samples indicated that no cyanide soluble or CIL recoverable gold was remaining.

For the Siga sample, 14.5% of the remaining gold was associated with the hydrochloric acid digestible minerals (pyrrhotite, calcite etc) ~36% of the gold was associated with nitric acid digestible minerals (pyrite, arsenopyrite, etc.). The final 46% of gold was still not liberated at this grind and was associated with gangue and silicates.

For the Maga sample, ~24% of the remaining gold was associated with the hydrochloric acid digestible minerals (pyrrhotite, calcite etc). ~46% of the gold was associated with nitric acid digestible minerals (pyrite, arsenopyrite, etc.). The final 31% of gold was still not liberated at this grind and was associated with gangue and silicates.

13.6.7 Viscosity

A summary of the P17 composite sample viscosity at solids percentages between 30% to 55% at natural pH and pH 10.5 is shown in Figure 13.6.8. A summary of the Siga composite sample viscosity at solids percentages between 50% and 55% at natural pH and pH 10.5 is shown in Figure 13.6.9.

Viscosity of the Mac One thickened products are shown in Figure 13.6.10 which are at 1.86 SG for the P17 composite and 1.82 SG for the Siga composite samples respectively.

Figure 13.6.8 P17 Viscosity Results

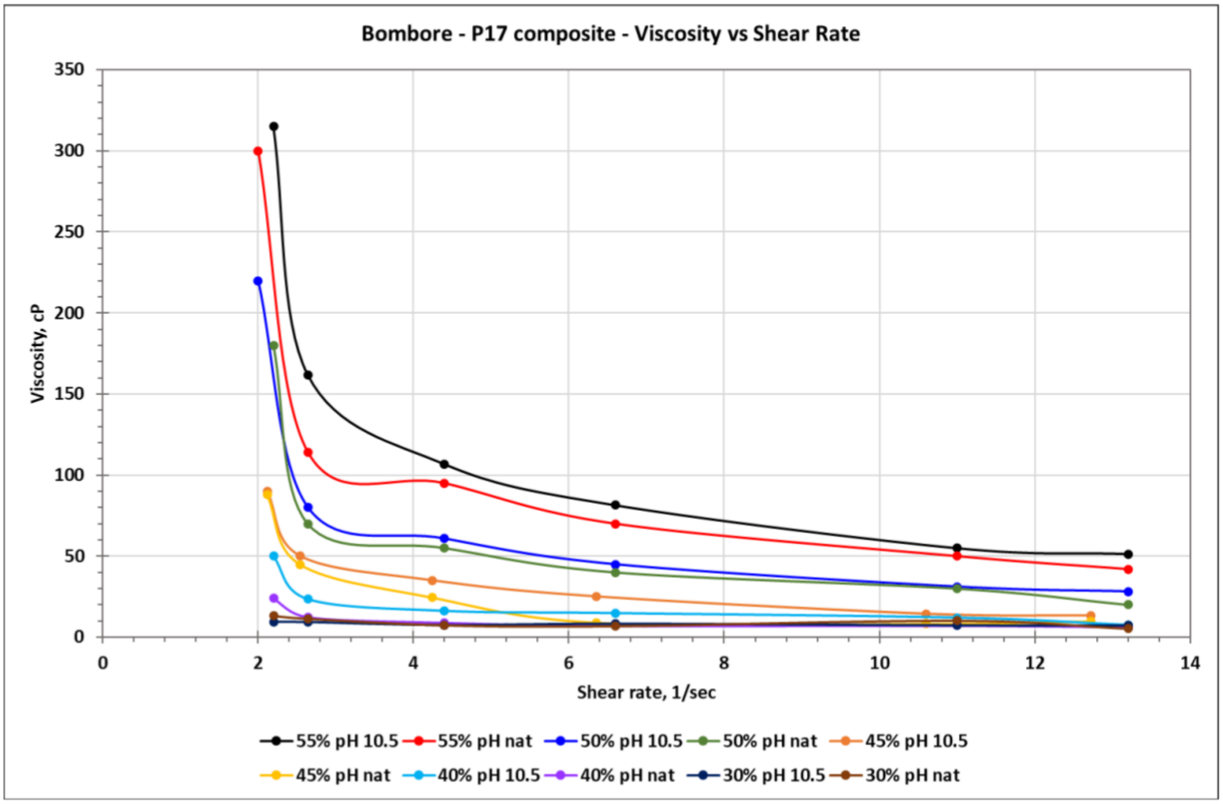


Figure 13.6.9 Siga Viscosity Results

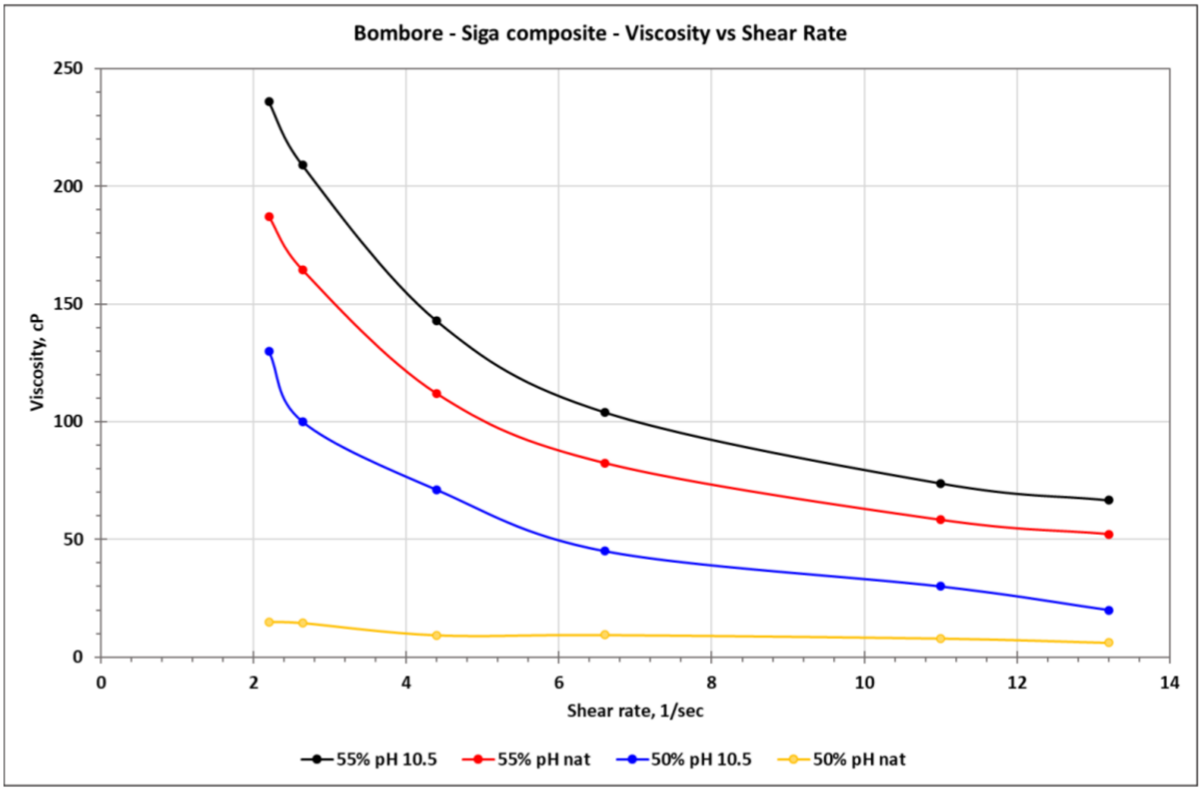
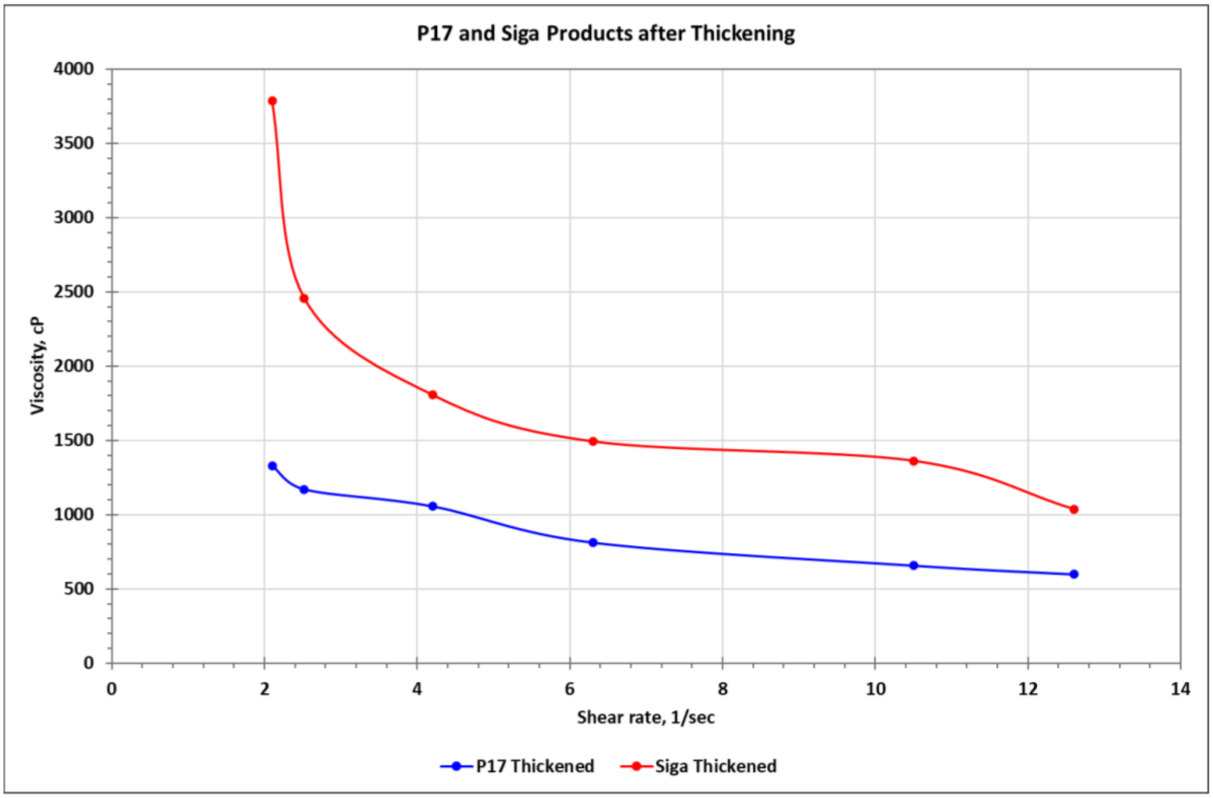


Figure 13.6.10 Thickened Product Viscosity Results



13.6.8 Dynamic Thickening

Dynamic thickening testwork was performed by Mac One on P17 and Siga P_{80} 75µm composites respectively.

For the P17 composite a solids flux of ±1.0 t/h per m^2, 8-12 g/t of Senfloc 5330 flocculant dosing, 19% ±1%w/w solids feedwell slurry density was recommended to achieve a maximum of 74% w/w solids underflow density and lower than 30 NTU turbidity in the overflow.

For the Siga composite a solids flux of ±1.0 t/h per m^2, 7-10 g/t of Senfloc 5330 flocculant dosing, 15% ±1%w/w solids feedwell slurry density was recommended to achieve a maximum of 70% w/w solids underflow density and lower than 100 NTU turbidity in the overflow.

It should be noted that Mac One recommended tails pumping should be performed on <1.4 SG as thickener underflows in excess of 1.5 SG would be too dense to be pumped by centrifugal pumps. For this reason, tailings thickening was excluded from the fresh ore plant design.

13.7 Results Interpretation

The following sections describe the main results that contributed to the development of the process design criteria for the Bomboré Project.

13.7.1 Comminution

The grind size selected for the existing oxide ore mill is P_{80} 125 µm. The testwork showed the oxide ore leach residue grades do not vary significantly with grind size and there appears to be little benefit from fine grinding. Currently, the oxide ore plant is delivering a slightly finer grind size of P_{90} 125 µm and is achieving a gold recovery of 91.8%.

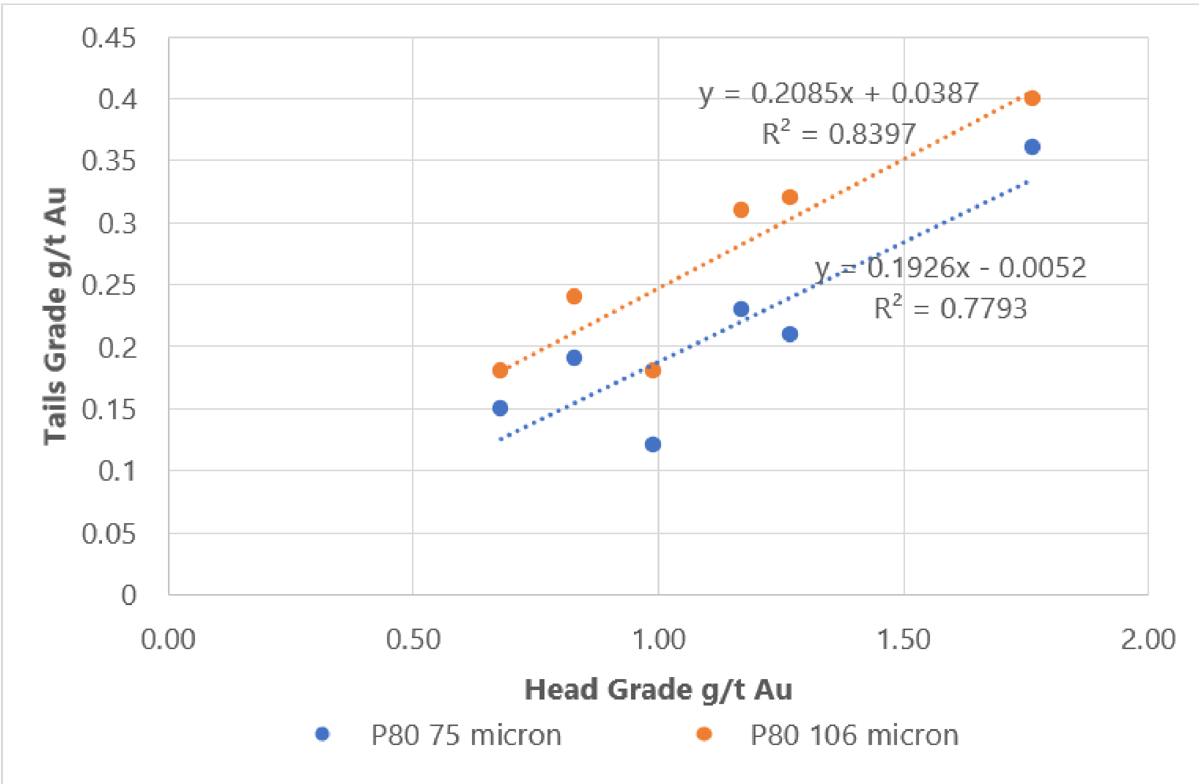
The oxide ore comminution data summarized by Orway Mineral Consultants (OMC) is presented in Table 13.7.1.

Table 13.7.1 Oxide Ore Comminution Data

Parameter	Units	Process Design Criteria			Notes
		Oxide	Transition Upper	Blend	
ROM Ore Types					
Design Ore Blend	%	85	15	100	Client
ROM Distribution					
ROM Fresh Feed, F_{80}	mm	7.4	23.2	9.8	KCA014009/96
ROM Properties					
Moisture Content	%H_2O	5.70	2.80	5.27	RPA Resource Update 2017
Specific Gravity	-			2.83	Outotec 11282017-TQ1-TM-001-R0
Ai	g	0.028	0.052	0.031	Testwork Ave
CWi	kWh/t	7.5	8.6	7.7	Testwork Max
RWi	kWh/t	5.4	7.8	5.8	Testwork - 1 Sample
BWi	kWh/t	4.3	7.8	4.8	85th Percentile

The grind size selected for the new hard rock plant is P_{80} 75 µm. The metallurgical testwork completed by Maelgwyn in 2023 showed there is a sensitivity of gold recovery to grind size for the fresh ore. Figure 13.7.1 below demonstrates a fixed tail increase of approximately 0.05 g/t is generated on the Siga fresh ore when the grind P_{80} changed from 75 µm to 106 µm after 24 hours of leach. Similarly, the Maga samples demonstrated a fixed tail increase of approximately 0.05 g/t with a P_{80} grind size of 106 µm compared to P_{80} of 75 µm. This is likely due to unliberated gold in the coarser particles.

Figure 13.7.1 Siga Grind Size Sensitivity



The fresh ore comminution data summarized by OMC is presented in Table 13.7.2.

Table 13.7.2 Fresh Ore Comminution Data

Block	Weathering Type	SMC Axb 85th %	CWi kWh/t	RWi kWh/t 85th %	BWi kWh/t 85th%	Ai g Ave	Ore SG Ave
Maga	Lower Trans	65.4	13.3	15.3	9.4	0.014	2.31
Maga	Fresh	25.0	13.3	15.3	16.8	0.261	2.75
P8P9	Lower Trans	48.5	10.8	16.3	15.6	0.175	2.51
P8P9	Fresh	28.6	10.8	16.3	17.4	0.445	2.68
Siga E/W	Lower Trans	24.2	11.4	15.4	16.1	0.049	2.93
Siga E/W	Fresh	24.2	11.4	15.4	16.1	0.049	2.93
Siga S	Lower Trans	90.0	19.8	15.4	8.6	0.049	2.55
Siga S	Fresh	38.3	19.8	15.4	14.0	0.163	2.72
P16	Lower Trans	50.0	19.8	17.1	17.5	0.601	2.77

Block	Weathering Type	SMC Axb 85th %	CWi kWh/t	RWi kWh/t 85th %	BWi kWh/t 85th%	Ai g Ave	Ore SG Ave
P16	Fresh	50.0	19.8	17.1	17.5	0.601	2.77
P17	Lower Trans	28.2	19.8	17.1	15.8	0.601	2.78
P17	Fresh	28.2	19.8	17.1	15.8	0.601	2.78
Nobsin*	Lower Trans	50.1	19.8	16.1	14.8	0.205	2.56
Nobsin*	Fresh	26.4	19.8	16.1	17.0	0.666	2.90
LOM Design	14:86 Ratio†	30.8	16.8	16.3	15.7	0.374	2.79

* OMC assumed Nobsin ore properties from overall testwork for lower transition and fresh ore types

† Ratio of Transition to Fresh Ore

13.7.2 Gold Recovery

Historic Recovery Assessment

The 2019 feasibility study used the following gold recovery inputs for the estimation of the mineral resources and reserves.

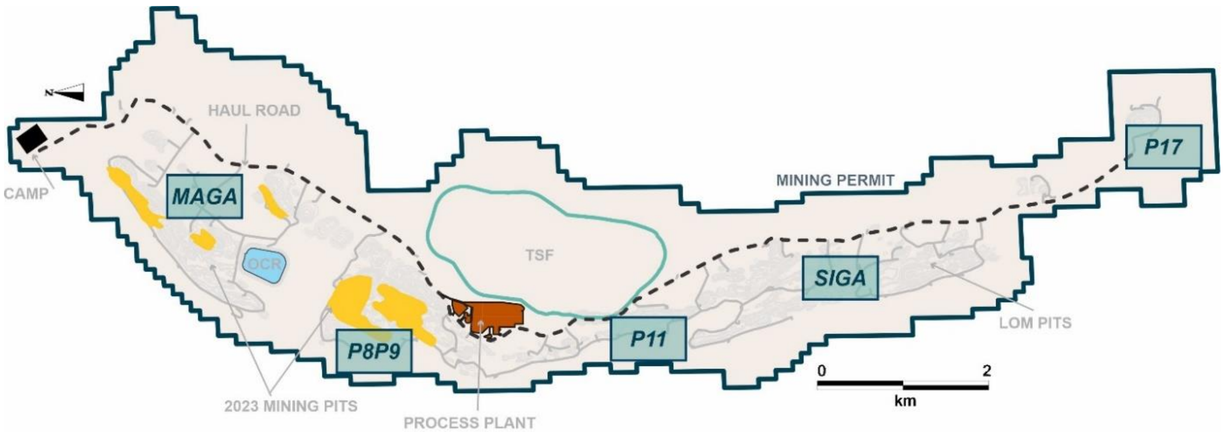
Table 13.7.3 Historic Gold Recovery Estimates

Ore Type	Resource Estimate Gold Recovery	Reserve Estimate Gold Recovery Equation (incl losses)
Upper Oxide	90.7%	(Head Grade – 0.078) / Head Grade x 100
Lower Oxide	88.4%	(Head Grade – 0.080) / Head Grade x 100
Upper Transition	86.0%	(Head Grade – 0.083) / Head Grade x 100
Lower Transition	82.5%	100% - 11.91 x $HG^{-0.38}$ – 1.05%
Fresh (except P17S)	81.7%	100% - 17.25 x $HG^{-0.24}$ – 1.05%
P17S	94%	96% - 1.05% = 94.95%

The values in Table 13.7.3 are based on leach residence times of 24 hours for the oxide and 48 hours for the fresh rock. The design grind size for the oxide rock was P_{80} 125 µm and the design grind size for the fresh rock was P_{80} 75 µm. The gold extraction model for the oxide plant was taken from testwork done by McClelland (2001) and SGS (2017). The gold extraction model for the hard rock plant was developed using drill core LeachWell assay results from Orezone database. As the LeachWell procedure employs an extended leach time, high cyanide concentration and powerful oxidizing agents to yield the best achievable gold recovery an additional 0.5% recovery reduction was included in the estimated losses.

The 2019 feasibility study did not differentiate fresh rock gold recovery between pits shown below in Figure 13.7.2, apart from P17. The metallurgical testwork, the mineralogy and the lithology of the deposits have shown that differences in some fresh rock recoveries can be expected.

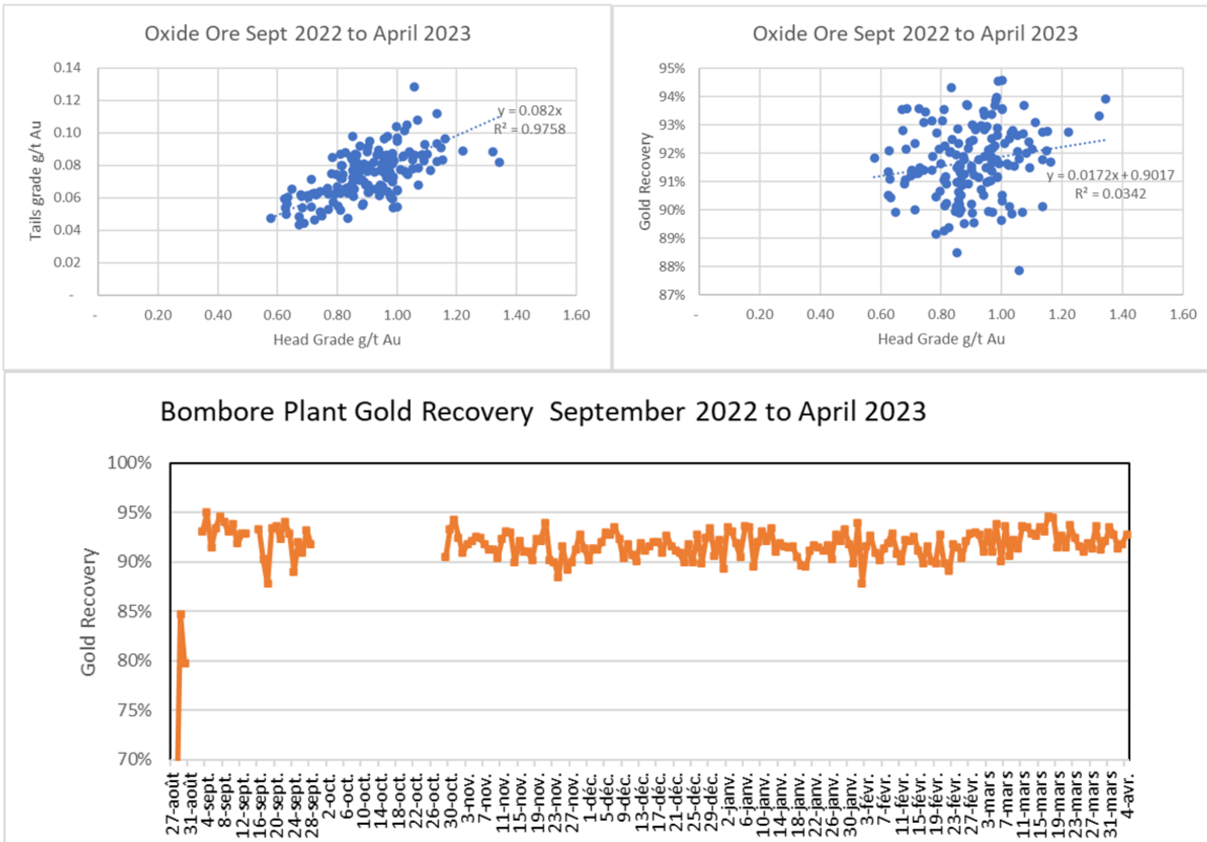
Figure 13.7.2 Mining Pit Locations



Existing Oxide Circuit Operation

Bomboré Gold Mine is currently processing oxide ore material at a rate of 5.9 Mtpa with a grind size of P_{90} 125 µm with a leach residence time of 21 hours. Since plant start up, the head grade has averaged 0.90 g/t and the tails grade has averaged 0.07 g/t (from September 2022 to the start of April 2023), Figure 13.7.3. The average gold recovery has been 91.8% and there is no obvious relationship between head grade and recovery. There is a very weak trend towards lower recovery from lower grade feed and no material has been processed at a grade lower than 0.6 g/t.

Figure 13.7.3 Oxide Plant Recovery (Sept 2022 to April 2023)



Exploration Data

The Bomboré exploration data base contains gold grades that are derived from a LeachWell assay technique which utilizes a cyanide leach of the pulverised drill core sample to determine cyanide extractable gold. The leach tails are then fire assayed to determine total gold grade. Therefore, the exploration database contains a very large number of cyanide soluble gold estimates which are a proxy for gold recovery. The LeachWell conditions do not replicate the actual plant, so LeachWell gold recovery data cannot be used without some adjustment. The exploration data is useful for recognising gold recovery trends and identifying refractory material.

The Siga and P8P9 deposits contain the most significant quantities of fresh ore and summary charts of LeachWell gold recovery values versus head grade are presented below.

Figure 13.7.4 Siga LeachWell Gold Extraction

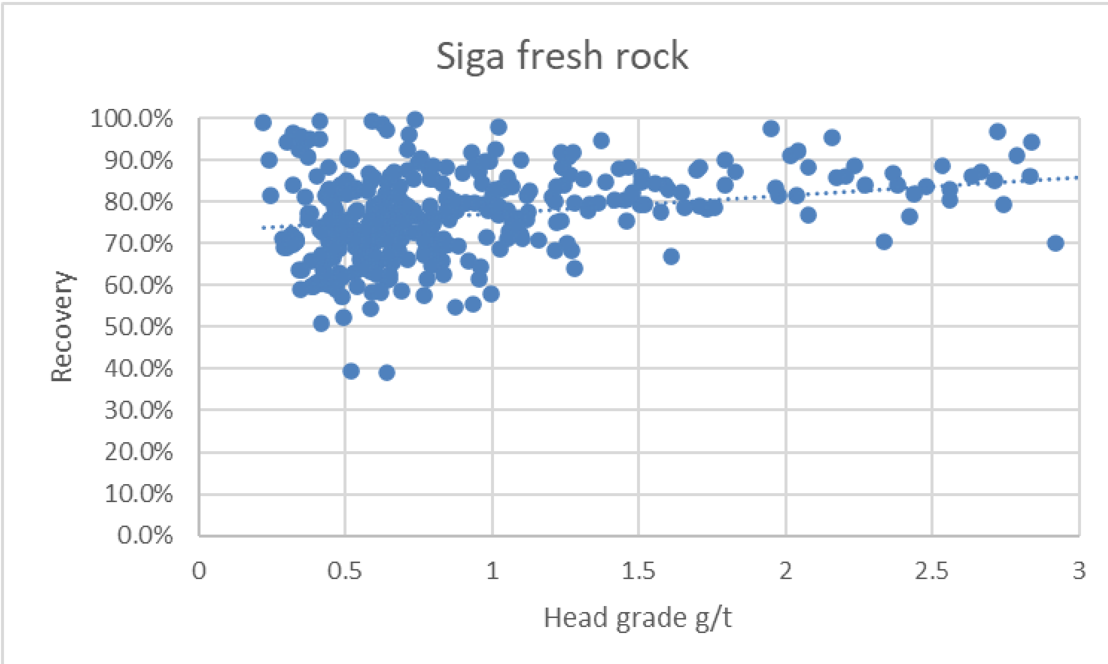


Figure 13.7.5 P8P9 LeachWell Gold Extraction

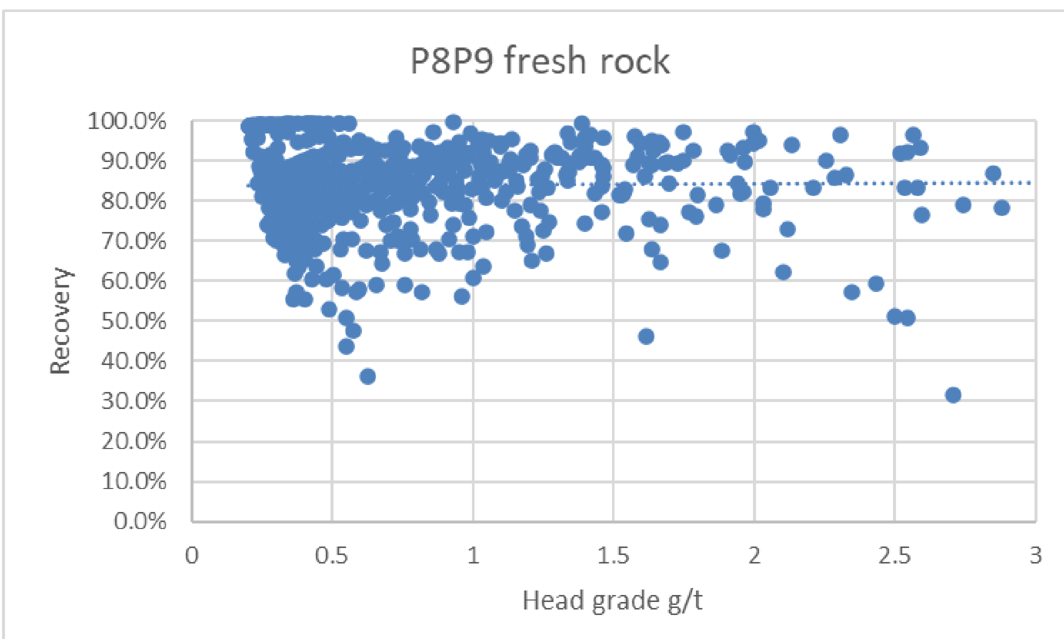


Figure 13.7.6 Exploration LeachWell Recovery Bin per Pit

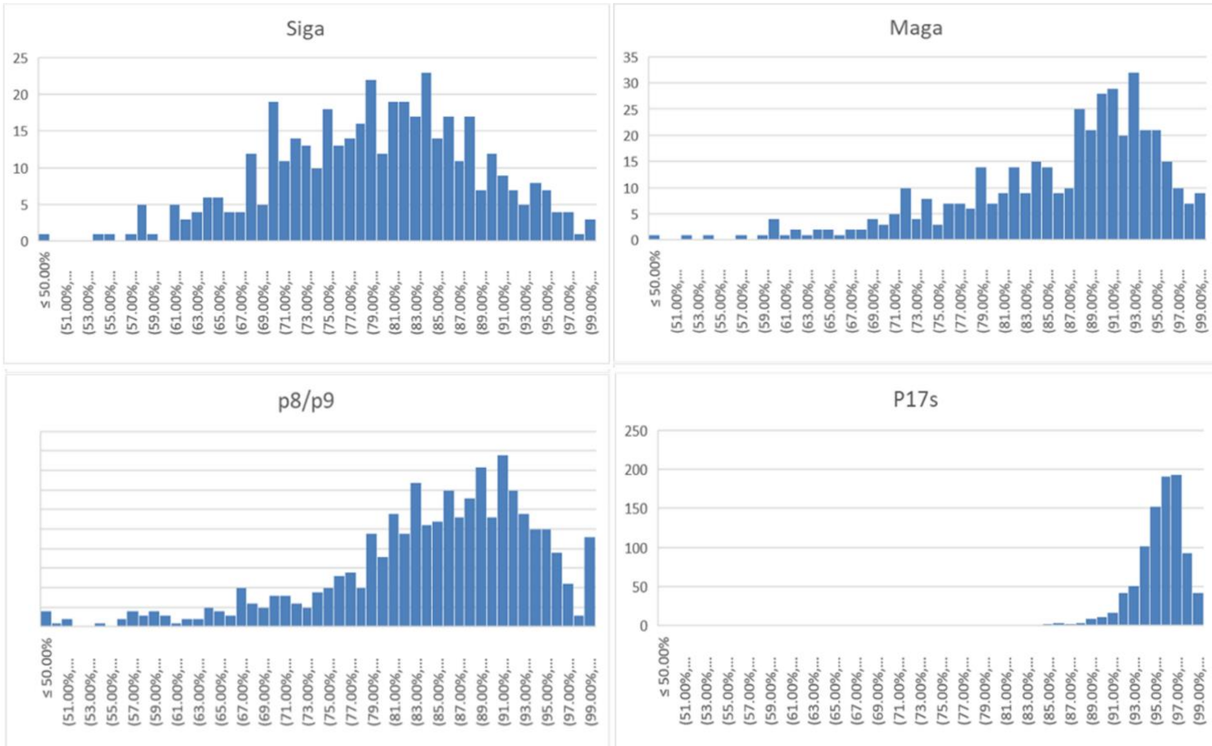


Figure 13.7.7 Exploration LeachWell Recovery Bin per Lithology

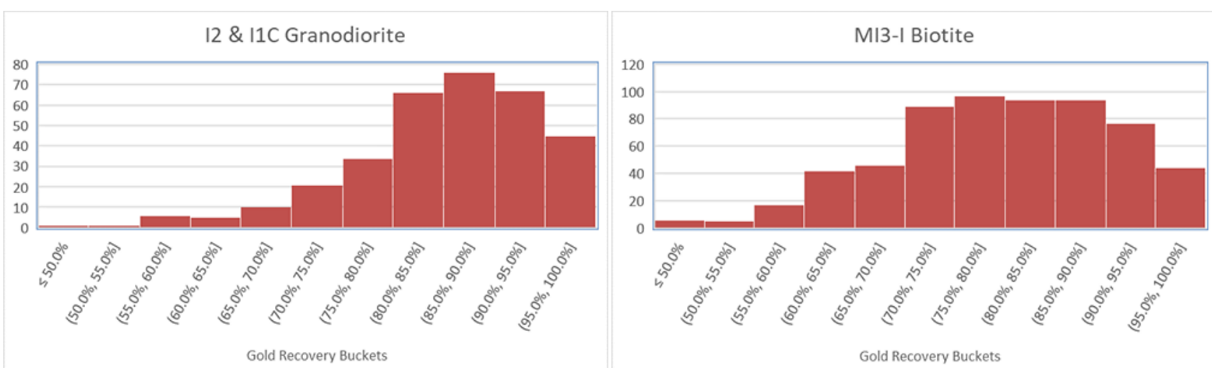


Figure 13.7.4 to Figure 13.7.7 show that there is no readily apparent relationship between gold recovery and gold grade in the fresh rock. The fresh rock gold recovery in Maga and P8P9 appears to be better than Siga, however the distribution is not precise. The gold recovery profile for P17S is clearly different from the other deposits with a consistent and high recovery. Gold recovery by fresh rock lithology does not exhibit any clear trends, however the granodiorite does have higher gold recoveries than the other lithologies, as it is the main host at P17S.

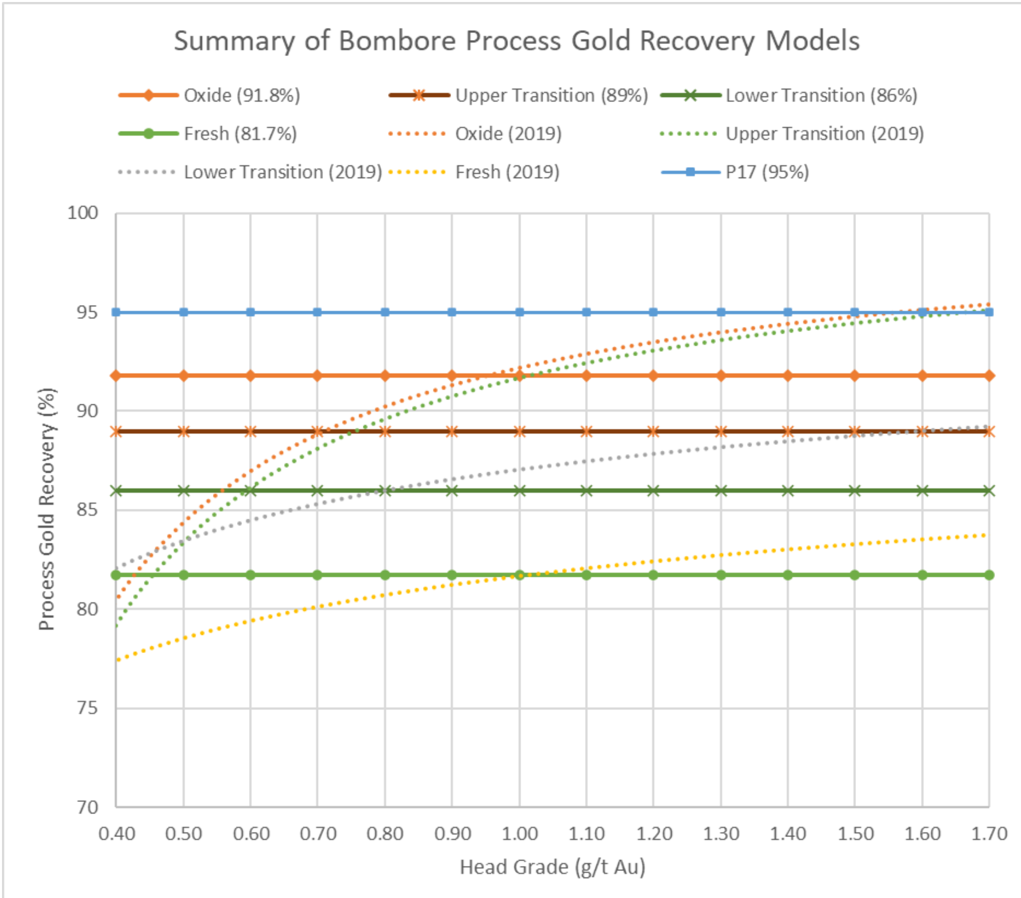
Feasibility Study Recovery Assessment

Extensive metallurgical testwork has been completed on Bomboré ore to confirm plant design parameters. The regolith, oxide and upper transition ore types will be processed in the existing oxide plant consisting of a single stage ball mill and CIL circuit. The lower transition and fresh ore types will be processed in the new hard rock plant consisting of a single stage SAG mill and CIL circuit. The gold recoveries anticipated from the ore types are presented in Table 13.7.4 below and compared with the previous recovery model in Figure 13.7.8 below.

Table 13.7.4 Recommended Gold Recovery % (including 1.05% solution losses)

Pit Zone	Oxide and Regolith	Upper Transition	Lower Transition	Fresh
Maga	91.8	89.0	86.0	81.7
P8P9	91.8	89.0	86.0	84.0
Siga	91.8	89.0	86.0	81.7
P11	91.8	89.0	86.0	81.7
P16	91.8	89.0	86.0	81.7
P17	95.0	95.0	95.0	95.0

Figure 13.7.8 Comparison of Old vs New Recovery Predictions



<u>Oxide Ore Recovery</u>

Current plant data show the plant has achieved an average oxide ore gold recovery of 91.8% independent of feed grade.

<u>Upper and Lower Transition Ore Gold Recovery</u>

SGS Lakefield conducted some leaching testwork in 2017 on oxide, upper transition, and lower transition material. The findings were that the Medium Grade upper transition recovery was one percentage point lower than oxide and the lower transition recovery was three percentage points lower than oxide (85.5% vs 84.8% vs 82.6%). The Base Met work in 2018/9 included leach testwork on P8P9 fresh rock and P8P9 lower transition samples. The average recoveries were 90.5% and 87.6% respectively.

Transition material gold recovery is expected to be lower than the oxide ore but higher than the fresh rock. The exploration LeachWell data was used to develop a head grade vs recovery curve for the 2019 feasibility study. The average gold recovery in the Siga lower transition ore was 85.5%. The thin and unpredictable weathering profile also makes the transition ore difficult to schedule in the mine plan. The recommendation is to use a simple single recovery value as the test data does not support a curve. The recommended gold recovery of 89% and 86% for the upper and lower transition ores respectively is based on an average head grade of 0.8 g/t and losses of 1.05%.

P17 Ore Recovery

The metallurgical testwork on the P17 orebodies (including P17S) supports a laboratory recovery of 96% to 97% on all ore types and all feed grades. After allowing for plant losses, the recommended gold recovery for P17 ore is 95%.

P11 and P16 Ore Recovery

There was no metallurgical testwork done on the P11 or P16 orebody since the 2012 McClelland study. The exploration LeachWell data shows that the P11 material is similar to the Siga material. The overall average LeachWell recovery was 87%. The recommendation is to assume the Siga values for P11 and P16 ore gold recovery.

Fresh Ore Gold Recovery

The 2019 feasibility study treated all fresh rock the same and applied a curve that increased gold recovery with head grade. The average fresh ore recovery reported was 81.7% for Siga, and Maga was slightly higher at 82.7% due to the higher grade.

The recent Maelgwyn testwork on Siga demonstrated that the gold recovery at a P_{80} 75 µm grind with a 24 hour leach appears to be independent of head grade and averages about 81.1%. The gold recovery at a grind of P_{80} 106 µm with a 24 hour leach exhibits a fixed tail increase of about 0.05 g/t, most likely due to insufficient liberation as a result of the coarse grind. The 106 µm grind equates to a recovery loss of about 5% at the average Siga grade of 1g/t compared to the 75 µm grind. Gold recovery does increase at longer leach times but not significantly. From 24 to 32 hours the recovery increased by about 1%. The Base Met metallurgical test work showed that a gold recovery of 82.7% could be expected from the Siga samples.

The proposed new hard rock plant is designed to deliver a grind size of P_{80} 75 µm and a leach time of 27.7 hours. The prior NI 43-101 study had an average fresh rock recovery of 81.7% for all pits at the average grade of 1 g/t and this value appears to be still valid for the proposed new hard rock plant with a 75 µm grind and a 27.7 hour leach time. The recommendation is to continue to use the previous NI 43-01 study derived recovery of 81.7% (which includes losses of 1.05%) for the Siga, P11, P16 and Maga fresh ore gold recovery.

The recommended gold recovery for the P8P9 fresh ore is 84%, derived from the LeachWell database average. The Base Met metallurgical testwork demonstrated that the P8P9 samples had higher gold recoveries than the Siga samples (90.5% vs 82.7%). Mineralogical examination of the P8P9 samples showed the presence of free pyrrhotite similar to P17S and this is associated with higher gold recovery.

13.7.3 Reagent Consumption

Oxide Ore

The oxide plant reagents consumption as given by current operational data indicates a 0.27 kg/t NaCN consumption and 1.65 kg/t Quicklime consumption compared the previous NI 43-101 report indicating 0.19 kg/t NaCN and 1.86 kg/t Quicklime consumption.

Fresh Ore

The data from the 2019 Base Met and 2023 Maelgwyn testwork was used as it closely conforms to the final plant design. The average consumptions were weighted with the quantity of fresh ore from 2022 preliminary mining estimates to derive a LOM average consumption, as given in Table 13.7.5 below.

Table 13.7.5 Fresh Ore LOM Reagents Consumption

2019 & 2023 Variability Testwork	NaCN kg/t	CaO kg/t	Fresh Resource	
			Weighting	%
Maga	0.46	1.32		12.3%
Siga	0.50	0.72		64.8%
P8P9	0.65	1.29		14.3%
P17S	0.44	0.80		8.6%
LOM Average	**0.51**	**0.88**		

Note the design LOM average cyanide consumption was derated by 10% to account for the difference of testwork and plant design leach residence time.

13.7.4 Summary of Metallurgical Design Criteria

A summary of the metallurgical inputs to the Oxide Plant and hard rock plant process design criteria are presented in Table 13.7.6 and Table 13.7.7, respectively.

Table 13.7.6 Summary of Metallurgical Criteria for Oxide Plant

Criteria	Units	Design	Notes / Source
Plant Throughput	tpa	5,900,000	Orezone
Ore Type	-	Upper & Lower Oxide Upper Transition	Mine plan (2022)
Design Ore Blend - Oxide	%	85.6	Mine plan (2022)
- Upper Transition	%	14.4	Mine plan (2022)
Head Grade - Gold (Design)	g/t Au	1.0	Lycopodium / Orezone
- Gold (LOM average)	g/t Au	0.67	Mine plan
Gold Recovery Estimation at 1 g Au/t			
- Oxide	%	91.8	Recovery plan
- Upper Transition	%	89.0	Recovery plan
- Per Design Ore Blend	%	91.4	Calculated
Ore Specific Density	t/m^3	2.8	Testwork
Ore Bulk Density	t/m^3	1.65	Lycopodium / Orezone
Crushing Work Index (CWi)	kWh/t	7.7	Testwork
Rod Mill Work Index (RWi)	kWh/t	5.7	Testwork
Bond Ball Mill Work Index (BWi)	kWh/t	4.8	Testwork
Bond Abrasion Index (Ai)	g	0.031	Testwork
Grind Size P_{80}	µm	125	Lycopodium
CIL Circuit Residence Time	hrs	24	Testwork
	hrs	21	Actual (Orezone)
CIL Slurry Density (for saprolitic ore)	% solids	~40%	Lycopodium
Sodium Cyanide Addition	kg/t NaCN	0.27	Orezone
Lime Addition	kg/t CaO	1.65	Orezone

Table 13.7.7 Summary of Metallurgical Criteria for Hard Rock Plant

Criteria	Units	Design	Notes / Source
Plant Throughput	tpa	4,400,000	Orezone
Design Ore Blend - Lower Transition	%	15.1	Mine plan (2022)
- Fresh	%	84.9	Mine plan (2022)
Head Grade - Gold (Design)	g/t Au	1.90	Lycopodium / Orezone
- Gold (LOM)	g/t Au	1.02	Mine plan (2022)
Gold Recovery Estimation			
- Lower Transition	%	86.0	Recovery plan

Criteria	Units	Design	Notes / Source
- LT & Fresh P17	%	95.0	Recovery plan
- Fresh P8P9	%	84.0	Recovery plan
- Fresh	%	81.7	Recovery plan
- LOM Design	%	83.5	Calculated
Ore Specific Density	t/m^3	2.79	Testwork
Ore Bulk Density	t/m^3	1.65	Lycopodium / Orezone
Crushing Work Index (CWi)	kWh/t	16.8	Testwork
Rod Mill Work Index (RWi)	kWh/t	16.3	Testwork
Bond Ball Mill Work Index (BWi)	kWh/t	15.7	Testwork
A x b Parameter		30.8	Testwork
Bond Abrasion Index (Ai)	g	0.374	Testwork
Grind Size P_{80}	µm	75	Testwork
CIL Circuit Residence Time	hrs	24	Testwork
CIL Slurry Density	% Solids	~45%	Testwork
Pre-Leach Thickener Solids Loading	$t/m^2 \cdot h$	1.00	Testwork
Sodium Cyanide Addition	kg/t NaCN	0.51	Testwork
Quicklime Addition	kg/t CaO	0.88	Testwork

14.0 MINERAL RESOURCE ESTIMATE

14.1 Summary

The Bomboré Project Mineral Resource Estimate encompasses seven zones: B1, B2, P11, Siga, P16, P17N and P17 (Figure 14.1.1). A total of 378 individual mineralization domains have been incorporated within the updated Mineral Resource Estimate. Geological models for each of the seven zones were created by Orezone, and then audited by the Authors. Most of these models overlap with the neighbouring models, but the Mineral Resources reported from each model in this Technical Report are restricted to reporting limits that are complementary at the Property scale (Figure 14.1.1). Block models for P17, B1 and B2 were developed by Orezone and then audited by the Authors. Block models for the P11, Siga, P16 and P17N Zones were developed by the Authors who also generated the USD$1,700/oz gold pit shells constraining the current Mineral Resource Estimates.

Block grades for gold were estimated for each mineralization domain by Inverse Distance Cubed (ID3) weighting of capped composites using a minimum of three, four or five composites and a maximum of twelve composites. Sample selection was restricted to a maximum of four or five composite samples from a single drill hole. The orientation of the search ellipsoids within each individual mineralization domain were defined by Orezone geologists based on the local geology. Ordinary Kriging (OK) estimates were also developed for comparison, and Nearest Neighbour (NN) models were used for validation. The ID3 methodology was selected for Mineral Resource reporting, since many of the variograms developed for the mineralization domains were not of sufficient quality for use with OK. Issues associated with the variogram modelling included high nugget contributions, pronounced drill hole effects associated with different lag distances, and multiple directions of anisotropy.

The updated 2023 Mineral Resource Estimate, before taking into account stockpiles of unprocessed material, contains 4.5 million Measured and Indicated gold ounces and 0.6 million Inferred gold ounces (Tables 14.1. and 14.2.).

In general, estimated blocks within 25 m of three or more drill holes are classified as Measured, blocks within 50 m of three or more drill holes are classified as Indicated, and additional estimated blocks are classified as Inferred. In some cases, peripheral blocks within the defined veins are classified as Exploration Potential and are not included in the Mineral Resource Estimate.

Mineral Resources are reported within optimized pit shells at the appropriately selected cut-off (Table 14.1.1). Orezone reports whole block volumes using only those blocks where the block centroid lies within the controlling wireframe. A factor has been applied to the oxide Mineral Resource Estimates to discount artisanal mining.

Table 14.1.1 Bomboré Mineral Resource Estimate Cut-Off Grades (Au g/t)

Cut-Off Grades (Au g/t)							
Unit	**B1**	**B2**	**P11**	**Siga**	**P16**	**P17N**	**P17**
Regolith	0.25	0.25	0.25	0.25	0.25	0.25	0.25
Oxide	0.25	0.25	0.25	0.25	0.25	0.25	0.25
Upper Transition	0.25	0.25	0.25	0.25	0.25	0.25	0.25
Lower Transition	0.45	0.45	0.45	0.45	0.45	0.45	0.45
Fresh	0.45	0.45	0.45	0.45	0.45	0.45	0.45

Figure 14.1.1 Bomboré Mineral Resource Estimate Zones

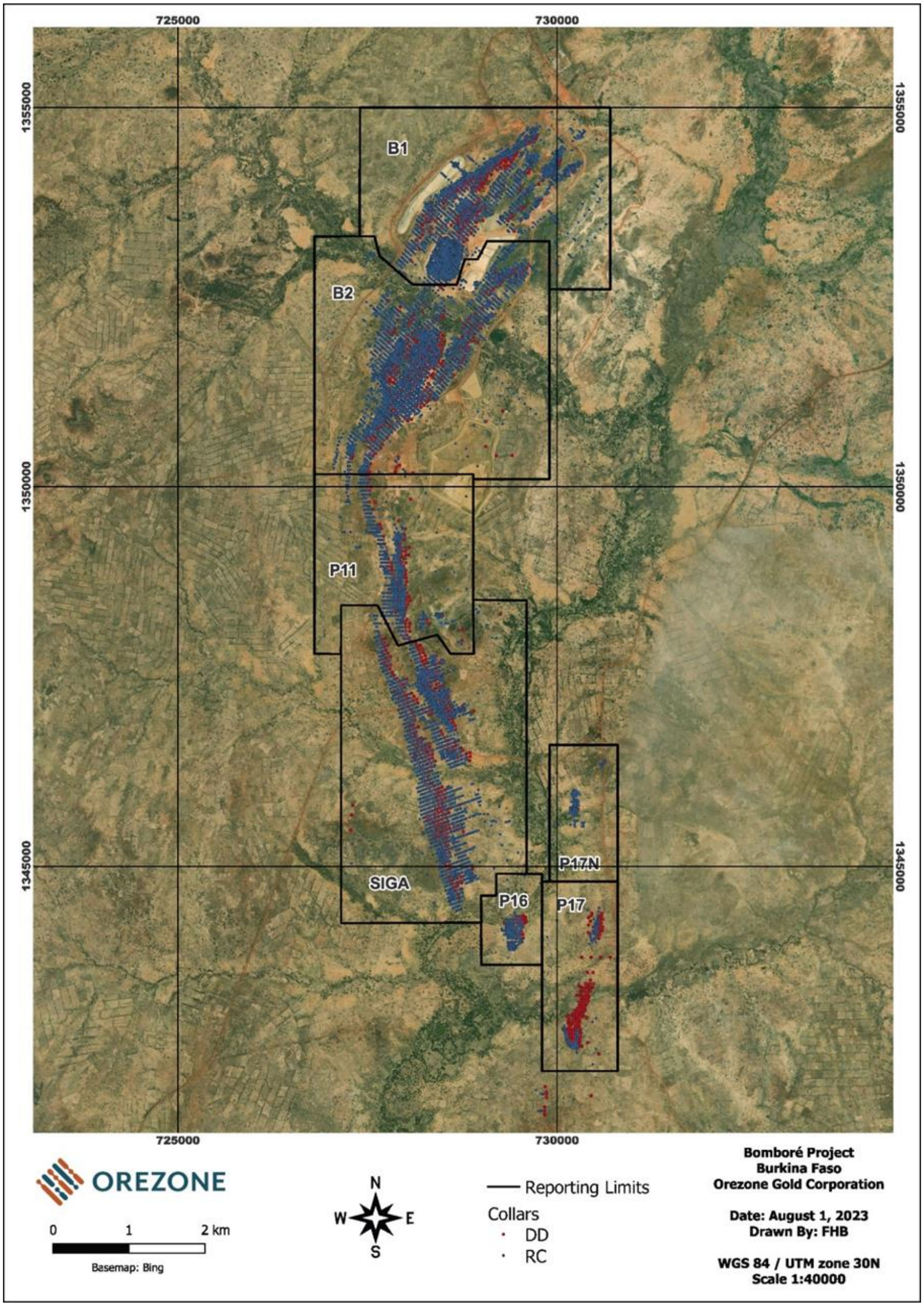


Table 14.1.2 Bomboré Mineral Resource Estimate Details

Total	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	NA	27,530	0.78	692
Indicated	NA	151,735	0.78	3,814
Meas + Ind	NA	179,265	0.78	4,515
Inferred	NA	20,015	0.95	610

Regolith	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.25	5	0.29	0
Indicated	0.25	8	0.48	0
Meas + Ind	0.25	13	0.41	0
Inferred	0.25	0.2	0.39	0

Oxide	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.25	10,162	0.57	178
Indicated	0.25	49,451	0.54	855
Meas + Ind	0.25	59,613	0.54	1,042
Inferred	0.25	2,435	0.55	43

Trans Upper	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.25	6251	0.62	125
Indicated	0.25	23,470	0.60	456
Meas + Ind	0.25	29,721	0.61	581
Inferred	0.25	843	0.63	17

Trans Lower	Au Cut-off (g/t)	Tonnes (k)	Au (g/t)	Au (koz)
Measured	0.45	3,228	0.92	95
Indicated	0.45	12,117	0.94	365
Meas + Ind	0.45	15,345	0.93	460
Inferred	0.45	463	0.92	14

Fresh	Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.45	7,883	1.16	294
Indicated	0.45	66,690	1.00	2,139
Meas + Ind	0.45	74,573	1.01	2,433
Inferred	0.45	16,274	1.02	536

Notes for Table 14.1.2

"Oxide" includes Regolith, Oxide and Transitional Upper units reported at a cut-off of 0.25g/t Au.

"Hard Rock" includes Transitional Lower and Fresh units reported at a cut-off of 0.45g/t Au.

Mineral resources, which are not mineral reserves, do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues.

Mineral resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.

The inferred mineral resource in this estimate has a lower level of confidence than that applied to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of the inferred mineral resource could be upgraded to an indicated mineral resource with continued exploration.

Totals may differ due to rounding.

Mineral resources are reported within an optimized pit shell at a gold price of $1,700/troy oz.

Mineral resources are inclusive of mineral reserves, however, exclude ore stockpiles.

The mineral resource estimates include oxide grade reduction factors applied by Orezone based on recent mine to mill reconciliation data.

Table 14.1.3 Bomboré Mineral Resource Estimate Summary

Category	Cut-off	Measured			Indicated			Measured + Indicated			Inferred		
		Tonnes	Au Grade	Contained Gold	Tonnes	Grade	Contained Gold	Tonnes	Grade	Contained Gold	Tonnes	Grade	Contained Gold
	g/t	k	g/t	koz	k	g/t	koz	k	g/t	koz	k	g/t	koz
Oxides	0.25	16,419	0.59	303	72,928	0.56	1,311	89,347	0.57	1,623	3,278	0.57	60
Hard-rock	0.45	11,111	1.09	389	78,807	0.99	2,503	89,918	1.00	2,892	16,737	1.02	549
Total		27,530	0.79	701	151,735	0.78	3,814	179,265	0.78	4,515	20,015	0.95	610

"Oxide" includes Regolith, Oxide and Transitional Upper units reported at a cut-off of 0.25g/t Au.

"Hard Rock" includes Transitional Lower and Fresh units reported at a cut-off of 0.45g/t Au.

Mineral resources, which are not mineral reserves, do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues.

Mineral resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.

The inferred mineral resource in this estimate has a lower level of confidence than that applied to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of the inferred mineral resource could be upgraded to an indicated mineral resource with continued exploration.

Totals may differ due to rounding.

Mineral resources are reported within an optimized pit shell at a gold price of $1,700/troy oz.

Mineral resources are inclusive of mineral reserves, however, exclude ore stockpiles.

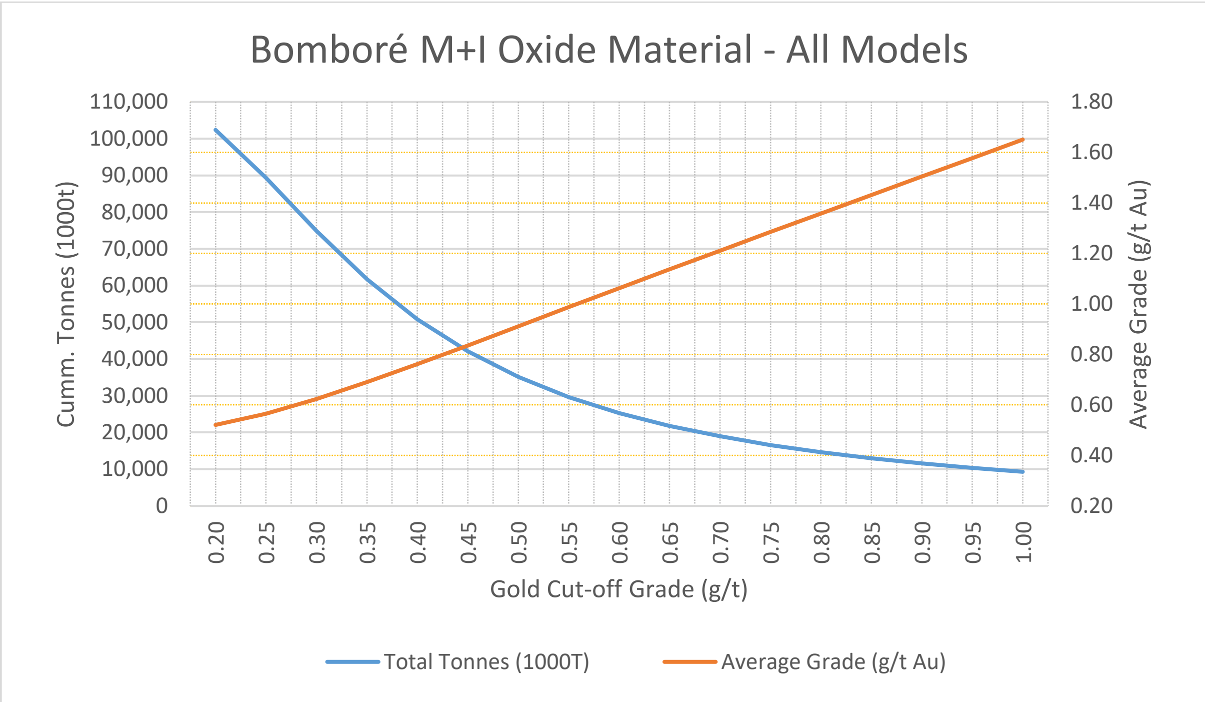
The mineral resource estimates include oxide grade reduction factors applied by Orezone based on recent mine to mill reconciliation data.

The sensitivity of the Mineral Resource block grades to changes in cut-off grade was evaluated by calculating the total Mineral Resource inventory at various cut-offs within the Mineral Resource pit for the Measured and Indicated Resources (Table 14.1.41 and Figure 14.1.2) and the Inferred Resources (Table 14.1.5 and Figure 14.1.3). The values presented are intended for the sole purpose of demonstrating the sensitivity of the Mineral Resource with respect to the cut-off grade.

Table 14.1.4 Grade Tonnage Sensitivity Table – Measured and Indicated Mineral Resources

OXIDE MATERIAL MEASURED + INDICATED				HARD ROCK MATERIAL MEASURED + INDICATED		
Au Cut-Off	Tonnes	Au		Au Cut-Off	Tonnes	Au
g/t	k	g/t		g/t	k	g/t
0.20	102,380	0.52		0.20	155,495	0.72
0.25	89,347	0.56		0.25	143,848	0.76
0.30	74,892	0.62		0.30	129,530	0.81
0.35	61,677	0.69		0.35	115,148	0.87
0.40	50,781	0.76		0.40	101,856	0.93
0.45	42,070	0.83		0.45	89,918	1.00
0.50	35,110	0.91		0.50	79,310	1.07
0.55	29,628	0.99		0.55	70,097	1.14
0.60	25,296	1.06		0.60	62,148	1.22
0.65	21,804	1.14		0.65	55,237	1.29
0.70	18,966	1.21		0.70	49,287	1.36
0.75	16,592	1.28		0.75	44,050	1.44
0.80	14,635	1.36		0.80	39,523	1.52
0.85	12,967	1.43		0.85	35,536	1.59
0.90	11,558	1.50		0.90	32,121	1.67
0.95	10,351	1.58		0.95	29,169	1.75
1.00	9,302	1.65		1.00	26,573	1.82

Figure 14.1.2 Grade Tonnage vs. COG Charts – Measured and Indicated Mineral Resources



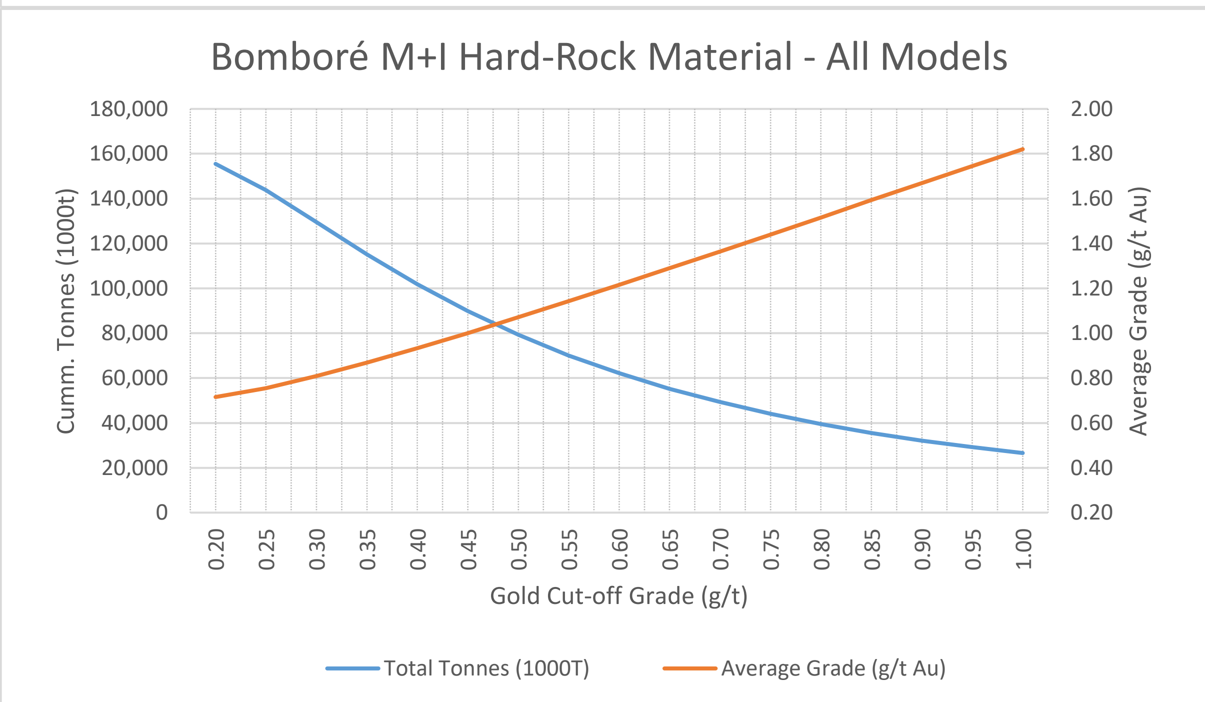
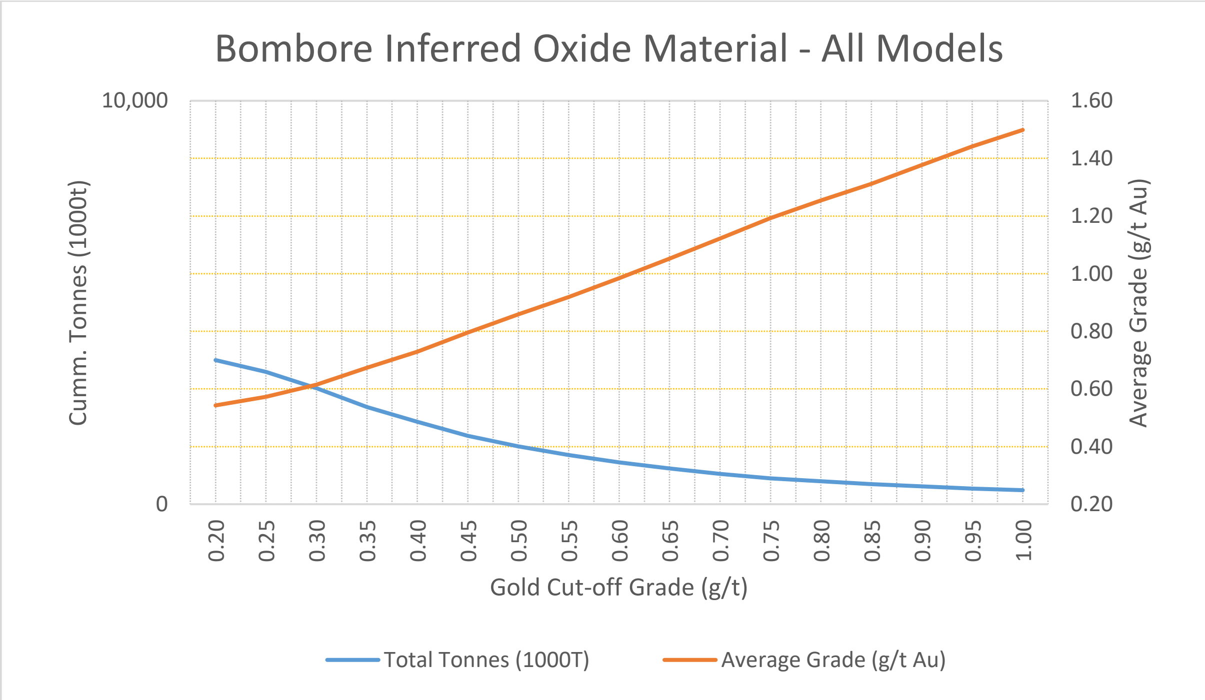
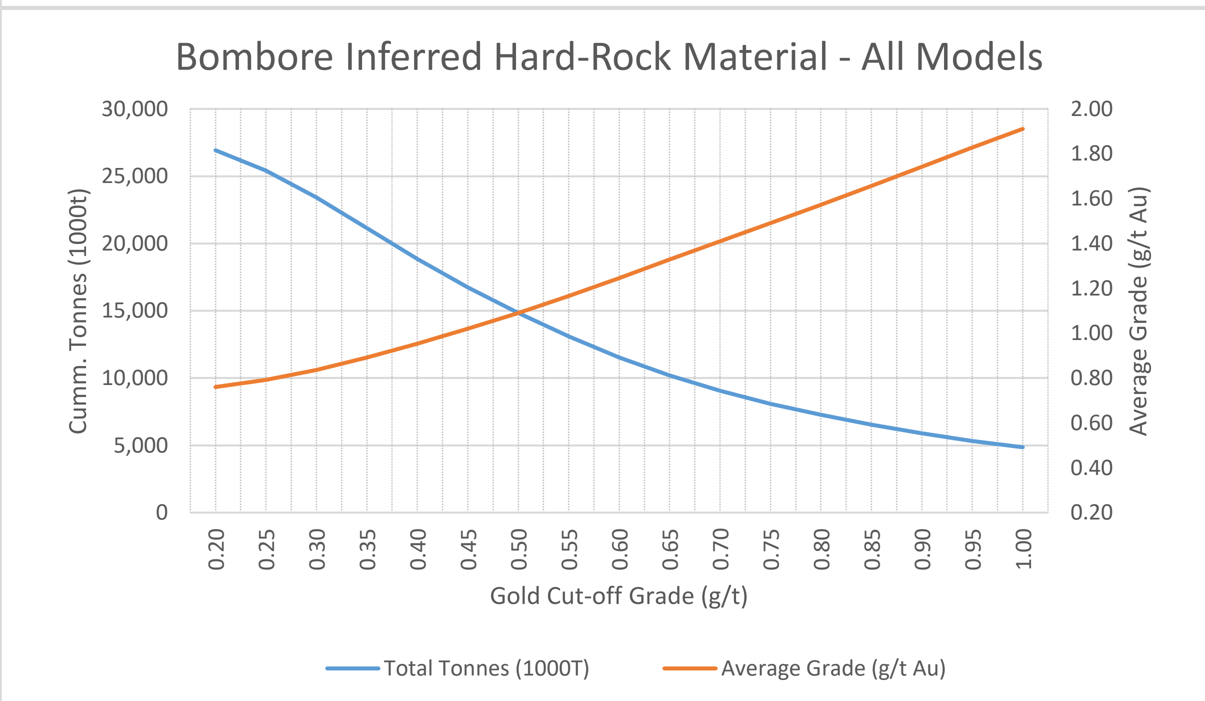


Table 14.1.5 Grade Tonnage Sensitivity Table – Inferred Mineral Resources

OXIDE MATERIAL INFERRED				HARD ROCK MATERIAL INFERRED		
Au Cut-Off	**Tonnes**	**Au**		**Au Cut-Off**	**Tonnes**	**Au**
g/t	k	g/t		g/t	k	g/t
0.20	3,572	0.54		0.20	26,927	0.76
0.25	3,278	0.57		0.25	25,419	0.79
0.30	2,879	0.61		0.30	23,414	0.84
0.35	2,410	0.67		0.35	21,136	0.89
0.40	2,044	0.73		0.40	18,857	0.95
0.45	1,697	0.80		0.45	16,736	1.02
0.50	1,434	0.86		0.50	14,843	1.09
0.55	1,223	0.92		0.55	13,100	1.17
0.60	1,037	0.98		0.60	11,534	1.25
0.65	883	1.05		0.65	10,187	1.33
0.70	752	1.12		0.70	9,046	1.41
0.75	644	1.19		0.75	8,089	1.49
0.80	567	1.25		0.80	7,269	1.57
0.85	502	1.31		0.85	6,533	1.66
0.90	440	1.38		0.90	5,892	1.74
0.95	389	1.44		0.95	5,334	1.83
1.00	348	1.50		1.00	4,860	1.91

Figure 14.1.3 Grade Tonnage vs. COG Charts – Inferred Mineral Resources





14.2 Introduction

14.2.1 Mineral Resource Estimates

The Mineral Resource Estimate presented herein has been prepared following the guidelines of the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1 and in conformity with generally accepted "CIM Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines. Mineral Resources have been classified in accordance with the "CIM Standards on Mineral Resources and Reserves: Definition and Guidelines" as adopted by CIM Council on 10 May 2014:

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no guarantee that all or any part of the Mineral Resource will be converted into a Mineral Reserve. Confidence in the estimate of Inferred Mineral Resources is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure.

The Authors are not aware of any known permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource Estimate. The Authors do note that restrictions imposed by the current environmental permit may impact on open pit mining activities within the local flood plain, potentially reducing access to portions of the Mineral Resource. The Authors also note that Orezone selected slightly more conservative cut-off gold grades for reporting pit constrained Mineral Resources compared to recent operational costs.

All Mineral Resource estimation work reported herein was carried out or reviewed by Fred Brown, P.Geo., Eugene Puritch, P.Eng., FEC, CET and Antoine Yassa P.Geo., all independent Qualified Persons as defined by National Instrument 43-101 by reason of education, affiliation with a professional association and past relevant work experience.

Portions of the background information and technical data for this Technical Report were obtained from previously filed National Instrument 43-101 Technical Reports. The Authors have assumed that previous companies' reports, maps and other data are complete and accurate. Orezone has reviewed a draft version of this Report for factual errors prior to issue.

Mineral Resource modelling was carried out using Leapfrog GEO™. Mineral Resource estimation was carried out using GEMS™ and Surpac™ software. Variography was carried out using Snowden Supervisor™. Open-pit optimization was carried out using the NPV SchedulerTM software program.

The effective date of this Mineral Resource Estimate is March 28, 2023.

14.3 Bomboré B1 Zone

14.3.1 Data Supplied

The B1 Zone model was developed by Orezone and reviewed by the Authors. Topography, mineralization, lithology and oxidation state three-dimensional wireframes were created by Orezone using Leapfrog Geo™ 2022.1.1. Drill hole distances are reported in metres and gold grades are reported as g/t. The collar coordinates were provided in the WGS84 Zone 30N UTM coordinate reference system. The supplied drill hole database for the B1 Zone contains 3,366 unique collar records (Table 14.3.1 and Figure 14.3.1).

Table 14.3.1 B1 Zone Drill Hold Database Summary

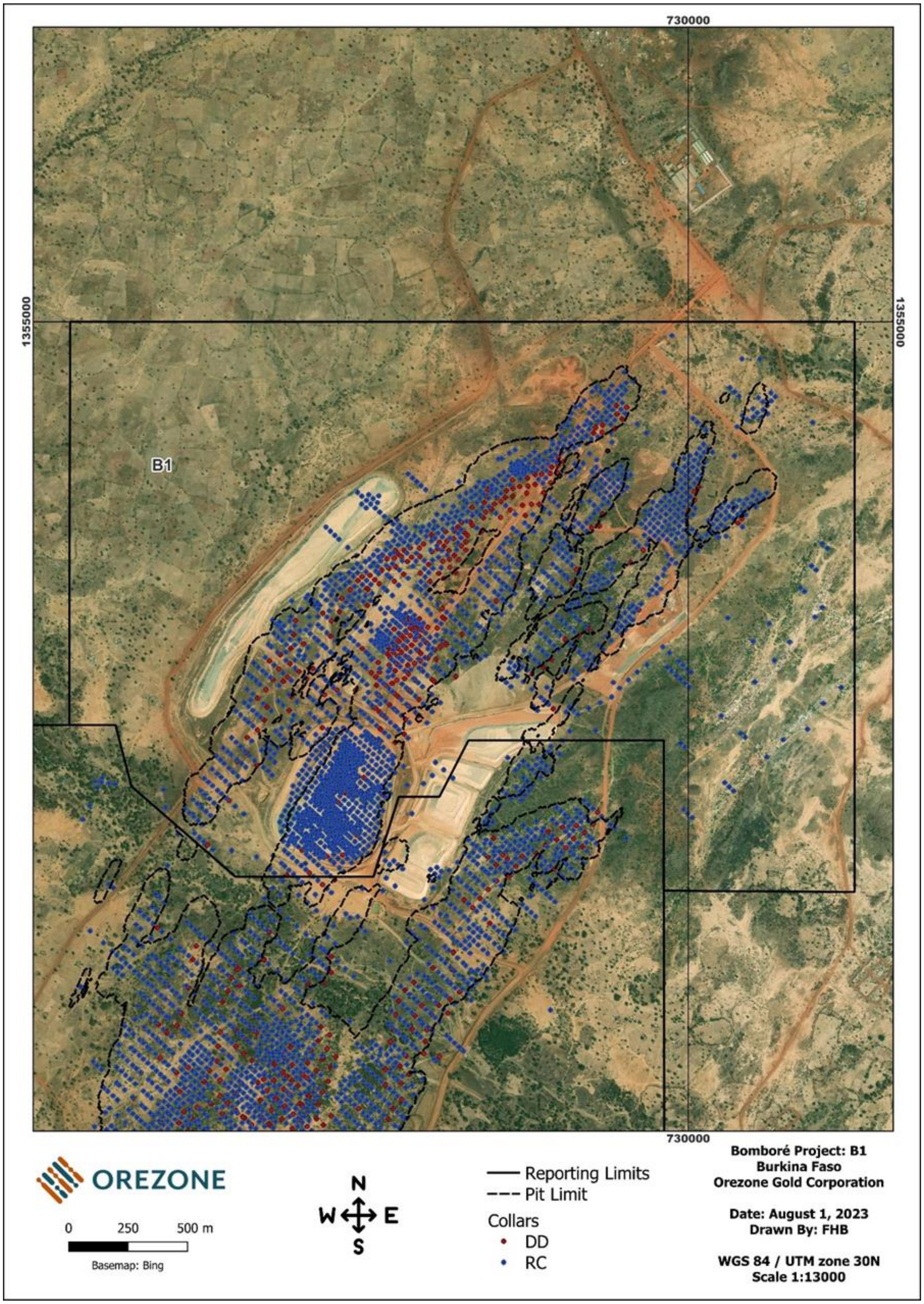
Type	Count	Metres
Pressure Metre Drill Hole	3	46.5
Channel	7	15.5
Diamond Core Drill Hole (DD)	311	51,384.0
Water Drill Hole	8	700.0
Auger drill hole	473	2,551.0

Type	Count	Metres
Rotary Air Blast Drill Hole (RAB)	169	7,754.0
Reverse Circulation Drill Hole (RC)	2,212	131,204.0
Trench	183	1,990.0
Total	3,366	195,645.0

Industry standard validation checks were carried out on the supplied databases, and minor corrections made where necessary. The Authors typically validate a Mineral Resource database by checking for inconsistencies in naming conventions or analytical units, duplicate entries, interval, length or distance values less than or equal to zero, blank or zero-value assay results, out-of-sequence intervals, intervals or distances greater than the reported drill hole length, inappropriate collar locations, and missing interval and coordinate fields.

No significant errors were noted with the supplied databases. The Authors consider that the drill hole database supplied is suitable for Mineral Resource estimation. The drill hole data were imported into a Surpac™ format MS-Access™ database, and imported wireframes were assigned a unique rock code. The supplied wireframes were used to back-tag the assay, bulk density and composite tables with mineralization domain, oxide state and lithology codes.

Figure 14.3.1 B1 Zone Drill Hole Plan View

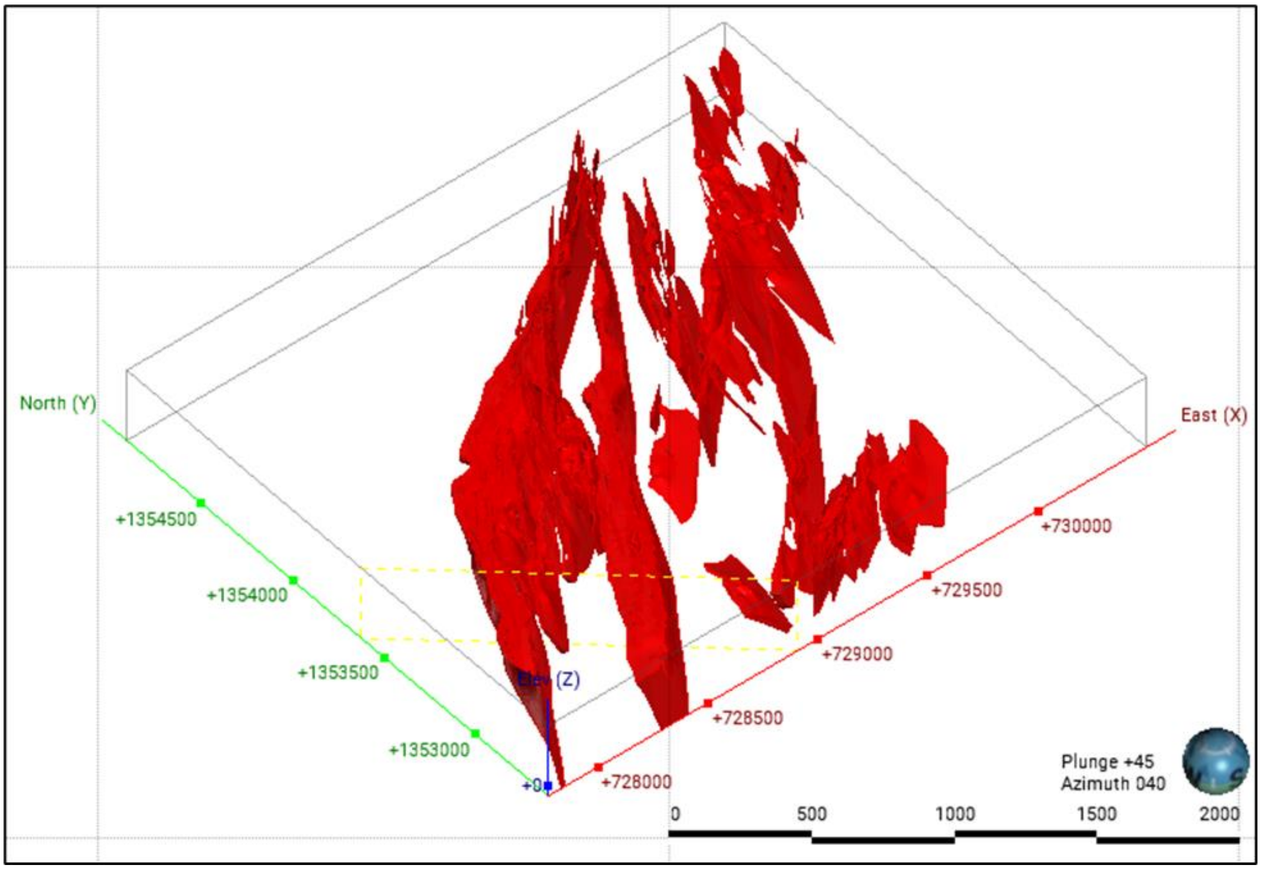


14.3.2 Exploratory Data Analysis

A total of 85 distinct mineralization domain wireframes were supplied (Figure 14.3.2). Each domain was assigned a unique three-digit rock code. Of the 85 domains, four domains are responsible for 79% of the Mineral Resource Estimate by volume: MB_B1_01 (Rock Code 101), MB_B1_13 (Rock Code 113), MB_B1_20 (Rock Code 120), and MB_B1_32 (Rock Code 132).

The average nearest-neighbour drill hole collar distance for DD and RC is 21.3 m, the average DD length is 165.2 m and the average RC drill hole length is 59.3 m. Summary statistics for the supplied assay data are provided below (Table 14.3.2).

Figure 14.3.2 Isometric View of B1 Zone Mineralization Domains



View looking along azimuth 40 degrees

Table 14.3.2 B1 Zone Assay Summary Statistics (DD and RC)

Domain	Rock Code	Count	Average Length m	Minimum Length m	Maximum Length m	Average Au g/t	Minimum Au g/t	Maximum Au g/t	Std Dev g/t	CoV
Unconstrained		102,067	1.05	0.50	9.00	0.086	0.001	28.031	0.227	0.05
MB_B1_00	100	86	1.03	1.00	2.00	0.675	0.007	4.853	0.847	1.25
MB_B1_01	101	7,380	1.07	0.70	5.00	0.863	0.001	93.390	1.806	2.09
MB_B1_02	102	78	1.02	1.00	1.50	0.539	0.004	12.757	1.454	2.68
MB_B1_03	103	431	1.05	0.50	2.50	0.345	0.011	2.976	0.365	1.06
MB_B1_04	104	10	1.00	1.00	1.00	2.148	0.216	5.794	1.962	0.87
MB_B1_05	105	1,533	1.15	0.50	5.50	0.637	0.005	9.786	0.688	1.08
MB_B1_06	106	1,591	1.04	0.50	4.00	1.001	0.009	17.618	1.953	1.95
MB_B1_07	107	477	1.05	0.50	3.00	0.624	0.015	9.038	0.887	1.42
MB_B1_08	108	39	1.05	0.80	2.00	0.354	0.043	2.630	0.430	1.20
MB_B1_09	109	2,068	1.08	0.70	5.00	0.943	0.005	40.344	2.266	2.41
MB_B1_10	110	442	1.18	0.80	3.50	0.715	0.007	21.319	1.713	2.39
MB_B1_11	111	474	1.09	0.50	2.50	1.037	0.011	20.214	1.854	1.78
MB_B1_12	112	717	1.08	0.45	4.50	0.989	0.017	49.729	2.649	2.68
MB_B1_13	113	25,805	1.04	0.50	7.50	0.378	0.001	46.897	0.619	1.64
MB_B1_14	114	28	1.13	1.00	3.00	2.861	0.031	39.200	7.492	2.57
MB_B1_15	115	53	1.13	1.00	3.00	0.444	0.078	1.891	0.389	0.87
MB_B1_16	116	16	1.78	1.00	4.00	0.822	0.153	1.953	0.622	0.73
MB_B1_17	117	117	1.06	1.00	2.00	0.567	0.022	7.644	0.863	1.51
MB_B1_18	118	86	1.19	1.00	3.00	0.475	0.005	5.620	0.754	1.58
MB_B1_19	119	66	1.24	1.00	2.55	0.443	0.009	2.654	0.483	1.08
MB_B1_20	120	8,444	1.06	0.70	3.00	0.433	0.001	66.261	1.205	2.78

Domain	Rock Code	Count	Average Length m	Minimum Length m	Maximum Length m	Average Au g/t	Minimum Au g/t	Maximum Au g/t	Std Dev g/t	CoV
MB_B1_21	121	618	1.05	1.00	3.00	0.428	0.020	4.785	0.488	1.14
MB_B1_22	122	253	1.06	1.00	2.00	0.328	0.023	2.502	0.278	0.85
MB_B1_23	123	578	1.03	0.90	2.00	0.331	0.014	11.042	0.527	1.59
MB_B1_24	124	203	1.02	0.80	2.00	0.449	0.023	4.169	0.557	1.24
MB_B1_25	125	626	1.00	0.85	1.20	0.609	0.028	14.593	1.131	1.86
MB_B1_26	126	239	1.00	1.00	1.00	0.350	0.013	3.460	0.342	0.98
MB_B1_27	127	432	1.00	1.00	1.00	0.406	0.027	4.066	0.387	0.95
MB_B1_28	128	269	1.00	1.00	1.00	0.412	0.011	4.828	0.528	1.28
MB_B1_29	129	227	1.01	0.85	1.50	0.857	0.015	50.804	3.623	4.22
MB_B1_30	130	130	1.00	1.00	1.00	0.331	0.056	1.256	0.233	0.70
MB_B1_31	131	115	1.01	1.00	1.50	0.392	0.061	3.958	0.439	1.11
MB_B1_32	132	7,647	1.03	0.60	4.50	0.343	0.005	27.714	0.598	1.74
MB_B1_33	133	34	1.00	1.00	1.00	0.640	0.029	3.520	0.700	1.08
MB_B1_34	134	42	1.00	1.00	1.00	1.035	0.003	10.631	1.792	1.71
MB_B1_35	135	57	1.00	1.00	1.00	0.44	0.012	2.971	0.486	1.09
MB_B1_36	136	587	1.00	0.80	1.00	0.670	0.006	77.395	3.328	4.96
MB_B1_37	137	618	1.01	0.70	3.00	0.640	0.012	64.402	2.919	4.55
MB_B1_38	138	609	1.01	0.70	2.00	0.594	0.008	7.383	0.732	1.24
MB_B1_39	139	817	1.00	0.90	1.50	0.438	0.001	8.968	0.560	1.28
MB_B1_40	140	1,013	1.00	1.00	3.00	0.594	0.006	138.857	4.398	7.40
MB_B1_41	141	21	1.00	1.00	1.00	0.694	0.017	3.658	0.824	1.16
MB_B1_42	142	48	1.00	1.00	1.00	0.742	0.030	5.759	1.076	1.11
MB_B1_43	143	41	1.00	1.00	1.00	0.508	0.028	2.205	0.447	0.87

Domain	Rock Code	Count	Average Length m	Minimum Length m	Maximum Length m	Average Au g/t	Minimum Au g/t	Maximum Au g/t	Std Dev g/t	CoV
MB_B1_44	144	96	1.02	1.00	2.00	0.548	0.048	5.158	0.727	1.32
MB_B1_45	145	124	1.01	0.80	2.00	0.541	0.067	3.685	0.612	1.13
MB_B1_46	146	241	1.00	0.80	1.50	0.544	0.006	16.803	1.213	2.23
MB_B1_47	147	87	1.02	1.00	1.60	0.410	0.016	4.061	0.505	1.23
MB_B1_48	148	191	1.00	1.00	1.00	0.395	0.026	2.797	0.429	1.08
MB_B1_49	149	21	1.10	1.00	3.00	0.412	0.154	0.962	0.199	0.47
MB_B1_50	150	11	1.00	1.00	1.00	0.413	0.043	0.882	0.264	0.61
MB_B1_51	151	67	1.00	1.00	1.00	1.256	0.007	17.429	2.388	1.89
MB_B1_52	152	122	1.00	1.00	1.00	0.348	0.036	1.419	0.263	0.75
MB_B1_53	153	86	1.02	1.00	2.00	0.635	0.016	4.881	0.853	1.34
MB_B1_54	154	48	1.00	1.00	1.00	0.380	0.027	1.248	0.246	0.64
MB_B1_55	155	10	1.00	1.00	1.00	1.168	0.015	4.908	1.398	1.14
MB_B1_56	156	58	1.00	1.00	1.00	0.719	0.014	2.380	0.612	0.84
MB_B1_57	157	52	1.00	1.00	1.00	1.457	0.009	26.625	3.835	2.61
MB_B1_58	158	24	1.00	1.00	1.00	0.592	0.035	1.958	0.580	0.96
MB_B1_59	159	100	1.00	1.00	1.00	0.454	0.061	2.350	0.484	1.06
MB_B1_60	160	168	1.00	1.00	1.00	0.333	0.006	1.641	0.275	0.82
MB_B1_61	161	121	1.04	1.00	2.50	0396	0.042	2.060	0.355	0.89
MB_B1_62	162	102	1.04	1.00	2.00	0.346	0.017	1.935	0.313	0.90
MB_B1_63	163	56	1.09	1.00	2.00	0.507	0.091	2.065	0.428	0.84
MB_B1_64	164	61	1.00	1.00	1.00	0.278	0.037	1.266	0.233	0.83
MB_B1_65	165	114	1.00	1.00	1.50	0.433	0.070	15.755	1.468	3.38
MB_B1_66	166	163	1.00	1.00	1.00	0.404	0.029	6.332	0.627	1.55

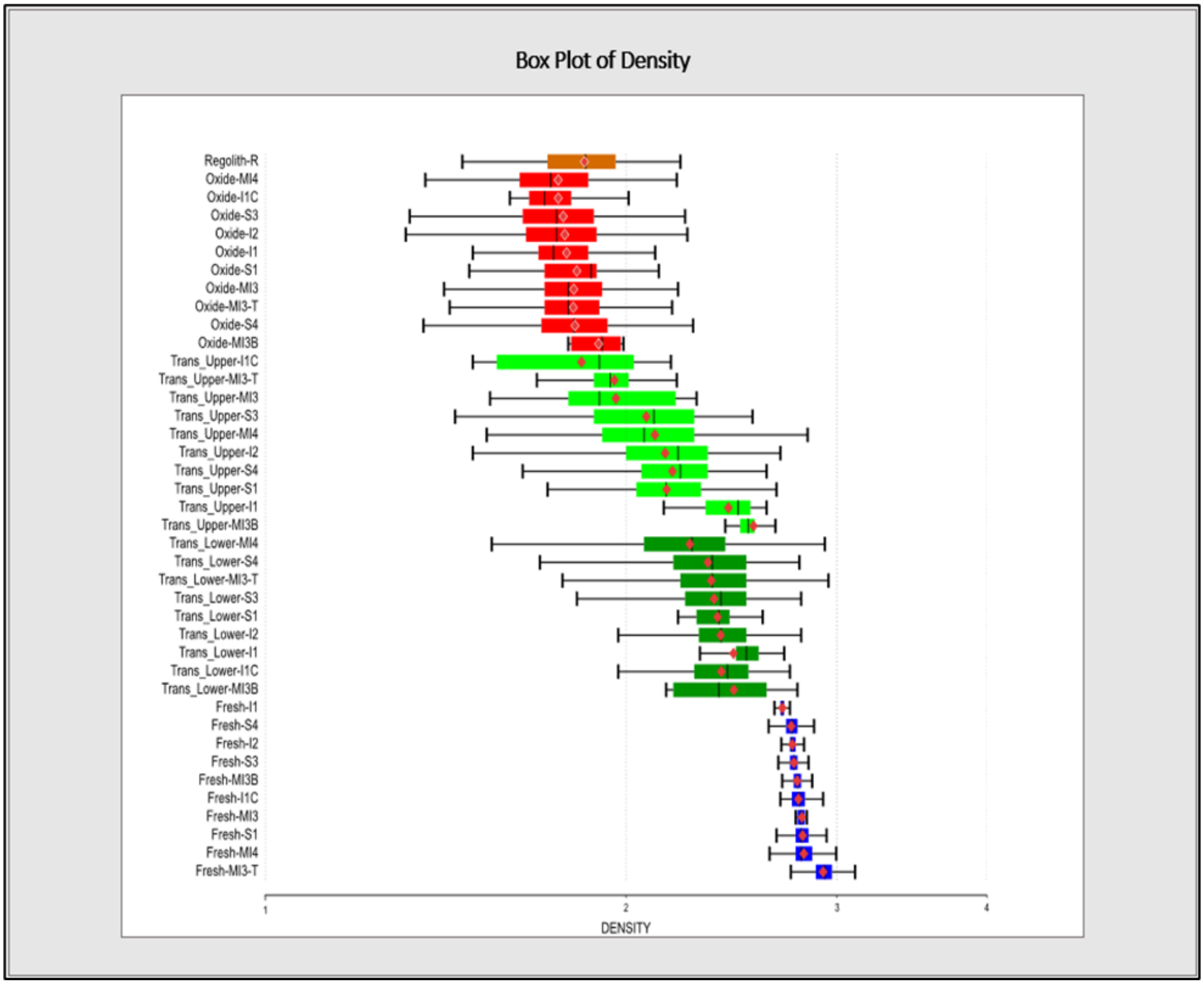
Domain	Rock Code	Count	Average Length m	Minimum Length m	Maximum Length m	Average Au g/t	Minimum Au g/t	Maximum Au g/t	Std Dev g/t	CoV
MB_CFU_01	167	669	1.02	0.70	2.55	0.581	0.016	63.264	2.490	4.28
MB_CFU_02	168	156	1.00	1.00	1.00	0.425	0.018	9.853	0.852	2.00
MB_CFU_03	169	27	1.02	1.00	1.50	0.592	0.142	5.381	1.038	1.72
MB_CFU_04	170	302	1.07	1.00	5.00	1.919	0.022	175.084	12.806	6.66
MB_CFU_05	171	561	1.09	1.00	3.00	0.631	0.009	65.894	2.934	4.65
MB_CFU_06	172	841	1.06	1.00	7.50	0.912	0.005	192.463	7.492	8.22
MB_CFU_07	173	253	1.12	1.00	3.50	0.439	0.021	6.706	0.716	1.63
MB_CFU_08	174	465	1.12	1.00	4.00	0.785	0.003	60.406	3.250	4.18
MB_CFU_09	175	153	1.03	1.00	2.30	0.562	0.045	17.597	1.752	3.11
MB_CFU_10	176	43	1.05	1.00	2.00	0.512	0.046	4.716	0.826	1.59
MB_CFU_11	177	64	1.08	1.00	2.00	0.574	0.031	8.620	1.167	2.02
MB_CFU_12	178	69	1.14	1.00	3.00	0.404	0.023	1.590	0.363	0.89
MB_CFU_13	179	41	1.00	1.00	1.00	0.416	0.046	1.365	0.301	0.71
MB_CFU_14	180	65	1.05	1.00	2.00	0.666	0.013	7.980	1.159	1.73
MB_CFU_15	181	59	1.00	1.00	1.00	0.377	0.012	2.832	0.446	1.17
MB_CFU_16	182	23	1.09	1.00	2.00	0.401	0.032	3.220	0.640	1.56
MB_CFU_17	183	181	1.00	1.00	1.00	0.426	0.034	15.986	1.267	2.96
MB_CFU_18	184	41	1.00	1.00	1.00	0.560	0.066	6.782	1.178	2.08

The supplied database contains 27,177 point bulk density measurements from drill hole core, with values ranging from 1.09 to 3.33 t/m^3 (Table 14.3.3). Bulk density measurements were back-tagged by mineralization domain, lithology and oxidation state models. Bulk density measurements display a differing range of values based on oxidation state (Figure 14.3.3).

Table 14.3.3 B1 Zone Bulk Density Summary Statistics

Unit	Count	Average Bulk Density t/m^3	Minimum Bulk Density t/m^3	Maximum Bulk Density t/m^3	Std Dev	CoV
Regolith	123	1.85	1.46	2.22	0.16	0.026
Oxide	4,811	1.79	1.09	3.02	0.18	0.032
Transitional Upper	1,731	2.13	1.44	2.90	0.24	0.059
Transitional Lower	3,237	2.37	1.47	2.95	0.22	0.048
Fresh	17,275	2.79	1.84	3.33	0.08	0.007
Total	27,177	2.52	1.09	3.33	0.42	0.174

Figure 14.3.3 Boxplot of B1 Zone Bulk Density (t/m³) by Oxide / Lithology State



14.3.3 Block Model

A rotated block model was established using Surpac™ 2022 software with the block model limits selected so as to cover the extent of the mineralized structures, the potential open pit dimensions, and to reflect the general orientation of the mineralized domains (Table 14.3.4). The block model consists of separate variables for estimated grades, rock codes, bulk density and classification attributes. Separate variables were coded with unique mineralization domain, lithology and oxide state rock codes.

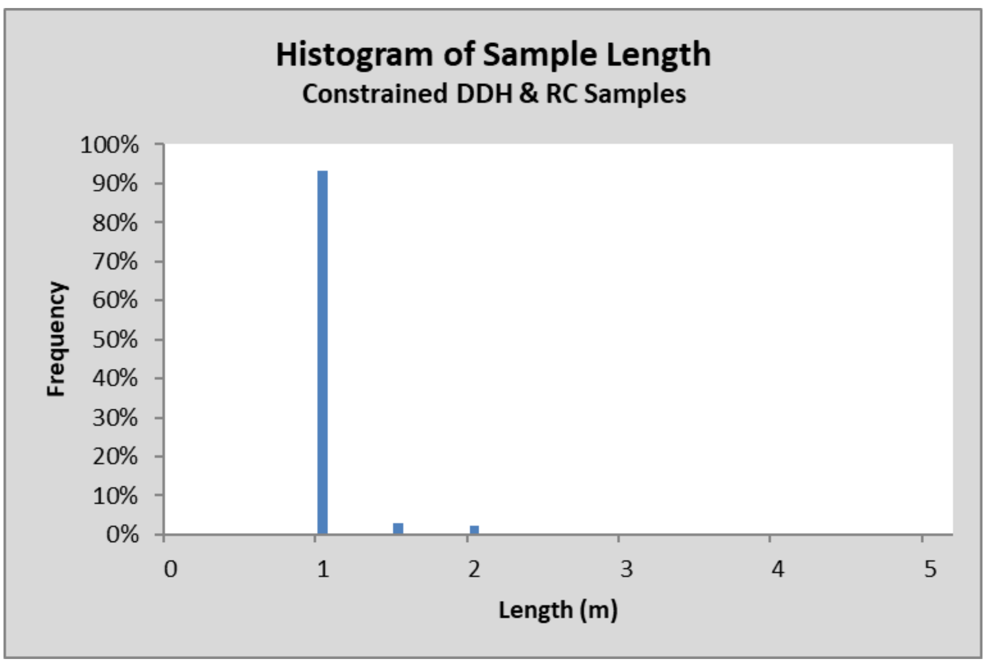
Table 14.3.4 B1 Zone Block Model Set Up

	Minimum	Maximum	Block Size (m)	Number of Blocks
Easting (X)	726,205	730,265	2.00	4,060
Northing (Y)	1,353,675	1,357,675	6.25	4,000
Elevation (max Z)	-50	331	3.00	381
Rotation	42° clockwise			

14.3.4 Compositing

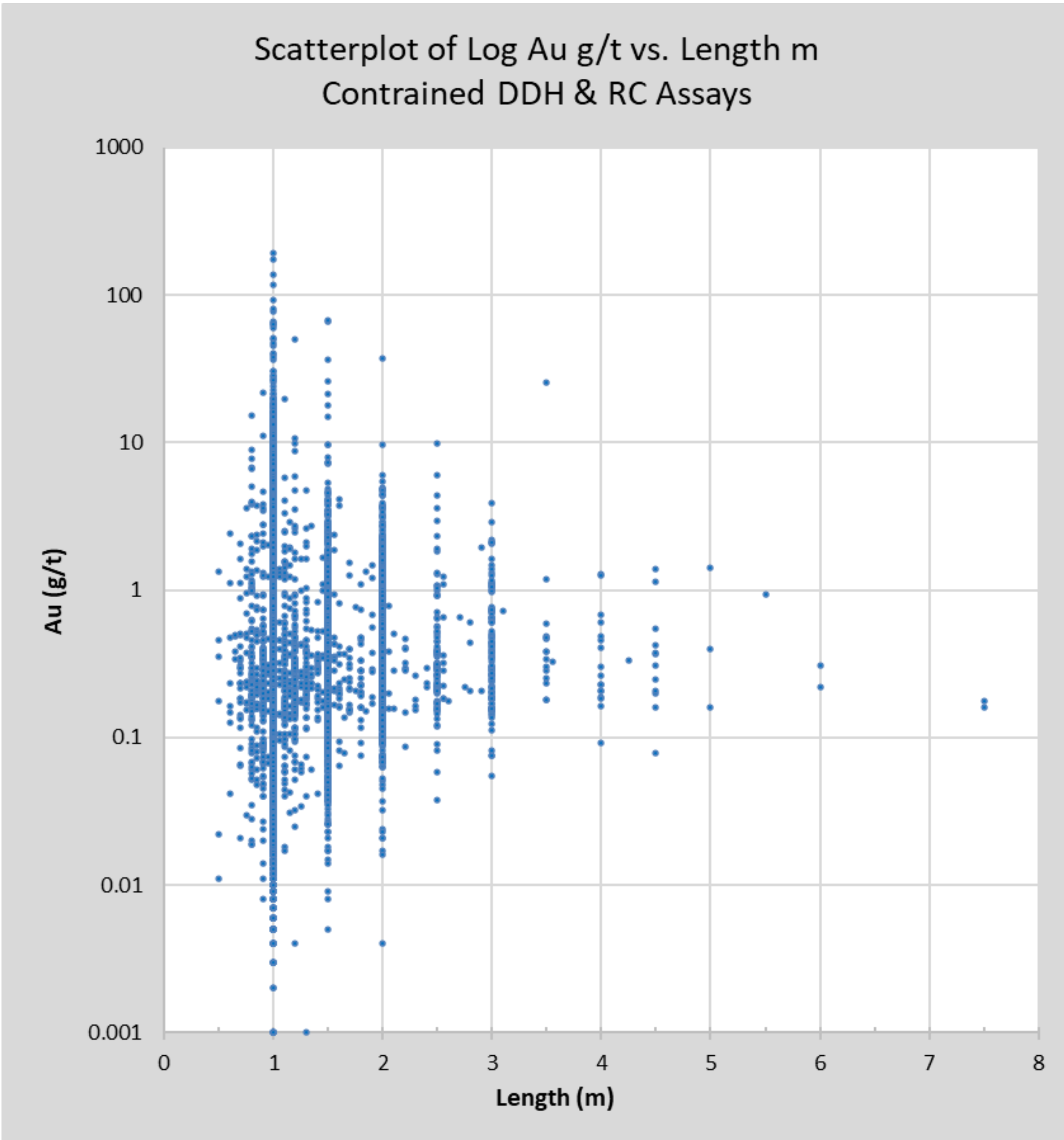
Assay sample lengths for DD and RC drill holes range from 0.50 to 9.00 m, with an average sample length of 1.05 m (Figure 14.3.4). The average sample length for DD assays is 1.11 m, and the average sample length for RC assays is 1.02 m. Constrained assay sample lengths within the defined mineralization domains range from 0.50 to 7.50 m, with an average sample length of 1.05 m. Approximately 93% of the constrained assay samples display a sample length equal to 1.00 m.

Figure 14.3.4 Histogram of Constrained B1 Zone Assay Sample Lengths



No correlation was observed between sample gold grade and sample length for the constrained assay samples (Figure 14.3.5).

Figure 14.3.5 Scatterplot of Assay Sample Lengths Versus Gold Grade



Based on the predominance of 1.00 m sample lengths, Orezone decided that 1.00 m downhole composites are appropriate for all constrained capped assay samples. The downhole compositing process used a 'best fit' approach that results in composites of slightly variable length, but of equal contiguous length within a given drill hole intersection, ensuring the composite length is as close as possible to the nominated 1.00 m composite length. Length-weighted composites were calculated within the defined mineralization domains. Missing sample intervals in the data were ignored and treated as null values during the compositing process. The compositing process started at the first point of intersection between the drill hole and the domain wireframe intersected and halted upon exit from the wireframe. Residual composites less than 1.00 m were retained.

The wireframes that represent the interpreted mineralization domains were used to back-tag a rock code variable into the composite workspace. The composite data were then visually validated against the mineralization wireframes and extracted for analysis and grade estimation. Summary composite statistics are listed in Table 14.3.5.

Table 14.3.5 B1 Zone Composite Summary Statistics

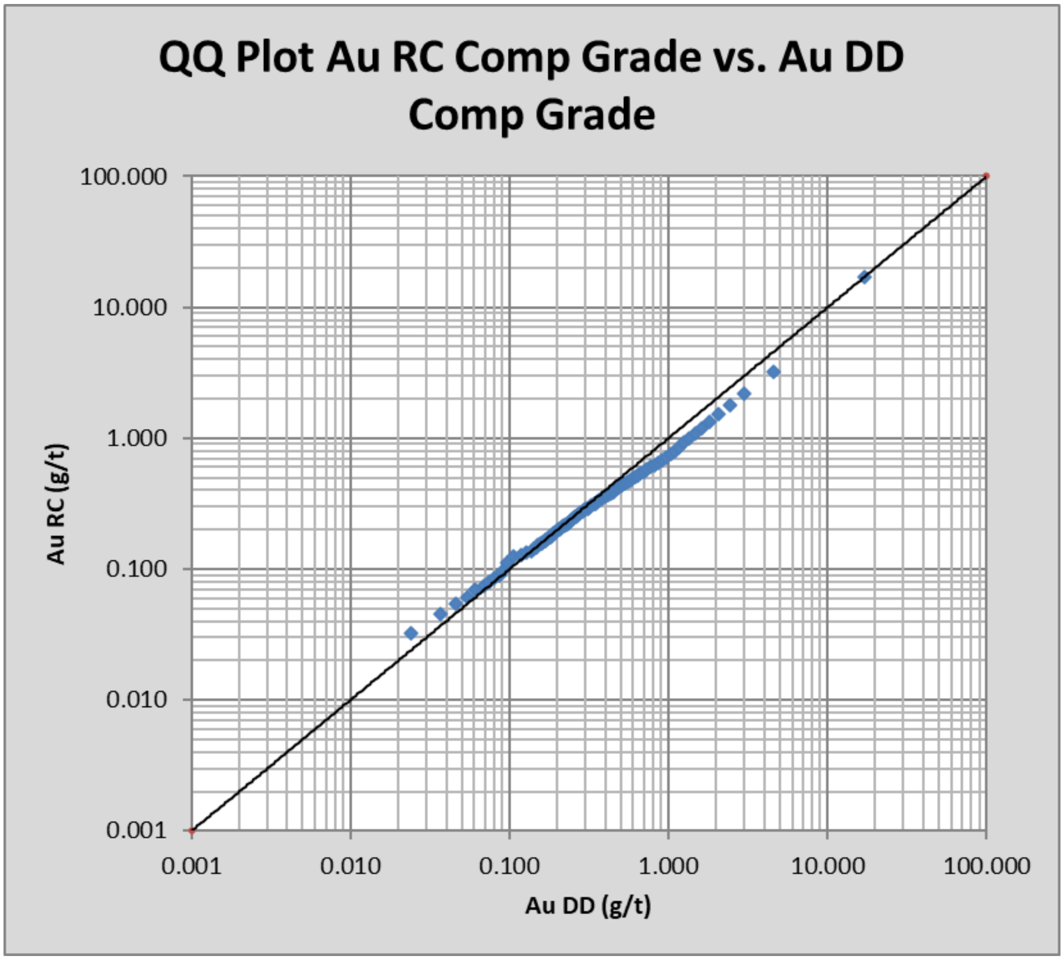
Domain	Count	Avg Au g/t	Min Au g/t	Max Au g/t	Std Dev g/t	CoV
100	89	0.655	0.007	3.000	0.733	2.10
101	7,957	0.826	0.001	17.000	1.143	6.21
102	80	0.414	0.004	3.000	0.482	2.72
103	455	0.336	0.024	2.000	0.329	1.74
104	12	1.576	0.216	3.000	1.123	2.28
105	1,795	0.635	0.003	4.000	0.622	1.77
106	1,660	0.973	0.009	12.000	1.734	0.98
107	504	0.608	0.015	5.000	0.790	1.29
108	42	0.304	0.043	1.000	0.225	6.64
109	2,240	0.888	0.005	15.000	1.759	0.94
110	521	0.629	0.021	6.000	0.997	1.52
111	526	0.990	0.011	10.000	1.616	2.09
112	785	0.853	0.018	10.000	1.429	8.54
113	26,942	0.372	0.003	10.000	0.455	4.46
114	32	1.546	0.035	5.000	1.589	2.27
115	60	0.438	0.078	1.891	0.375	5.79
116	29	0.577	0.153	1.953	0.537	1.32
117	124	0.507	0.022	2.500	0.527	1.12
118	103	0.387	0.005	1.800	0.383	2.10
119	82	0.428	0.009	1.500	0.367	1.53
120	8,947	0.411	0.001	9.000	0.586	1.65
121	649	0.417	0.020	2.500	0.432	1.77
122	268	0.325	0.023	1.500	0.251	1.95
123	607	0.313	0.014	2.400	0.286	1.93
124	208	0.438	0.023	2.500	0.472	3.33
125	626	0.580	0.028	7.000	0.857	2.48
126	239	0.338	0.013	1.500	0.263	1.51
127	437	0.394	0.027	1.900	0.325	1.92
128	269	0.373	0.011	1.500	0.328	1.82

Domain	Count	Avg Au g/t	Min Au g/t	Max Au g/t	Std Dev g/t	CoV
129	229	0.637	0.015	8.000	1.211	2.61
130	130	0.331	0.056	1.256	0.232	1.79
131	117	0.374	0.061	1.500	0.300	1.70
132	10,866	0.282	0.005	7.000	0.381	4.84
133	34	0.579	0.029	1.700	0.500	1.21
134	42	0.684	0.003	1.700	0.567	1.84
135	58	0.392	0.004	1.200	0.315	2.21
136	587	0.527	0.009	7.000	0.755	2.07
137	627	0.482	0.012	5.000	0.614	1.30
138	596	0.592	0.008	5.000	0.681	2.31
139	828	0.425	0.001	4.000	0.455	2.89
140	1,020	0.447	0.006	4.000	0.512	2.82
141	21	0.488	0.017	1.000	0.279	3.94
142	51	0.613	0.030	2.500	0.692	1.81
143	41	0.491	0.028	1.500	0.389	0.96
144	98	0.527	0.068	3.000	0.602	3.33
145	137	0.494	0.009	3.000	0.573	1.04
146	240	0.471	0.006	3.000	0.529	1.07
147	89	0.368	0.016	1.300	0.294	1.32
148	190	0.375	0.026	1.500	0.330	0.93
149	23	0.390	0.154	0.962	0.203	2.99
150	11	0.413	0.043	0.882	0.264	1.40
151	77	0.946	0.002	6.000	1.379	1.36
152	122	0.344	0.036	1.419	0.250	1.55
153	88	0.564	0.016	2.600	0.595	2.03
154	48	0.380	0.027	1.248	0.246	1.03
155	10	0.825	0.077	1.500	0.524	2.26
156	63	0.670	0.013	2.362	0.609	1.47
157	52	0.926	0.009	4.000	1.069	1.52
158	24	0.605	0.035	1.886	0.561	2.49
159	100	0.451	0.061	2.302	0.475	1.55
160	169	0.332	0.006	1.641	0.274	1.56
161	120	0.382	0.042	1.500	0.310	1.03
162	107	0.342	0.017	1.935	0.306	2.10
163	62	0.490	0.091	2.065	0.419	1.26

Domain	Count	Avg Au g/t	Min Au g/t	Max Au g/t	Std Dev g/t	CoV
164	61	0.251	0.037	0.600	0.151	0.92
165	70	0.292	0.071	0.600	0.155	1.01
166	164	0.383	0.029	2.973	0.463	1.57
167	683	0.494	0.016	6.000	0.590	1.37
168	160	0.373	0.017	2.500	0.417	1.45
169	28	0.350	0.142	0.800	0.227	1.95
170	333	0.698	0.017	7.000	1.016	1.15
171	614	0.523	0.011	9.000	0.915	1.76
172	917	0.498	0.005	7.000	0.756	1.21
173	286	0.431	0.021	4.000	0.594	0.65
174	521	0.716	0.003	15.000	1.699	1.13
175	158	0.413	0.047	4.000	0.598	1.07
176	45	0.337	0.046	0.800	0.239	2.07
177	69	0.495	0.031	2.946	0.654	3.10
178	89	0.352	0.018	1.590	0.339	1.42
179	41	0.410	0.046	1.365	0.299	1.64
180	68	0.604	0.013	3.981	0.798	8.03
181	59	0.329	0.012	1.000	0.256	1.67
182	25	0.279	0.032	0.600	0.186	1.20
183	181	0.306	0.034	1.200	0.234	2.07
184	41	0.317	0.066	0.800	0.202	2.34
Total	**77,978**	**0.475**	**0.001**	**17.000**	**0.775**	**3.05**

Examination of the RC and DD composite grade distributions suggests a slight positive bias between the RC and DD composite sample populations at grades <0.15 g/t, and a negative bias between these two populations at grades higher than 0.20 g/t (Figure 14.3.6). These biases are not considered critical, and no correction factors were applied.

Figure 14.3.6 QQ Plot of B1 Zone RC Composite Grades vs. DD Composite Grades



14.3.5 Treatment of Extreme Values

Capping thresholds were determined for each of the mineralized domains prior to compositing of the assay data using histograms with cumulative frequency, normal distribution, log-probability plots and graphical inspection of the spatial grade distribution. Potential outliers are not markedly clustered in localized high-grade areas and sub-domaining is therefore not warranted. Raw assays are capped to the defined threshold prior to compositing within the mineralization domains. A total of seven domains do not require capping (Table 14.3.6). The average capped assay gold grade is 6% lower than the raw assay grade.

Table 14.3.6 B1 Zone Capping Thresholds and Summary Statistics Table

Domain	Count	Cap g/t	Number Capped	Uncapped Mean g/t	Capped Mean g/t	Mean Above Cap g/t	Change %
100	86	3.00	3	0.675	0.638	4.054	-5
101	7,380	17.00	7	0.863	0.840	41.214	-3
102	78	3.00	1	0.539	0.413	12.757	-23
103	431	2.00	4	0.345	0.341	2.503	-1
104	10	3.00	2	2.148	1.677	5.353	-22
105	1,533	4.00	6	0.637	0.630	5.819	-1
106	1,591	12.00	12	1.001	0.980	14.783	-2
107	477	5.00	1	0.624	0.616	9.038	-1
108	39	1.00	1	0.354	0.312	2.630	-12
109	2,068	15.00	9	0.943	0.903	24.101	-4
110	442	6.00	6	0.715	0.624	12.738	-13
111	474	10.00	1	1.037	1.016	20.214	-2
112	717	10.00	5	0.989	0.884	24.958	-11
113	25,805	10.00	11	0.378	0.374	20.149	-1
114	28	5.00	2	2.861	1.393	25.554	-51
115	53	1.89	1	0.444	0.444	1.891	0
116	16	1.95	1	0.822	0.822	1.953	0
117	117	2.50	3	0.567	0.513	4.607	-10
118	86	1.80	4	0.475	0.405	3.301	-15
119	66	2.65	1	0.443	0.443	2.654	0
120	8,444	9.00	10	0.433	0.413	25.966	-5
121	618	2.50	5	0.428	0.421	3.387	-2
122	253	1.50	1	0.328	0.324	2.502	-1
123	578	11.04	1	0.331	0.331	11.042	0
124	203	2.50	2	0.449	0.435	3.880	-3
125	626	4.00	10	0.609	0.550	7.675	-10
126	239	3.46	1	0.350	0.350	3.460	0
127	432	1.90	3	0.406	0.397	3.214	-2
128	269	1.50	8	0.412	0.373	2.799	-9
129	227	2.50	6	0.857	0.495	16.193	-42
130	130	1.26	0	0.331	0.331		0
131	115	1.50	1	0.392	0.371	3.958	-5
132	7,647	27.71	1	0.343	0.343	27.714	0

Domain	Count	Cap g/t	Number Capped	Uncapped Mean g/t	Capped Mean g/t	Mean Above Cap g/t	Change %
133	34	1.70	2	0.640	0.579	2.734	-10
134	42	1.70	4	1.035	0.684	5.379	-34
135	57	1.20	3	0.444	0.402	2.010	-10
136	587	7.00	2	0.670	0.531	47.880	-21
137	618	5.00	4	0.640	0.483	29.316	-25
138	609	5.00	2	0.594	0.587	7.097	-1
139	817	4.00	3	0.438	0.429	6.508	-2
140	1,013	4.00	5	0.594	0.446	33.832	-25
141	21	1.00	3	0.694	0.488	2.439	-30
142	48	2.50	4	0.742	0.634	3.800	-15
143	41	1.50	1	0.508	0.491	2.205	-3
144	96	3.00	1	0.548	0.526	5.158	-4
145	124	3.00	1	0.541	0.535	3.685	-1
146	241	3.00	3	0.544	0.473	8.763	-13
147	87	1.80	1	0.410	0.384	4.061	-6
148	191	1.50	5	0.395	0.373	2.336	-6
149	21	0.96	1	0.412	0.412	0.962	0
150	11	0.88	1	0.413	0.413	0.882	0
151	67	6.00	1	1.256	1.085	17.429	-14
152	122	1.42	0	0.348	0.348		0
153	86	2.60	3	0.635	0.574	4.374	-10
154	48	1.25	0	0.380	0.380		0
155	10	1.50	1	1.168	0.827	4.908	-29
156	58	2.38	1	0.719	0.719	2.380	0
157	52	4.00	2	1.457	0.926	17.797	-36
158	24	1.96	0	0.592	0.592		0
159	100	2.35	1	0.454	0.454	2.350	0
160	168	1.64	1	0.333	0.333	1.641	0
161	121	1.50	2	0.396	0.387	2.033	-2
162	102	1.94	0	0.346	0.346		0
163	56	2.07	0	0.507	0.507		0
164	61	0.60	5	0.278	0.251	0.931	-10
165	114	0.60	6	0.433	0.276	3.585	-36
166	163	3.00	1	0.404	0.383	6.332	-5
167	669	6.00	1	0.581	0.495	63.264	-15

Domain	Count	Cap g/t	Number Capped	Uncapped Mean g/t	Capped Mean g/t	Mean Above Cap g/t	Change %
168	156	2.50	1	0.425	0.378	9.853	-11
169	27	0.80	4	0.592	0.357	2.391	-40
170	302	7.00	4	1.919	0.714	97.926	-63
171	561	9.00	2	0.631	0.521	40.049	-18
172	841	7.00	5	0.912	0.510	74.549	-44
173	253	4.00	2	0.439	0.424	5.892	-3
174	465	12.00	3	0.785	0.651	32.851	-17
175	153	4.00	2	0.562	0.418	14.985	-26
176	43	0.80	5	0.512	0.341	2.269	-33
177	64	3.00	1	0.574	0.486	8.620	-15
178	69	1.59	1	0.404	0.404	1.590	0
179	41	1.37	0	0.416	0.416		0
180	65	4.00	1	0.666	0.605	7.980	-9
181	59	1.00	3	0.377	0.332	1.886	-12
182	23	0.60	1	0.401	0.287	3.220	-28
183	181	1.20	5	0.426	0.307	5.523	-28
184	41	0.80	3	0.560	0.317	4.127	-43
Total	**71,301**		**244**	**0.523**	**0.490**	**14.869**	**-6**

14.3.6 Variography & Continuity Analysis

Orezone performed three-dimensional continuity analyses (variography) within the mineralized sub-domains to determine appropriate estimation inputs to the grade interpolation process.

The variogram modelling process followed by Orezone involves the following steps:

- Calculate and model the downhole variogram to characterize the Nugget Effect.
- Calculate a fan of variograms within the plane of greatest continuity to identify the direction of maximum continuity within the plane.
- Model the variogram in the direction of maximum continuity and the orthogonal directions.

Variogram parameters were derived for the better sampled mineralization sub-domains that is, MB_B1_01, MB_B1_06, MB_B1_39, and MB_B1_40. Variogram modelling for the sparsely sampled sub-domains was inconclusive, and the variography used to estimate these sub-domains has been adopted from adjacent mineralized domains, but the directions and anisotropy ratios have been tailored to best suit the geometry of each of the sub-domains.

The better variogram ranges from the above-mentioned sampled mineralized domains were respectively 56, 45, 51 and 50 m, indicating that the maximum spatial continuity is greater than the average drill hole spacing.

14.3.7 Estimation & Classification

The bulk density data were extracted from the drill hole database and grouped according to their host rock lithology within the Fresh weathering unit, or within their host weathering unit above the Fresh weathering unit. Separate capping values were determined for each lithology type and oxidation state. The bulk density values for each block were then estimated using the capped bulk density values and anisotropic Inverse Distance Squared (ID2) linear interpolation using a minimum of three and maximum of 15 composites within a 120 m diameter search envelope. Sample selection was restricted to a maximum of three composite samples from a single drill hole. Grade estimation was performed by weathering unit within the weathered portion of the deposits, and by lithological domain within the fresh Birimian portion of the deposits. These weathering and lithological units define hard boundaries for bulk density estimation. For those areas of limited sample data, the mean value of the specific lithology type in the Fresh domain or within the weathering unit above the Fresh unit was applied. Orezone applies a reduction factor of 5.5% and 2.5% to individual samples in the oxide and transition zones, respectively, for the block model dry bulk density calculations. Bulk density block values were adjusted by these factors after estimation.

Block grades for gold were estimated by Inverse Distance Cubed (ID3) weighting of capped composites in three separate passes, using a minimum of five and maximum of 15 composites for Pass 1, and a minimum of three and maximum of 15 composites for Pass 2 and Pass 3. Sample selection was restricted to a maximum of two composite samples from a single drill hole.

The orientation of the search ellipsoids was defined by Orezone geologists based on the local geology within each respective mineralization domain. The defined orientations were then converted to Surpac™ format rotations. Capped Ordinary Kriging (OK) and capped Nearest Neighbour models (NN) were also generated using the same search parameters. Only composites derived from DD and RC drill holes were used for grade estimation. Search and grade estimation were constrained by the individual mineralization domains, which define hard boundaries for grade estimation.

The parameters used to define classification limits included spatial analysis, drill hole spacing, distance from nearest grade composite, and the observed continuity of the mineralization. Mineral Resources were classified algorithmically as follows:

Measured Mineral Resources are defined by:

- A drill spacing of less than 25 m x 25 m.
- Average True Distance to Samples: 0 to 25 m.
- Conditional bias slope > 0.75.
- A minimum of five composite samples.
- A maximum of two composite samples per drill hole.
- Three or more drill holes.
- Estimation Pass 1.

Indicated Mineral Resources are defined by:

- A drill spacing of less than 50 m x 50 m.
- Average True Distance to Sample: 25 to 50 m.
- A minimum of three composite samples.
- A maximum of two composite samples per drill hole.
- Two or more drill holes.
- Estimation Pass 1 or 2.

Inferred Mineral Resources include all estimated mineralization defined by:

- A drill spacing greater than 50 m x 50 m.
- Average True Distance to sample: 50 to 100 m.
- Two or more drill holes.
- A minimum of three composite samples.
- A maximum of two composite samples per drill hole.
- Estimation Pass 1, 2 or 3.

All other mineralization remained unclassified.

14.3.8 Mineral Resource Estimate

Mineral Resources reported herein have been constrained within an optimized pit shell (Figure 14.3.7). The results from the optimized pit shell are used solely for the purpose of reporting Mineral Resources and include Measured, Indicated and Inferred Mineral Resources. Orezone reports whole block volumes for only those blocks where the block centroid lies within the controlling wireframe. Orezone applied a 0.95 factor to the B1 Zone oxide grade to account for artisanal mining.

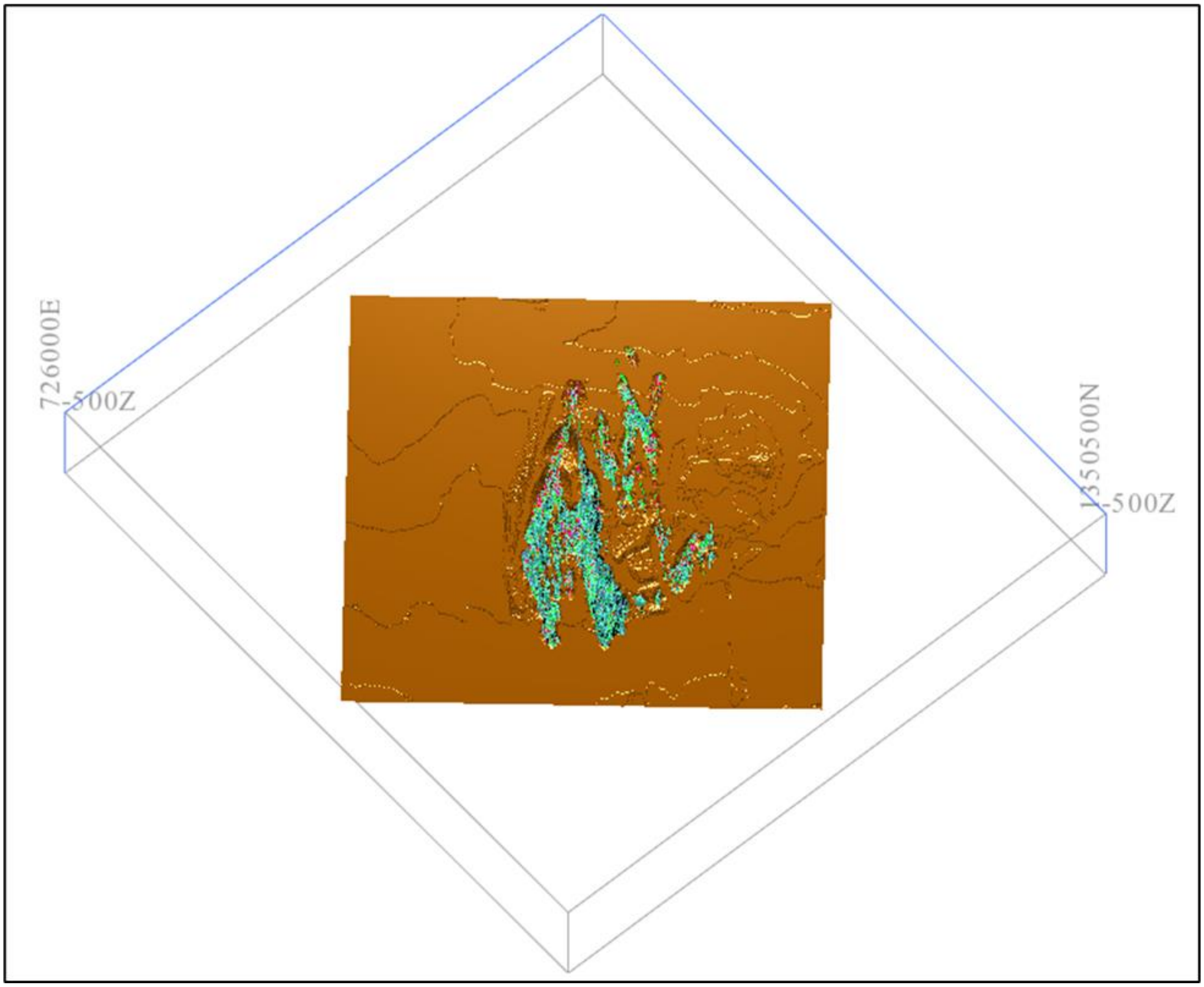
Mineral Resources are reported based on the cut-offs listed in Table 14.3.7 and a gold price of USD 1,700 per ounce.

Table 14.3.7 B1 Zone MRE Reporting Cut-Off Grades

Unit	Au g/t
Regolith	0.25
Oxide	0.25
Trans Upper	0.25
Trans Lower	0.45
Fresh	0.45

The Mineral Resources have an effective date of March 28, 2023 (Table 14.3.8).

Figure 14.3.7 Isometric View of B1 Zone Optimized Pit Shell



View looking northeast

Table 14.3.8 B1 Zone Mineral Resources Estimate*

Total	Cut-off	Tonnes k	Au g/t	Au koz
Measured	NA	1,512	0.44	21
Indicated	NA	30,281	0.74	716
Meas + Ind	NA	31,793	0.72	737
Inferred	NA	3,068	1.02	101

Regolith	Cut-off	Tonnes k	Au g/t	Au koz
Measured	0.25	0	0.00	0
Indicated	0.25	0	0.00	0
Meas + Ind	0.25	0	0.00	0
Inferred	0.25	0	0.00	0
Oxide	**Cut-off**	**Tonnes k**	**Au g/t**	**Au koz**
Measured	0.25	704	0.43	10
Indicated	0.25	13,599	0.51	223
Meas + Ind	0.25	14,303	0.51	233
Inferred	0.25	910	0.60	18
Trans Upper	**Cut-off**	**Tonnes (k)**	**Au (g/t)**	**Au (koz)**
Measured	0.25	793	0.45	11
Indicated	0.25	5,728	0.60	110
Meas + Ind	0.25	6,521	0.58	121
Inferred	0.25	298	0.64	6
Trans Lower	**Cut-off**	**Tonnes (k)**	**Au (g/t)**	**Au (koz)**
Measured	0.45	15	0.64	0
Indicated	0.45	2,964	0.97	93
Meas + Ind	0.45	2,979	0.97	93
Inferred	0.45	163	0.84	4
Fresh	**Cut-off**	**Tonnes (k)**	**Au (g/t)**	**Au (koz)**
Measured	0.45	0	0.00	0
Indicated	0.45	7,990	1.13	290
Meas + Ind	0.45	7,990	1.13	290
Inferred	0.45	1,697	1.33	73

* Mineral Resources are inclusive of Mineral Reserves. Totals may differ due to rounding.

14.3.9 Validation

The block model was validated visually by the inspection of successive vertical cross-section lines and plan views in order to confirm that the block models correctly reflect the distribution of high-grade and low-grade values.

The mean block grade estimated for each sub-domain was then compared to the mean composite grade of the corresponding sub-domain (Table 14.3.9). Although these two parameters are not strictly comparable due to data clustering and volume influences, they do provide a useful validation tool for detecting any major biases and allow the comparison between input composite grade and the estimated block grade.

Table 14.3.9 Comparison of Au Composite Grade and OK, ID3 and NN Estimated Block Grades

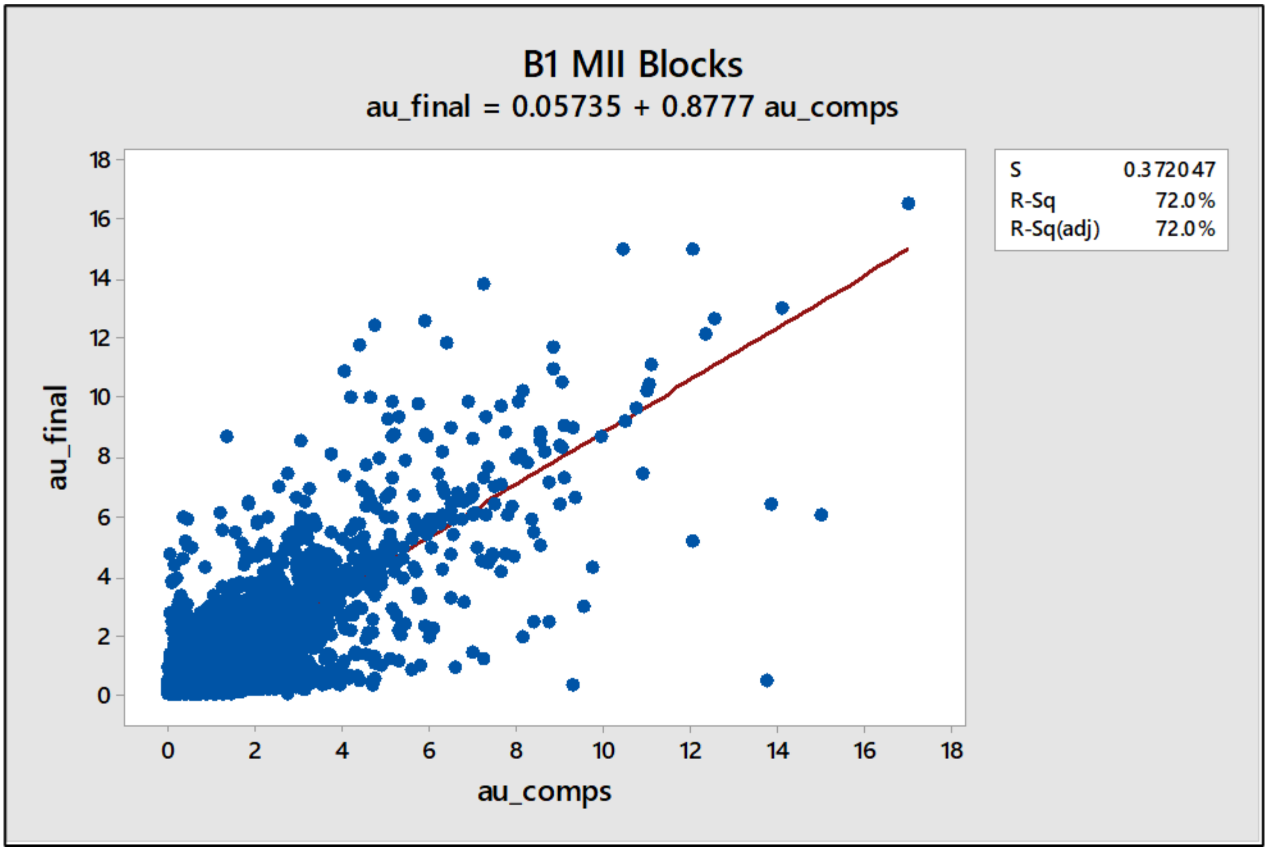
Domain	Tonnes	Number of Samples	Mean Composite Au Grade g/t	OK Estimated Mean Au Grade g/t	IDW Estimated Mean Au Grade g/t	NN Estimated Mean Au Grade g/t	OK Relative Difference %	IDW Relative Difference %	NN Relative Difference %
100	364,990	89	0.65	0.67	0.77	0.69	3	18	6
101	25,832,742	7,957	0.83	0.83	0.82	0.84	0	-1	1
102	326,929	80	0.41	0.77	0.68	1.10	88	66	168
103	1,688,462	455	0.34	0.40	0.41	0.41	18	21	21
104	16,425	12	1.58	1.92	1.87	1.89	22	18	20
105	4,385,921	1,795	0.63	0.60	0.60	0.60	-5	-5	-5
106	5,409,135	1,660	0.97	0.71	0.71	0.68	-27	-27	-30
107	1,007,160	504	0.61	0.54	0.55	0.55	-11	-10	-10
108	44,234	42	0.30	0.35	0.33	0.36	17	10	20
109	3,495,043	2,240	0.89	0.86	0.86	0.82	-3	-3	-8
110	462,354	521	0.63	0.63	0.62	0.66	0	-2	5
111	972,608	526	0.99	1.09	1.03	1.05	10	4	6
112	2,774,318	785	0.85	0.90	0.87	0.85	6	2	0
113	94,756,576	26,942	0.37	0.34	0.34	0.33	-8	-8	-11
114	59,485	32	1.55	1.78	1.88	2.07	15	21	34
115	112,772	60	0.44	0.42	0.43	0.40	-5	-2	-9
116	25,407	29	0.58	0.64	0.73	0.58	10	26	0
117	321,000	124	0.51	0.53	0.64	0.39	4	25	-24
118	88,936	103	0.39	0.37	0.39	0.35	-5	0	-10
119	216,438	82	0.45	0.43	0.45	0.45	-4	0	0
120	78,332,511	8,947	0.41	0.42	0.42	0.43	2	2	5
121	2,502,647	649	0.42	0.43	0.44	0.40	2	5	-5
122	675,906	268	0.32	0.33	0.33	0.32	3	3	0
123	6,449,602	607	0.31	0.35	0.36	0.33	13	16	6
124	1,429,010	208	0.44	0.43	0.43	0.44	-2	-2	0

Domain	Tonnes	Number of Samples	Mean Composite Au Grade g/t	OK Estimated Mean Au Grade g/t	IDW Estimated Mean Au Grade g/t	NN Estimated Mean Au Grade g/t	OK Relative Difference %	IDW Relative Difference %	NN Relative Difference %
125	1,822,859	626	0.55	0.49	0.52	0.43	-11	-5	-22
126	1,971,143	239	0.34	0.32	0.33	0.31	-6	-3	-9
127	1,393,988	437	0.39	0.37	0.36	0.38	-5	-8	-3
128	664,872	269	0.37	0.34	0.34	0.35	-8	-8	-5
129	1,053,348	229	0.50	0.45	0.46	0.43	-10	-8	-14
130	342,801	130	0.33	0.32	0.32	0.34	-3	-3	3
131	1,076,561	117	0.37	0.48	0.49	0.50	30	32	35
132	51,583,817	10,866	0.29	0.24	0.25	0.23	-17	-14	-21
133	25,221	34	0.58	0.54	0.56	0.55	-7	-3	-5
134	231,654	42	0.68	0.68	0.68	0.65	0	0	-4
135	594,454	58	0.39	0.27	0.36	0.16	-31	-8	-59
136	2,943,207	587	0.53	0.47	0.48	0.46	-11	-9	-13
137	4,008,784	627	0.48	0.56	0.57	0.51	17	19	6
138	874,126	596	0.59	0.59	0.58	0.58	0	-2	-2
139	3,479,173	828	0.42	0.44	0.44	0.42	5	5	0
140	7,866,074	1,020	0.45	0.45	0.45	0.44	0	0	-2
141	54,480	21	0.49	0.57	0.58	0.56	16	18	14
142	436,523	51	0.61	0.50	0.51	0.48	-18	-16	-21
143	137,892	41	0.49	0.41	0.44	0.39	-16	-10	-20
144	157,575	98	0.53	0.48	0.37	0.49	-9	-30	-8
145	585,482	137	0.49	0.57	0.55	0.58	16	12	18
146	507,384	240	0.47	0.43	0.45	0.44	-9	-4	-6
147	11,920	89	0.38	0.58	0.58	0.41	53	53	8
148	147,618	190	0.37	0.39	0.39	0.39	5	5	5
149	151,713	23	0.39	0.48	0.49	0.44	23	26	13
150	90,843	11	0.41	0.45	0.45	0.47	10	10	15
151	112,419	77	0.95	1.04	1.02	1.11	9	7	17
152	570,467	122	0.34	0.34	0.34	0.36	0	0	6
153	238,511	88	0.56	0.50	0.52	0.46	-11	-7	-18
154	76,271	48	0.38	0.31	0.32	0.31	-18	-16	-18
155	21,480	10	0.82	0.55	0.71	0.43	-33	-13	-48
156	81,084	63	0.67	0.68	0.65	0.67	1	-3	0
157	56,906	52	0.93	1.12	1.02	1.18	20	10	27

Domain	Tonnes	Number of Samples	Mean Composite Au Grade g/t	OK Estimated Mean Au Grade g/t	IDW Estimated Mean Au Grade g/t	NN Estimated Mean Au Grade g/t	OK Relative Difference %	IDW Relative Difference %	NN Relative Difference %
158	14,857	24	0.60	0.56	0.56	0.54	-7	-7	-10
159	786,376	100	0.45	0.50	0.52	0.49	11	16	9
160	674,919	169	0.33	0.38	0.37	0.35	15	12	6
161	459,996	120	0.38	0.34	0.36	0.33	-11	-5	-13
162	240,610	107	0.34	0.33	0.33	0.32	-3	-3	-6
163	141,072	62	0.49	0.45	0.49	0.43	-8	0	-12
164	205,580	61	0.25	0.24	0.24	0.26	-4	-4	4
165	2,059,100	70	0.29	0.32	0.31	0.34	10	7	17
166	3,077,005	164	0.38	0.38	0.38	0.40	0	0	5
167	3,038,800	683	0.49	0.48	0.48	0.51	-2	-2	4
168	1,527,355	160	0.37	0.35	0.36	0.31	-5	-3	-16
169	155,239	28	0.35	0.46	0.43	0.38	31	23	9
170	874,828	333	0.70	0.56	0.58	0.52	-20	-17	-26
171	1,331,167	614	0.52	0.45	0.48	0.44	-13	-8	-15
172	1,501,293	917	0.50	0.45	0.47	0.45	-10	-6	-10
173	1,172,048	286	0.43	0.39	0.41	0.37	-9	-5	-14
174	2,345,656	521	0.69	0.47	0.50	0.49	-32	-28	-29
175	464,377	158	0.41	0.37	0.38	0.37	-10	-7	-10
176	116,735	45	0.34	0.34	0.36	0.37	0	6	9
177	71,591	69	0.50	0.53	0.58	0.56	6	16	12
178	262,994	89	0.35	0.30	0.32	0.35	-14	-9	0
179	221,720	41	0.41	0.40	0.43	0.41	-2	5	0
180	125,603	68	0.60	0.52	0.52	0.54	-13	-13	-10
181	119,047	59	0.33	0.36	0.37	0.35	9	12	6
182	206,562	25	0.28	0.33	0.32	0.38	18	14	36
183	1,339,948	181	0.31	0.32	0.31	0.35	3	0	13
184	149,273	41	0.32	0.34	0.36	0.31	6	13	-3
Total	**337,805,015**	**77,978**	**0.441**	**0.424**	**0.425**	**0.419**	**-4**	**-4**	**-5**

An additional validation check was completed by comparing the average grade of the composites in a block to the associated model block grade estimate (Figure 14.3.8).

Figure 14.3.8 B1 Zone Validation Plot Between Block Grades and Average Composite Grades



The volume estimated was also checked against the reported volume of the individual mineralization domains. Estimated volumes are based on a 0.001 g/t gold cut-off and whole block volumes (Table 14.3.10). The results fall within acceptable limits for linear estimation.

Table 14.3.10 B1 Zone Volume Comparison

Mineralization Domain	Rock Code	Wireframe Volume k m³	Estimated Whole Block Volume k m³	Solid Volume versus Block Volume %
MB_B1_00	100	162	162	0
MB_B1_01	101	11,416	11,422	0
MB_B1_02	102	130	130	0
MB_B1_03	103	700	699	0
MB_B1_04	104	6	6	4
MB_B1_05	105	2,001	2004	0
MB_B1_06	106	2,159	2,155	0

Mineralization Domain	Rock Code	Wireframe Volume k m^3	Estimated Whole Block Volume k m^3	Solid Volume versus Block Volume %
MB_B1_07	107	364	364	0
MB_B1_08	108	16	16	-2
MB_B1_09	109	1,416	1414	0
MB_B1_10	110	209	209	0
MB_B1_11	111	392	392	0
MB_B1_12	112	1,050	1053	0
MB_B1_13	113	56,461	56,600	0
MB_B1_14	114	32	32	1
MB_B1_15	115	49	49	0
MB_B1_16	116	15	14	-1
MB_B1_17	117	126	125	0
MB_B1_18	118	43	43	0
MB_B1_19	119	95	95	0
MB_B1_20	120	53,554	53,581	0
MB_B1_21	121	1,021	1,024	0
MB_B1_22	122	280	278	-1
MB_B1_23	123	3,134	3,132	0
MB_B1_24	124	579	579	0
MB_B1_25	125	1,033	1,034	0
MB_B1_26	126	908	908	0
MB_B1_27	127	949	950	0
MB_B1_28	128	300	299	0
MB_B1_29	129	626	625	0
MB_B1_30	130	150	150	0
MB_B1_31	131	622	623	0
MB_B1_32	132	35,525	35,533	0
MB_B1_33	133	14	14	0
MB_B1_34	134	124	124	0
MB_B1_35	135	268	268	0
MB_B1_36	136	1,484	1,485	0
MB_B1_37	137	2,091	2,092	0
MB_B1_38	138	403	404	0
MB_B1_39	139	1,897	1,898	0
MB_B1_40	140	3,713	3,709	0

Mineralization Domain	Rock Code	Wireframe Volume k m^3	Estimated Whole Block Volume k m^3	Solid Volume versus Block Volume %
MB_B1_41	141	36	36	0
MB_B1_42	142	178	179	0
MB_B1_43	143	63	63	0
MB_B1_44	144	98	97	0
MB_B1_45	145	247	249	1
MB_B1_46	146	234	236	1
MB_B1_47	147	1119	1,121	0
MB_B1_48	148	83	82	-1
MB_B1_49	149	67	67	0
MB_B1_50	150	35	35	1
MB_B1_51	151	54	54	0
MB_B1_52	152	276	275	0
MB_B1_53	153	104	105	0
MB_B1_54	154	43	43	0
MB_B1_55	155	13	13	-2
MB_B1_56	156	40	39	-1
MB_B1_57	157	25	27	7
MB_B1_58	158	8	8	4
MB_B1_59	159	333	333	0
MB_B1_60	160	319	318	0
MB_B1_61	161	219	221	1
MB_B1_62	162	131	130	0
MB_B1_63	163	60	59	-1%
MB_B1_64	164	93	93	0
MB_B1_65	165	1,380	1,379	0
MB_B1_66	166	2,034	2,035	0
MB_CFU_01	167	1374	1,372	0
MB_CFU_02	168	695	693	0
MB_CFU_03	169	65	65	-1
MB_CFU_04	170	360	359	0
MB_CFU_05	171	591	592	0
MB_CFU_06	172	684	685	0
MB_CFU_07	173	488	488	0
MB_CFU_08	174	940	939	0

Mineralization Domain	Rock Code	Wireframe Volume k m^3	Estimated Whole Block Volume k m^3	Solid Volume versus Block Volume %
MB_CFU_09	175	204	203	0
MB_CFU_10	176	52	54	2
MB_CFU_11	177	39	39	0
MB_CFU_12	178	111	112	1
MB_CFU_13	179	90	89	0
MB_CFU_14	180	56	56	0
MB_CFU_15	181	58	57	0
MB_CFU_16	182	97	97	0
MB_CFU_17	183	569	568	0
MB_CFU_18	184	64	65	0
TOTAL		**199,348**	**199,528**	**0**

14.3.10 B1 Cut-Off Sensitivity

The sensitivity of the Mineral Resource block grades to changes in cut-off grade was evaluated by calculating the total Mineral Resource inventory at various cut-offs within the Mineral Resource pit (Table 14.3.11). The values presented are intended for the sole purpose of demonstrating the sensitivity of the Mineral Resource with respect to the cut-off grade.

Table 14.3.11 B1 Grade Tonnage Sensitivity Table

OXIDE MEASURED + INDICATED				OXIDE INFERRED		
Au Cut-Off	Tonnes	Au		Au Cut-Off	Tonnes	Au
g/t	k	g/t		g/t	k	g/t
1.00	1,786.3	1.60		1.00	147.4	1.55
0.95	2,004.5	1.53		0.95	165.3	1.48
0.90	2,253.8	1.46		0.90	188.1	1.41
0.85	2,542.9	1.39		0.85	215.6	1.34
0.80	2,895.0	1.31		0.80	248.2	1.27
0.75	3,304.7	1.24		0.75	279.5	1.21
0.70	3,787.3	1.18		0.70	318.8	1.15
0.65	4,375.4	1.10		0.65	370.0	1.08
0.60	5,098.7	1.03		0.60	423.3	1.02
0.55	6,009.3	0.96		0.55	489.0	0.96
0.50	7,189.0	0.89		0.50	560.5	0.90
0.45	8,767.9	0.81		0.45	654.3	0.84

OXIDE MEASURED + INDICATED				OXIDE INFERRED		
Au Cut-Off	Tonnes	Au		Au Cut-Off	Tonnes	Au
g/t	k	g/t		g/t	k	g/t
0.40	10,782.8	0.73		0.40	769.5	0.77
0.35	13,442.8	0.66		0.35	908.8	0.71
0.30	16,884.5	0.59		0.30	1,064.5	0.65
0.25	20,823.6	0.53		0.25	1,208.0	0.61
0.20	24,614.9	0.48		0.20	1,339.9	0.57

SULPHIDE MEASURED + INDICATED				SULPHIDE INFERRED		
Au Cut-Off	Tonnes	Au		Au Cut-Off	Tonnes	Au
g/t	k	g/t		g/t	k	g/t
1.00	4,027.4	1.81		1.00	756.9	2.20
0.95	4,362.5	1.75		0.95	809.1	2.13
0.90	4,734.2	1.68		0.90	860.8	2.05
0.85	5,140.2	1.62		0.85	924.9	1.97
0.80	5,610.7	1.55		0.80	999.4	1.89
0.75	6,117.7	1.49		0.75	1,079.8	1.80
0.70	6,691.5	1.42		0.70	1,172.0	1.72
0.65	7,328.4	1.36		0.65	1,277.6	1.63
0.60	8,063.9	1.29		0.60	1,399.3	1.54
0.55	8,888.2	1.22		0.55	1,539.0	1.46
0.50	9,849.6	1.16		0.50	1,683.3	1.38
0.45	10,969.1	1.09		0.45	1,859.9	1.29
0.40	12,249.9	1.02		0.40	2,073.1	1.20
0.35	13,722.7	0.95		0.35	2,340.3	1.11
0.30	15,381.7	0.88		0.30	2,652.7	1.02
0.25	17,151.0	0.82		0.25	2,968.5	0.94
0.20	18,715.2	0.77		0.20	3,223.0	0.88

Oxide includes Regolith, Oxide and Transitional Upper units.
Sulphide includes Transitional Lower and Fresh units.
Totals may differ due to rounding.

14.4 Bomboré B2 Zone

14.4.1 Data Supplied

The B2 Zone model was developed by Orezone and reviewed by the Authors. Topography, mineralization, lithology, and oxidation state three-dimensional wireframes were created by Orezone using Leapfrog Geo™ 2022.1.1. Drill hole distances are reported in metres and gold grades are reported as g/t. The drill hole collar coordinates were provided in the WGS84 Zone 30N UTM coordinate reference system. The supplied drill hole database for the B2 Zone contains 4,421 unique drill hole collar records (Table 14.4.1 and Figure 14.4.1).

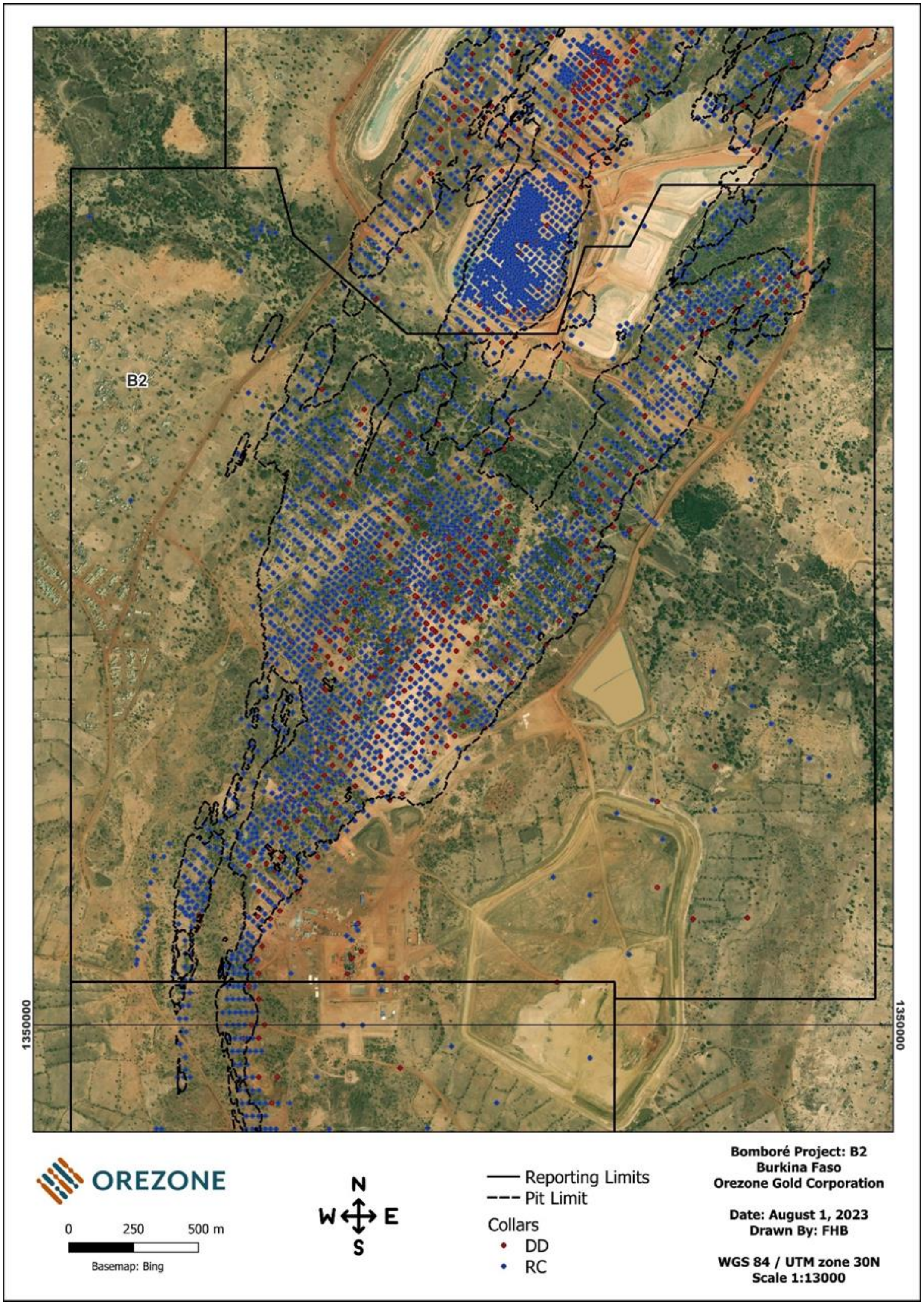
Table 14.4.1 B2 Zone Drill Hole Database Summary

Type	Count	Metres
Pressure Meter Drill Hole	11	243.5
Channel	1	2.0
Diamond Core Drill Hole (DD)	395	66,553.6
Water Drill Hole	13	898.6
Auger Drill Hole	528	2,637.5
Rotary Air Blast drill Hole (RAB)	322	11,421.0
Reverse Circulation Drill Hole (RC)	2,768	167,899.5
Trench	370	3,652.1
Total	4,421	253,314.3

Industry standard validation checks were carried out on the supplied databases, and minor corrections made where necessary. The Authors typically validate a Mineral Resource database by checking for inconsistencies in naming conventions or analytical units, duplicate entries, interval, length or distance values less than or equal to zero, blank or zero-value assay results, out-of-sequence intervals, intervals or distances greater than the reported drill hole length, inappropriate collar locations, and missing interval and coordinate fields.

No significant errors were noted with the supplied databases. The Authors consider that the drill hole database supplied is suitable for Mineral Resource estimation. The drill hole data were imported into a Surpac™ format MS-Access™ database, and imported wireframes were assigned a unique rock code. The supplied wireframes were used to back-tag the assay, bulk density and composite tables with mineralization domain, oxide state and lithology codes.

Figure 14.4.1 B2 Zone Drill Hold Plan View

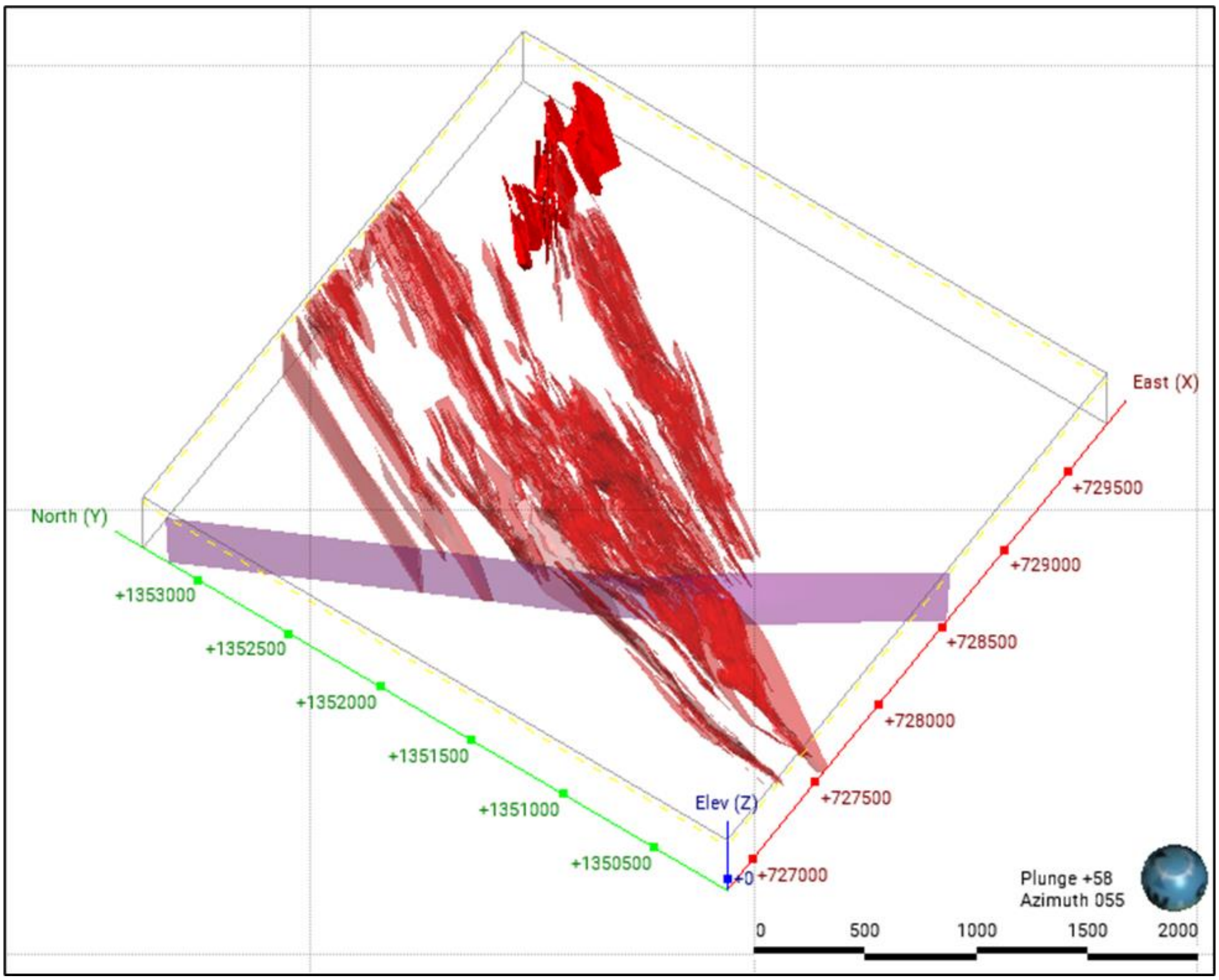


14.4.2 Exploratory Data Analysis

A total of 82 distinct mineralization domain wireframes were supplied (Figure 14.4.2). Each domain was assigned a unique three-digit rock code. Of the 82 domains, seven domains are responsible for 69% of the Mineral Resource by volume: MB_B1_20A (Rock Code 105), MB_B1_13A (Rock Code 104), MB_B1_32A (Rock Code 109), MB_B2_08 (Rock Code 125), MB_B2_10 (Rock Code 127), MB_B2_12 (Rock Code 129) and MB_P11_16 (Rock Code 180).

The average nearest-neighbour collar distance (for DD and RC) is 24.0 m, the average DD length is 168.5 m and the average RC drill hole length is 60.7 m. Summary statistics for the supplied assay data are provided below (Table 14.4.2).

Figure 14.4.2 Isometric View of B2 Zone Mineralization Domains



View looking along azimuth 40 degrees

Table 14.4.2 B2 Zone Assay Summary Statistics (DD and RC)

Domain	Code	Count	Average Length m	Minimum Length m	Maximum Length m	Average Au g/t	Minimum Au g/t	Maximum Au g/t	Std Dev g/t	CoV
Unconstrained		26,392	1.04	0.30	9.00	0.291	0.001	192.463	1.142	
MB_B1_03A	101	18	1.00	1.00	1.00	0.693	0.152	2.372	0.624	0.90
MB_B1_06A	102	51	1.01	1.00	1.50	0.652	0.071	3.335	0.728	1.12
MB_B1_12A	103	61	1.00	1.00	1.00	0.433	0.122	2.329	0.360	0.83
MB_B1_13A	104	19,516	1.02	0.65	6.00	0.414	0.005	46.897	0.613	1.48
MB_B1_20A	105	2,666	1.01	0.70	3.00	0.434	0.016	11.553	0.565	1.30
MB_B1_21A	106	40	1.00	1.00	1.00	0.512	0.078	2.091	0.422	0.82
MB_B1_23A	107	273	1.01	1.00	2.00	0.354	0.025	2.060	0.307	0.87
MB_B1_24A	108	57	1.01	1.00	1.50	0.366	0.079	1.338	0.285	0.78
MB_B1_32A	109	8,484	1.02	0.50	4.50	0.401	0.007	19.000	0.588	1.46
MB_B1_36	110	356	1.00	1.00	1.00	0.657	0.006	77.395	4.215	4.96
MB_B1_59	111	100	1.00	1.00	1.00	0.454	0.061	2.350	0.484	1.07
MB_B1_60	112	168	1.00	1.00	1.00	0.333	0.006	1.641	0.275	0.83
MB_B1_61A	113	7	1.00	1.00	1.00	0.553	0.228	1.209	0.340	0.61
MB_B2_00	114	26	1.00	1.00	1.00	0.418	0.093	1.369	0.359	0.86
MB_B2_00A	115	172	1.01	1.00	1.50	0.489	0.024	7.005	0.739	1.51
MB_B2_00B	116	130	1.03	0.50	2.00	0.929	0.071	55.762	4.872	5.24
MB_B2_00C	117	96	1.00	1.00	1.00	0.665	0.020	7.594	1.049	1.58
MB_B2_01	118	44	1.00	1.00	1.00	0.417	0.088	1.715	0.320	0.77
MB_B2_02	119	37	1.00	1.00	1.00	0.547	0.083	3.715	0.626	1.14
MB_B2_03	120	150	1.00	1.00	1.50	0.396	0.050	3.768	0.461	1.16
MB_B2_04	121	7	1.00	1.00	1.00	0.540	0.078	0.891	0.248	0.46

Domain	Code	Count	Average Length m	Minimum Length m	Maximum Length m	Average Au g/t	Minimum Au g/t	Maximum Au g/t	Std Dev g/t	CoV
MB_B2_05	122	1,385	1.00	1.00	1.50	0.595	0.007	39.368	1.488	2.50
MB_B2_06	123	1,514	1.04	1.00	3.50	0.354	0.024	11.884	0.565	1.60
MB_B2_07	124	1,669	1.02	1.00	3.50	0.422	0.024	12.204	0.518	1.23
MB_B2_08	125	4,735	1.00	0.70	5.55	0.498	0.001	27.059	0.837	1.68
MB_B2_09	126	1,723	1.13	0.60	4.50	1.215	0.023	50.158	2.744	2.26
MB_B2_10	127	17,562	1.07	0.50	9.00	0.685	0.001	66.836	1.493	2.18
MB_B2_11	128	871	1.06	0.75	2.55	0.589	0.004	66.363	2.390	4.06
MB_B2_12	129	6,186	1.02	0.60	3.50	0.578	0.004	71.050	1.452	2.51
MB_B2_13	130	1,284	1.05	0.80	4.05	0.480	0.016	31.798	1.041	2.17
MB_B2_14	131	1,789	1.05	0.75	4.50	0.477	0.005	15.842	0.824	1.72
MB_B2_15	132	352	1.01	1.00	2.50	0.353	0.010	7.840	0.660	1.87
MB_B2_16	133	85	1.02	1.00	1.50	0.418	0.039	2.940	0.484	1.16
MB_B2_17	134	94	1.04	1.00	1.70	0.430	0.009	4.275	0.561	1.31
MB_B2_18	135	180	1.01	1.00	2.50	0.622	0.016	15.361	1.336	2.15
MB_B2_HWA	136	18	1.11	1.00	2.00	0.402	0.150	0.874	0.199	0.50
MB_B2_HWB	137	199	1.04	1.00	2.00	0.376	0.008	6.576	0.572	1.52
MB_B2_HWC	138	830	1.06	0.70	4.50	0.871	0.014	102.807	3.853	4.42
MB_B2_HWC1	139	140	1.06	0.75	3.00	1.109	0.008	29.991	3.457	3.12
MB_B2_HWD	140	2,174	1.06	0.85	6.00	0.583	0.014	29.775	1.151	1.97
MB_B2_HWE	141	4,561	1.07	0.70	6.50	0.536	0.005	45.157	1.131	2.11
MB_B2_HWE1	142	432	1.06	0.50	4.50	0.402	0.023	7.839	0.598	1.49
MB_B2_HWF	143	159	1.13	1.00	2.00	0.427	0.016	2.214	0.381	0.89
MB_B2_HWG	144	760	1.03	0.50	3.00	0.472	0.023	31.156	1.199	2.54

Domain	Code	Count	Average Length m	Minimum Length m	Maximum Length m	Average Au g/t	Minimum Au g/t	Maximum Au g/t	Std Dev g/t	CoV
MB_B2_HWH	145	218	1.01	0.90	2.00	0.584	0.016	7.599	0.848	1.45
MB_CFU_01	146	669	1.02	0.70	2.55	0.581	0.016	63.264	2.490	4.28
MB_CFU_02	147	156	1.00	1.00	1.00	0.425	0.018	9.853	0.852	2.00
MB_CFU_03	148	27	1.02	1.00	1.50	0.592	0.142	5.381	1.038	1.72
MB_CFU_04	149	302	1.07	1.00	5.00	1.919	0.022	175.084	12.806	6.66
MB_CFU_05	150	561	1.09	1.00	3.00	0.631	0.009	65.894	2.934	4.65
MB_CFU_06	151	841	1.06	1.00	7.50	0.912	0.005	192.463	7.492	8.22
MB_CFU_07A	152	318	1.09	1.00	3.50	0.538	0.021	13.225	1.006	1.87
MB_CFU_08	153	465	1.12	1.00	4.00	0.785	0.003	60.406	3.250	4.18
MB_CFU_09	154	153	1.03	1.00	2.30	0.562	0.045	17.597	1.752	3.11
MB_CFU_10	155	43	1.05	1.00	2.00	0.512	0.046	4.716	0.826	1.59
MB_CFU_11	156	64	1.08	1.00	2.00	0.574	0.031	8.620	1.167	2.02
MB_CFU_12	157	69	1.14	1.00	3.00	0.404	0.023	1.590	0.363	0.89
MB_CFU_13	158	41	1.00	1.00	1.00	0.416	0.046	1.365	0.301	0.71
MB_CFU_14	159	65	1.05	1.00	2.00	0.666	0.013	7.980	1.159	1.73
MB_CFU_15	160	59	1.00	1.00	1.00	0.377	0.012	2.832	0.446	1.17
MB_CFU_16	161	22	1.09	1.00	2.00	0.273	0.032	0.545	0.184	1.56
MB_CFU_16A	162	180	1.08	1.00	2.00	0.665	0.023	6.057	0.877	1.32
MB_CFU_17	163	181	1.00	1.00	1.00	0.426	0.034	15.986	1.267	2.96
MB_CFU_18	164	41	1.00	1.00	1.00	0.560	0.066	6.782	1.178	2.08
MB_P11_01	165	458	1.04	1.00	2.00	0.464	0.040	10.224	0.876	1.89
MB_P11_02	166	897	1.04	0.70	2.50	0.542	0.022	104.519	3.563	6.57
MB_P11_03	167	115	1.06	0.90	2.00	0.538	0.018	4.021	0.648	1.20

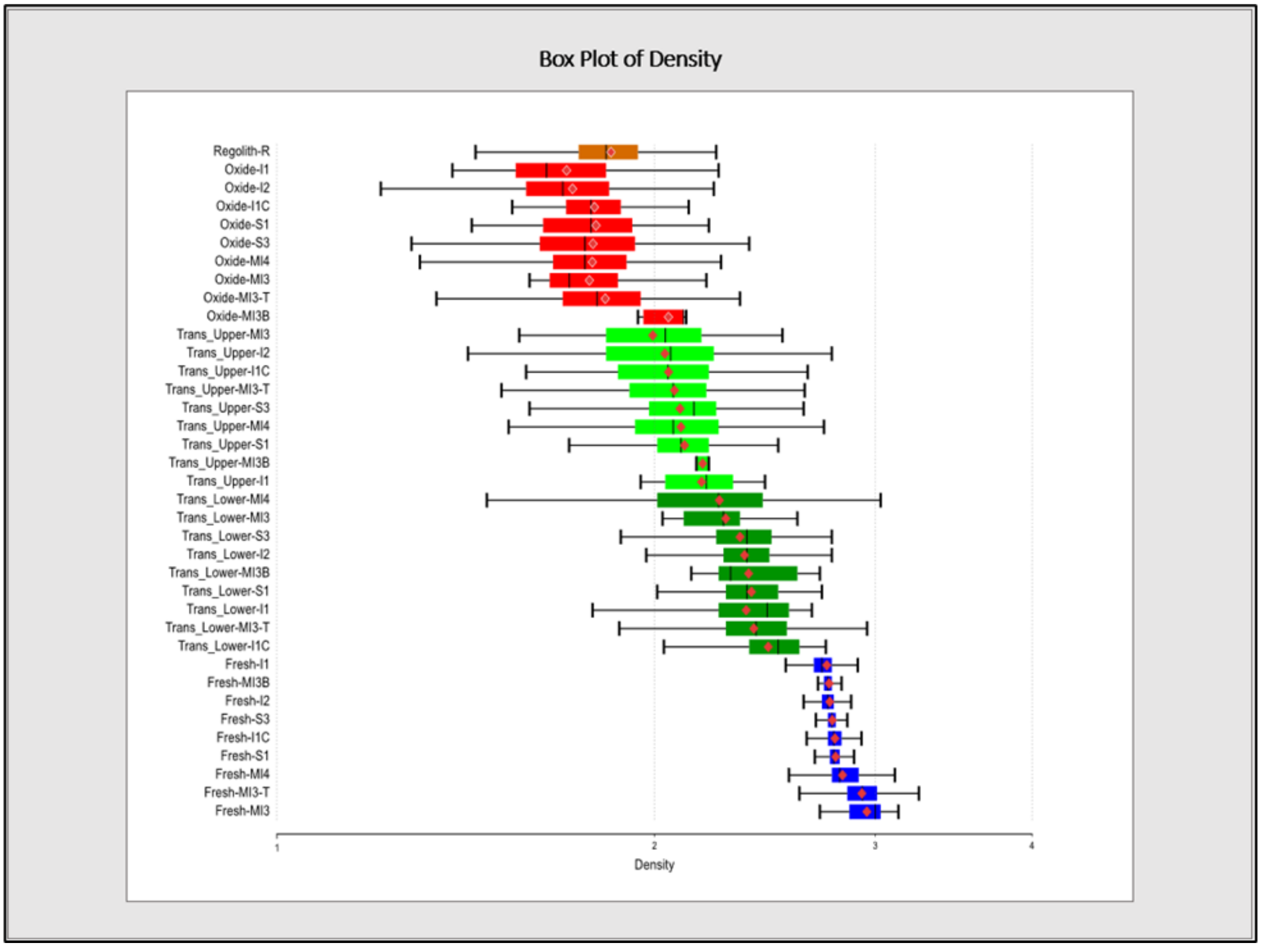
Domain	Code	Count	Average Length m	Minimum Length m	Maximum Length m	Average Au g/t	Minimum Au g/t	Maximum Au g/t	Std Dev g/t	CoV
MB_P11_04	168	198	1.05	1.00	2.00	0.420	0.016	5.253	0.493	1.17
MB_P11_05	169	151	1.00	1.00	1.00	0.567	0.064	5.646	0.659	1.16
MB_P11_06	170	109	1.01	1.00	2.00	0.457	0.023	9.137	0.908	1.99
MB_P11_07	171	26	1.00	1.00	1.00	0.534	0.043	3.009	0.574	1.07
MB_P11_08	172	29	1.13	0.75	2.00	0.619	0.032	2.385	0.675	1.09
MB_P11_09	173	565	1.13	1.00	2.50	0.614	0.045	55.000	2.990	4.87
MB_P11_10	174	4,582	1.06	0.70	2.55	0.580	0.001	96.254	1.680	2.90
MB_P11_11	175	210	1.00	1.00	1.50	0.299	0.034	1.289	0.190	0.63
MB_P11_12	176	82	1.00	1.00	1.00	0.340	0.138	1.633	0.276	0.81
MB_P11_13	177	95	1.00	1.00	1.00	0.254	0.062	1.285	0.174	0.68
MB_P11_14	178	17	1.00	1.00	1.00	0.417	0.174	2.085	0.470	1.13
MB_P11_15	179	26	1.00	1.00	1.00	0.309	0.084	0.720	0.184	0.59
MB_P11_16	180	3,822	1.04	0.80	4.40	0.466	0.007	13.047	0.652	1.40
MB_P11_17	181	34	1.00	1.00	1.00	0.577	0.066	2.221	0.448	0.78
MB_P11_18	182	14	1.00	1.00	1.00	0.534	0.156	1.651	0.400	0.75

The supplied database contains 36,231 point bulk density measurements from drill hole core, with values ranging from 1.09 to 3.77 t/m^3 (Table 14.4.3). Bulk density measurements were back-tagged by mineralization domain, lithology and oxidation state models. Bulk density measurements display a differing range of values based on oxidation state (Figure 14.4.3).

Table 14.4.3 B2 Zone Bulk Density Summary Statistics

Unit	Count	Average Bulk Density t/m^3	Minimum Bulk Density t/m^3	Maximum Bulk Density t/m^3	Std Dev	CoV
Regolith	263	1.84	1.29	2.50	0.17	0.09
Oxide	4,680	1.77	1.09	3.04	0.20	
Transitional Upper	2,254	2.07	1.41	2.96	0.24	
Transitional Lower	3,746	2.38	1.43	3.15	0.21	
Fresh	25,288	2.83	1.84	3.77	0.12	
Total	**36,231**	**2.59**	**1.09**	**3.77**	**0.42**	**0.16**

Figure 14.4.3 Boxplot of B2 Zone Bulk Density (t/m³) by Oxide / Lithology State



14.4.3 Block Model

A rotated block model was established using SurpacTM 2022 software with the block model limits selected so as to cover the extent of the mineralized structures, the potential open pit dimensions, and to reflect the general orientation of the mineralized zones (Table 14.4.4). The block model consists of separate variables for estimated grades, rock codes, bulk density and classification attributes. Separate variables were coded with unique mineralization domain, lithology and oxide state rock codes.

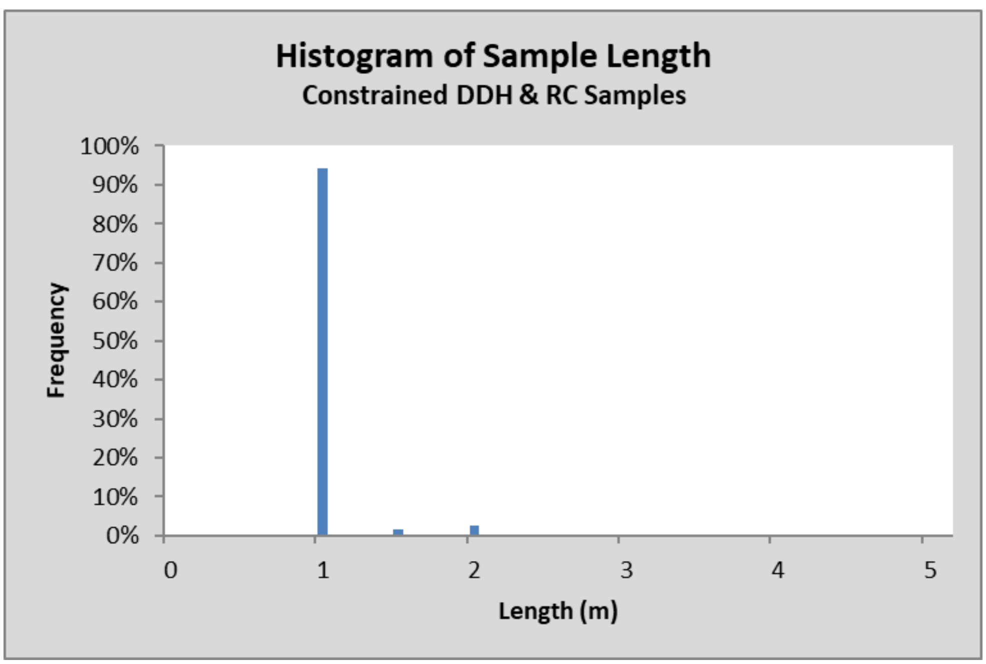
Table 14.4.4 B2 Zone Block Model Setup

Coordinate	Minimum	Maximum	Block Size m	Number of Blocks
Easting (X)	725,208.5	729,654.5	2.00	2,223
Northing (Y)	1,351,533	1,355,995.5	6.25	714
Elevation (max Z)	-100	350	3.00	150
Rotation	42° clockwise			

14.4.4 Compositing

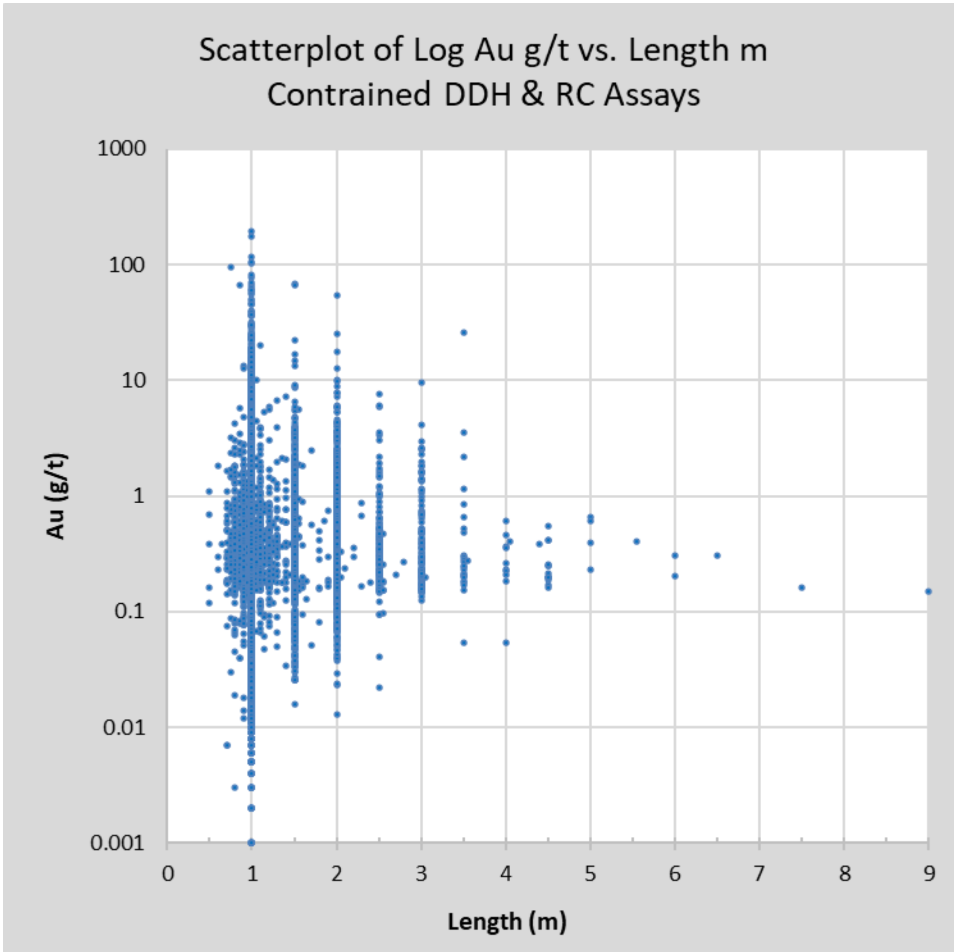
Assay sample lengths for DD and RC drill holes range from 0.50 to 9.00 m, with an average sample length of 1.05 m (Figure 14.4.4). The average sample length for DD assays is 1.11 m, and the average sample length for RC assays is 1.02 m. Constrained assay sample lengths within the defined mineralization domains range from 0.50 to 9.0 m, with an average sample length of 1.05 m. Approximately 93% of the constrained assay samples display a sample length equal to 1.00 m.

Figure 14.4.4 Histogram of B2 Zone Model Constrained Assay Sample Lengths



No correlation was observed between sample grade and sample length for the constrained assay samples (Figure 14.4.5).

Figure 14.4.5 Scatterplot of B2 Zone Assay Sample Lengths Versus Grade



Based on the predominance of 1.00 m sample lengths, Orezone decided that 1 m downhole composites are appropriate for all constrained capped assay samples. The downhole compositing process used a 'best fit' approach that results in composites of slightly variable length, of equal contiguous length within a given drill hole intersection, ensuring the composite length is as close as possible to the nominated 1m composite length. Length-weighted composites were calculated within the defined mineralization domains. Missing sample intervals in the data were ignored and treated as null values during the compositing process. The compositing process started at the first point of intersection between the drill hole and the domain wireframe intersected and halted upon exit from the wireframe. Residual composites less than 1.00 m were retained. The wireframes that represent the interpreted mineralization domains were used to back-tag a rock code variable into the composite workspace. The composite data were then visually validated against the mineralization wireframes and extracted for analysis and grade estimation. Summary composite statistics are listed in Table 14.4.5.

Table 14.4.5 B2 Zone Composite Summary Statistics

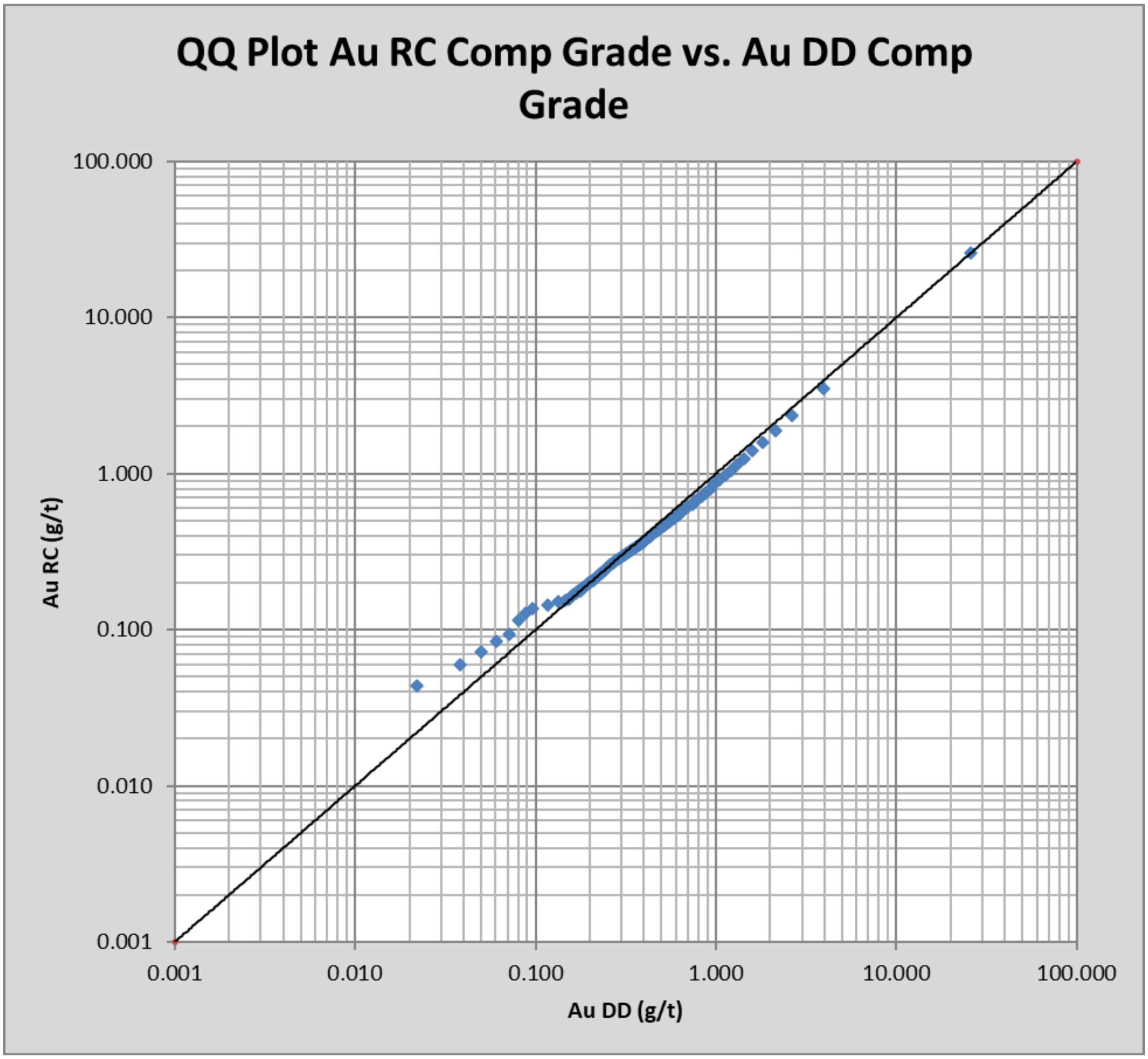
Domain	Count	Avg Au g/t	Min Au g/t	Max Au g/t	Std Dev g/t	CoV
101	18	0.619	0.152	1.500	0.449	0.71
102	51	0.651	0.074	3.335	0.727	1.11
103	61	0.414	0.122	1.200	0.280	0.67
104	19,988	0.408	0.001	10.000	0.481	1.18
105	2,716	0.427	0.016	5.000	0.478	1.12
106	40	0.512	0.078	2.091	0.421	0.81
107	276	0.355	0.025	2.060	0.306	0.86
108	58	0.382	0.079	1.338	0.310	0.80
109	8,702	0.395	0.008	8.000	0.489	1.24
110	587	0.527	0.009	7.000	0.755	1.43
111	100	0.451	0.061	2.302	0.475	1.05
112	169	0.332	0.006	1.641	0.274	0.82
113	7	0.553	0.228	1.209	0.340	0.57
114	26	0.418	0.093	1.369	0.359	0.84
115	175	0.438	0.024	2.500	0.442	1.01
116	171	0.524	0.071	4.000	0.614	1.17
117	96	0.625	0.020	4.000	0.831	1.32
118	44	0.392	0.088	1.000	0.238	0.60
119	37	0.454	0.083	1.000	0.285	0.62
120	151	0.382	0.050	2.000	0.371	0.97
121	41	1.249	0.078	12.861	2.121	1.68
122	1,406	0.541	0.007	7.000	0.788	1.46
123	2,283	0.265	0.014	2.500	0.264	1.00
124	1,716	0.410	0.024	3.000	0.392	0.96
125	6,068	0.405	0.001	10.000	0.593	1.46
126	1,953	1.101	0.016	12.000	1.819	1.65
127	18,702	0.672	0.001	26.000	1.246	1.85
128	926	0.500	0.004	7.000	0.685	1.37
129	6,324	0.551	0.004	11.000	0.787	1.43
130	1,366	0.457	0.023	5.000	0.503	1.10
131	1,871	0.459	0.005	6.000	0.598	1.30
132	353	0.323	0.010	3.000	0.388	1.20
133	87	0.389	0.039	1.500	0.356	0.91
134	98	0.379	0.009	1.500	0.323	0.85

Domain	Count	Avg Au g/t	Min Au g/t	Max Au g/t	Std Dev g/t	CoV
135	184	0.518	0.016	3.000	0.569	1.10
136	230	0.365	0.021	3.240	0.360	0.98
137	212	0.353	0.008	2.000	0.352	0.99
138	872	0.706	0.014	12.000	1.334	1.89
139	150	0.676	0.008	5.000	1.076	1.59
140	2,316	0.548	0.014	8.000	0.842	1.53
141	4,873	0.510	0.005	8.000	0.737	1.44
142	463	0.370	0.023	3.000	0.383	1.03
143	181	0.423	0.016	1.800	0.358	0.84
144	790	0.443	0.028	3.000	0.443	1.00
145	219	0.530	0.016	2.500	0.541	1.02
146	683	0.494	0.016	6.000	0.590	1.19
147	160	0.373	0.017	2.500	0.417	1.11
148	28	0.350	0.142	0.800	0.227	0.63
149	333	0.698	0.017	7.000	1.016	1.45
150	614	0.523	0.011	9.000	0.915	1.75
151	917	0.498	0.005	7.000	0.756	1.52
152	354	0.504	0.021	4.000	0.648	1.28
153	521	0.716	0.003	15.000	1.699	2.37
154	158	0.413	0.047	4.000	0.598	1.44
155	45	0.337	0.046	0.800	0.239	0.70
156	69	0.495	0.031	2.946	0.654	1.31
157	89	0.352	0.018	1.590	0.339	0.96
158	41	0.410	0.046	1.365	0.299	0.72
159	68	0.604	0.013	3.981	0.798	1.31
160	59	0.329	0.012	1.000	0.256	0.77
161	25	0.279	0.032	0.600	0.186	0.65
162	207	0.592	0.023	3.000	0.703	1.19
163	181	0.306	0.034	1.200	0.234	0.76
164	41	0.317	0.066	0.800	0.202	0.63
165	676	0.327	0.007	3.500	0.459	1.40
166	937	0.419	0.023	8.000	0.697	1.66
167	122	0.502	0.018	2.000	0.485	0.96
168	194	0.387	0.016	2.000	0.337	0.87
169	183	0.469	0.014	2.500	0.500	1.06

Domain	Count	Avg Au g/t	Min Au g/t	Max Au g/t	Std Dev g/t	CoV
170	113	0.383	0.023	2.000	0.375	0.98
171	26	0.476	0.043	1.500	0.344	0.71
172	33	0.570	0.032	2.385	0.627	1.08
173	641	0.486	0.045	8.000	0.984	2.02
174	4,862	0.553	0.001	11.000	0.690	1.25
175	213	0.296	0.009	1.289	0.188	0.63
176	82	0.340	0.138	1.633	0.276	0.81
177	96	0.260	0.076	1.267	0.180	0.69
178	17	0.353	0.174	1.000	0.254	0.70
179	26	0.309	0.084	0.720	0.184	0.58
180	3,949	0.452	0.005	5.000	0.504	1.12
181	28	0.530	0.066	1.500	0.353	0.65
182	14	0.480	0.156	1.000	0.263	0.53
Total	103,962	0.503	0.001	26.000	0.816	1.62

Examination of the RC and DD composite grade distributions suggests a positive bias between the RC and DD composite sample populations for grades <0.1 g/t, and a negative bias between these two populations for grades >0.3 g/t (Figure 14.4.6). These biases are not considered critical, and no correction factors were applied.

Figure 14.4.6 QQ Plot of B2 Zone RC Composite Grades versus DD Composite Grades



14.4.5 Treatment of Extreme Values

Capping thresholds were determined for each of the mineralized domains prior to compositing of the assay data using histograms with cumulative frequency, normal distribution, log-probability plots and graphical inspection of the spatial grade distribution. Potential outliers are not markedly clustered in localized high-grade areas and sub-domaining is therefore not warranted. Raw assays are capped to the defined threshold prior to compositing within the mineralization domains. A total of eight domains do not require capping (Table 14.4.6). The average capped assay grade is 6% lower than the raw assay grade.

Table 14.4.6 B2 Zone Capping Thresholds and Summary Statistics Table

Domain	Count	Cap g/t	Number Capped	Uncapped Mean g/t	Capped Mean g/t	Mean Above Cap g/t	Change %
101	18	1.50	2	0.693	0.619	2.160	-11
102	51	3.34	0	0.652	0.652		0
103	61	1.20	2	0.433	0.414	1.780	-4
104	19,516	10.00	8	0.414	0.410	17.684	-1
105	2,666	5.00	7	0.434	0.427	7.474	-1
106	40	2.09	1	0.512	0.512	2.091	0
107	273	2.06	1	0.354	0.354	2.060	0
108	57	1.34	0	0.366	0.366		0
109	8,484	8.00	6	0.401	0.397	13.804	-1
110	356	7.00	2	0.657	0.427	47.880	-35
111	100	2.35	1	0.454	0.454	2.350	0
112	168	1.64	1	0.333	0.333	1.641	0
113	7	1.21	0	0.553	0.553		0
114	26	1.37	0	0.418	0.418		0
115	172	2.50	4	0.489	0.443	4.489	-9
116	130	4.00	1	0.929	0.531	55.762	-43
117	96	4.00	2	0.665	0.625	5.926	-6
118	44	1.00	3	0.417	0.392	1.368	-6
119	37	1.00	3	0.547	0.454	2.141	-17
120	150	2.00	2	0.396	0.381	3.137	-4
121	7	0.89	1	0.540	0.540	0.891	0
122	1,385	7.00	6	0.595	0.550	17.271	-7
123	1,514	2.50	8	0.354	0.332	6.553	-6
124	1,669	3.00	6	0.422	0.413	5.517	-2
125	4,735	10.00	4	0.498	0.490	18.543	-1
126	1,723	12.00	21	1.215	1.112	20.499	-9
127	17,562	26.00	7	0.685	0.678	45.512	-1
128	871	7.00	3	0.589	0.507	30.788	-14
129	6,186	11.00	8	0.578	0.553	30.158	-4
130	1,284	5.00	4	0.480	0.453	13.793	-6
131	1,789	6.00	9	0.477	0.461	9.360	-4
132	352	3.00	2	0.353	0.326	7.662	-8
133	85	1.50	3	0.418	0.387	2.383	-7

Domain	Count	Cap g/t	Number Capped	Uncapped Mean g/t	Capped Mean g/t	Mean Above Cap g/t	Change %
134	94	1.50	3	0.430	0.382	2.988	-11
135	180	3.00	3	0.622	0.524	8.851	-16
136	18	0.87	1	0.402	0.401	0.874	0
137	199	2.00	2	0.376	0.350	4.640	-7
138	830	12.00	5	0.871	0.744	33.132	-15
139	140	5.00	5	1.109	0.727	15.708	-34
140	2,174	8.00	8	0.583	0.563	13.436	-3
141	4,561	8.00	14	0.536	0.515	14.610	-4
142	432	3.00	4	0.402	0.380	5.317	-5
143	159	1.80	1	0.427	0.424	2.214	-1
144	760	3.00	6	0.472	0.432	8.067	-8
145	218	2.50	5	0.584	0.530	4.852	-9
146	669	6.00	1	0.581	0.495	63.264	-15
147	156	2.50	1	0.425	0.378	9.853	-11
148	27	0.80	4	0.592	0.357	2.391	-40
149	302	7.00	4	1.919	0.714	97.926	-63
150	561	9.00	2	0.631	0.521	40.049	-18
151	841	7.00	5	0.912	0.510	74.549	-44
152	318	4.00	3	0.538	0.497	8.336	-8
153	465	12.00	3	0.785	0.651	32.851	-17
154	153	4.00	2	0.562	0.418	14.985	-26
155	43	0.80	5	0.512	0.341	2.269	-33
156	64	3.00	1	0.574	0.486	8.620	-15
157	69	1.59	1	0.404	0.404	1.590	0
158	41	1.37	0	0.416	0.416		0
159	65	4.00	1	0.666	0.605	7.980	-9
160	59	1.00	3	0.377	0.332	1.886	-12
161	22	0.55	0	0.273	0.273		0
162	180	3.00	6	0.665	0.636	3.867	-4
163	181	1.20	5	0.426	0.307	5.523	-28
164	41	0.80	3	0.560	0.317	4.127	-43
165	458	3.50	6	0.464	0.418	6.972	-10
166	897	8.00	3	0.542	0.427	42.636	-21
167	115	2.00	5	0.538	0.498	2.929	-8
168	198	2.00	3	0.420	0.402	3.135	-4

Domain	Count	Cap g/t	Number Capped	Uncapped Mean g/t	Capped Mean g/t	Mean Above Cap g/t	Change %
169	151	2.50	2	0.567	0.542	4.354	-4
170	109	2.00	1	0.457	0.391	9.137	-14
171	26	1.50	1	0.534	0.476	3.009	-11
172	29	2.39	0	0.619	0.619		0
173	565	8.00	3	0.614	0.458	37.528	-26
174	4,582	11.00	3	0.580	0.553	51.906	-5
175	210	1.29	0	0.299	0.299		0
176	82	1.63	1	0.340	0.340	1.633	0
177	95	0.80	1	0.254	0.249	1.285	-2
178	17	1.00	1	0.417	0.353	2.085	-15
179	26	0.72	1	0.309	0.309	0.720	0
180	3,822	5.00	13	0.466	0.453	8.757	-3
181	34	1.50	3	0.577	0.552	1.786	-4
182	14	1.00	2	0.534	0.480	1.379	-10
Total	**97,066**		**284**	**0.541**	**0.512**	**15.801**	**-5**

14.4.6 Variography & Continuity Analysis

Orezone performed three-dimensional continuity analyses (variography) within the mineralized sub-domains to determine appropriate estimation inputs to the grade interpolation process. The variogram modelling process followed by Orezone involves the following steps:

- Calculate and model the downhole variogram to characterize the Nugget Effect.
- Calculate a fan of variograms within the plane of greatest continuity to identify the direction of maximum continuity within the plane.
- Model the variogram in the direction of maximum continuity and the orthogonal directions.

Variogram parameters were derived for the better sampled mineralization sub-domains; that is, MB_B1_01, MB_B1_06, MB_B1_39, MB_B1_40, MB_B1_32A, and MB_P11_10. Acceptable variogram modelling for the sparsely sampled sub-domains was not possible, and the variography used to estimate these sub-domains has been adopted from adjacent mineralized domains, but the directions and anisotropy ratios have been tailored to best suit the geometry of each of the sub-domains.

Better variogram ranges from the above-mentioned sampled mineralized domains were, respectively, 56 m, 45 m, 51 m, 50 m, 51 m, and 54 m, indicating that the maximum spatial continuity is greater than the average drill hole spacing.

14.4.7 Estimation and Classification

The bulk density data were extracted from the drill hole database and were grouped according to their host rock lithology within the Fresh weathering unit, or within their host weathering unit above the Fresh weathering unit. Separate capping values were determined for each lithology type and oxidation state. The bulk density values for each block were then estimated using the capped bulk density values and the Inverse Distance Squared (ID2) anisotropic linear interpolation using a minimum of three and maximum of 15 composites within a 120 m diameter search envelope. Sample selection was restricted to a maximum of three composite samples from a single drill hole. Bulk density estimation was performed by weathering unit within the weathered portion of the deposits, and by lithological domain within the fresh Birimian portion of the deposits. These weathering and lithological units define hard boundaries for bulk density estimation. For those areas of limited sample data, the mean value of the specific lithology type in the Fresh domain or within the weathering unit above the Fresh unit was applied. Orezone applies a reduction factor of 5.5% and 2.5% to individual samples in the oxide and transition zones, respectively, for the block model dry bulk density calculations. Bulk density block values were adjusted by these factors after estimation.

Block grades for gold were estimated by Inverse Distance Cubed (ID3) weighting of capped composites in three separate passes, using a minimum of five and maximum of 15 composites for Pass 1, and a minimum of three and maximum of 15 composites for Pass 2 and Pass 3. Sample selection was restricted to a maximum of two composite samples from a single drill hole.

The orientation of the search ellipsoids was defined by Orezone geologists based on the local geology within each respective mineralization domain. The defined orientations were then converted to SurpacTM format rotations. Capped Ordinary Kriging (OK) and capped Nearest Neighbour models (NN) were also generated using the same search parameters. Only composites derived from DD and RC drill holes were used for grade estimation. Search and grade estimation were constrained by the individual mineralization domains, which define hard boundaries for grade estimation.

The parameters used to define classification limits included spatial analysis, drill hole spacing distance from nearest grade composite, and the observed continuity of the mineralization. Mineral Resources were classified algorithmically as follows:

Measured Mineral Resources are defined by:

- A drill spacing of less than 25 m x 25 m.

- Average True Distance to Samples: 0 to 25 m.

- Conditional bias slope >0.75.
- A minimum of five composite samples.
- A maximum of two composite samples per drill hole.
- Three or more drill holes.
- Estimation Pass 1.

Indicated Mineral Resources are defined by:

- A drill spacing of less than 50 m x 50 m.
- Average True Distance to Sample: 25 to 50 m.
- A minimum of three composite samples.
- A maximum of two composite samples per drill hole.
- Two or more drill holes.
- Estimation Pass 1 or 2.

Inferred Mineral Resources include all estimated mineralization defined by:

- A drill spacing greater than 50 m x 50 m.
- Average True Distance to sample: 50 to 100 m.
- Two or more drill holes.
- A minimum of three composite samples.
- A maximum of two composite samples per drill hole.
- Estimation Pass 1, 2 or 3.

All other mineralization remained unclassified.

14.4.8 Mineral Resource Estimate

Mineral Resources reported herein have been constrained within an optimized pit shell (Figure 14.4.7). The results from the optimized pit shell are used solely for the purpose of reporting Mineral Resources and include Measured, Indicated and Inferred Mineral Resources. Orezone reports whole block volumes for only those blocks where the block centroid lies within the controlling wireframe. Orezone applies a 0.95 factor to the B2 Zone oxide grade to account for artisanal mining.

Mineral Resources are reported based on the cut-offs listed in Table 14.4.7 and a gold price of USD 1,700/oz.

Table 14.4.7 B2 Zone MRE Reporting Cut-Off Grades

Unit	Au g/t
Regolith	0.25
Oxide	0.25
Trans Upper	0.25
Trans Lower	0.45
Fresh	0.45

The Mineral Resources have an effective date of March 28, 2023 (Table 14.4.8).

Figure 14.4.7 Isometric View of B2 Zone Optimized Pit Shell

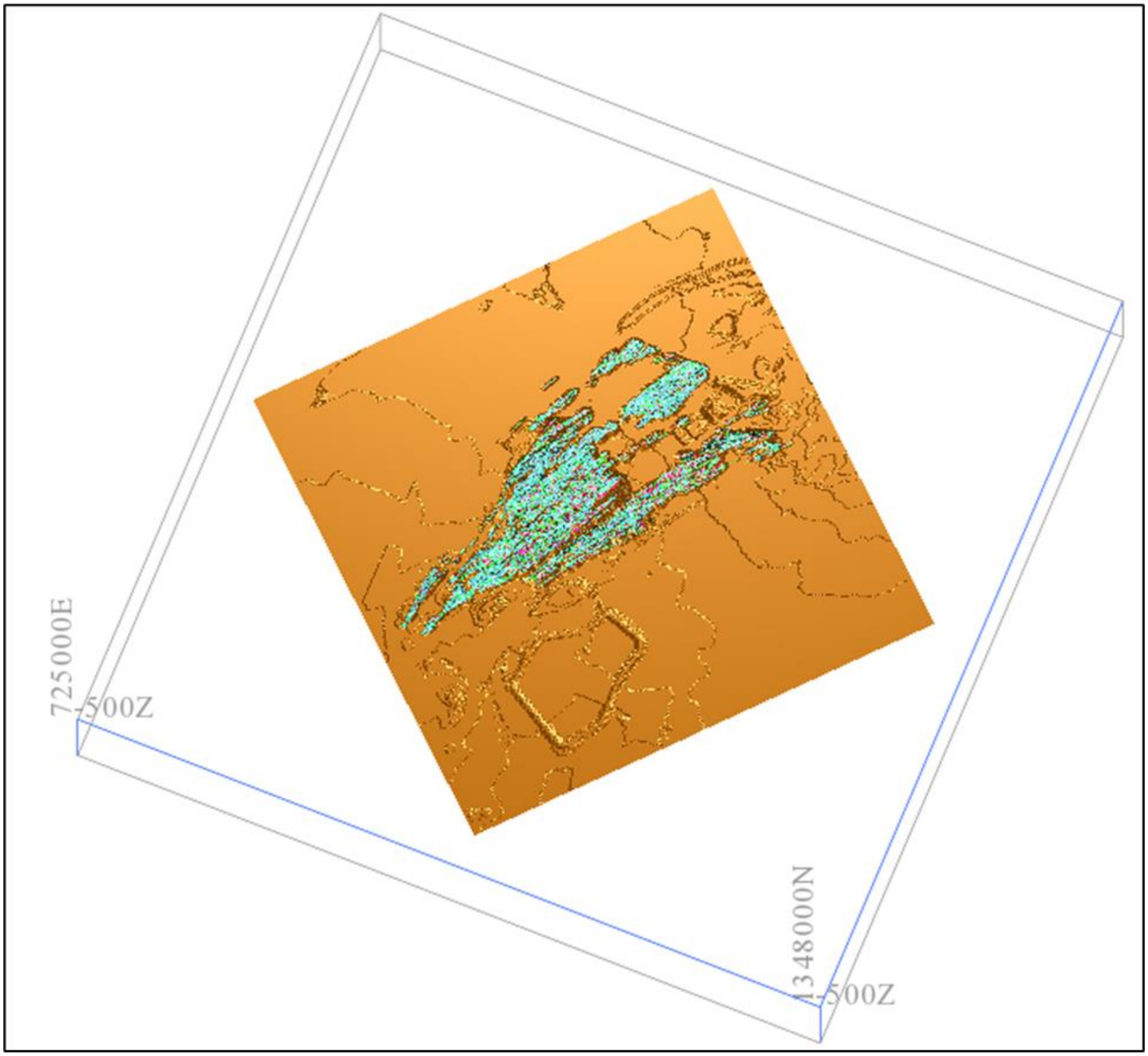


Table 14.4.8 B2 Zone Mineral Resource Estimates*

Total	Cut-off	Tonnes k	Au g/t	Au koz
Measured	NA	0	0.00	0
Indicated	NA	75,621	0.76	1,839
Meas + Ind	NA	75,621	0.76	1,839
Inferred	NA	12,638	0.97	392

Regolith	Cut-off	Tonnes k	Au g/t	Au koz
Measured	0.25	0	0.00	0
Indicated	0.25	0	0.00	0
Meas + Ind	0.25	0	0.00	0
Inferred	0.25	0	0.00	0

Oxide	Cut-off	Tonnes k	Au g/t	Au koz
Measured	0.25	0	0.00	0
Indicated	0.25	26,853	0.57	491
Meas + Ind	0.25	26,853	0.57	491
Inferred	0.25	1004	0.58	19

Trans Upper	Cut-off	Tonnes k	Au g/t	Au koz
Measured	0.25	0	0.00	0
Indicated	0.25	12,898	0.62	259
Mesa + Ind	0.25	12,898	0.62	259
Inferred	0.25	419	0.65	9

Trans Lower	Cut-off	Tonnes k	Au g/t	Au koz
Measured	0.45	0	0.00	0
Indicated	0.45	6,892	0.94	209
Meas + Ind	0.45	6,892	0.94	209
Inferred	0.45	241	0.97	8

Fresh	Cut-off	Tonnes k	Au g/t	Au koz
Measured	0.45	0	0.00	0
Indicated	0.45	28,978	0.94	879
Meas + Ind	0.45	28,978	0.94	879
Inferred	0.45	10,974	1.01	357

*Mineral Resources are inclusive of Mineral Reserves. Totals may differ due to rounding.

14.4.9 Validation

The block model was validated visually by the inspection of successive vertical cross-section lines and plan views in order to confirm that the block models correctly reflect the distribution of high-grade and low-grade values.

The mean block grade estimated for each sub-domain was then compared to the mean composite grade of the corresponding sub-domain (Table 14.4.9). Although these two parameters are not strictly comparable due to data clustering and volume influences, they do provide a useful validation tool in detecting any major biases and allow the comparison between input composite grade and the estimated block grade.

Table 14.4.9 Comparison of B2 Zone Au Composite Grade and OK, ID3 and NN Estimated Block Grades

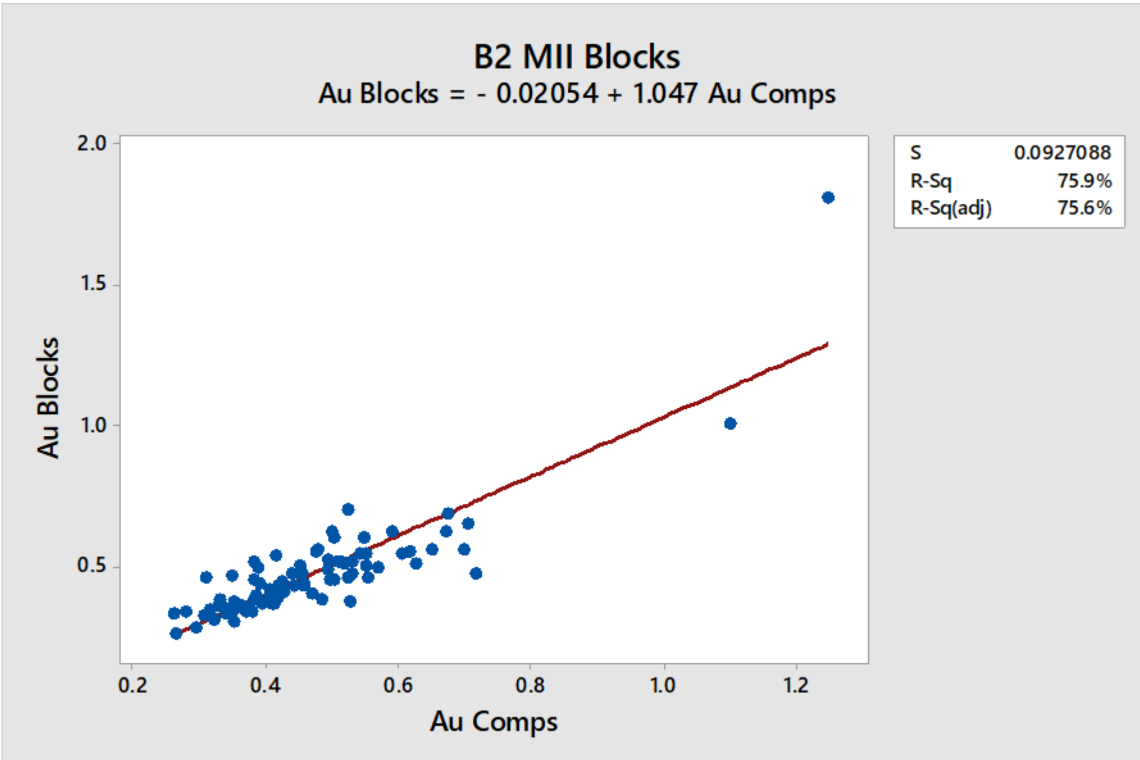
Domain	Tonnes	Number of Samples	Mean Composite Au Grade g/t	OK Estimated Mean Au Grade g/t	IDW Estimated Mean Au Grade g/t	NN Estimated Mean Au Grade g/t	OK Relative Difference %	IDW Relative Difference %	NN Relative Difference %
101	378,975	18	0.62	0.54	0.55	0.59	-13	-11	-5
102	483,730	51	0.65	0.53	0.56	0.55	-18	-14	-15
103	243,709	61	0.41	0.53	0.54	0.49	29	32	20
104	64,143,374	19,988	0.41	0.39	0.39	0.38	-5	-5	-7
105	25,725,124	2,716	0.43	0.42	0.41	0.38	-2	-5	-12
106	237,446	40	0.51	0.53	0.52	0.43	4	2	-16
107	2,751,807	276	0.35	0.36	0.36	0.32	3	3	-9
108	807,502	58	0.38	0.43	0.45	0.53	13	18	39
109	32,185,003	8,702	0.39	0.36	0.36	0.35	-8	-8	-10
110	1,625,463	363	0.43	0.38	0.37	0.37	-12	-14	-14
111	780,895	100	0.45	0.52	0.50	0.49	16	11	9
112	661,931	169	0.33	0.37	0.38	0.35	12	15	6
113	78,666	7	0.55	0.44	0.45	0.53	-20	-18	-4
114	67,127	26	0.42	0.44	0.43	0.31	5	2	-26
115	681,514	175	0.44	0.50	0.47	0.46	14	7	5
116	683,560	171	0.52	0.60	0.70	0.69	15	35	33
117	565,524	96	0.63	0.52	0.50	0.51	-17	-21	-19
118	912,802	44	0.39	0.46	0.44	0.34	18	13	-13
119	422,440	37	0.45	0.50	0.47	0.39	11	4	-13
120	300,828	151	0.38	0.46	0.52	0.55	21	37	45
121	330,820	41	1.25	1.54	1.80	3.05	23	44	144
122	7,469,205	1,406	0.54	0.52	0.54	0.54	-4	0	0
123	12,678,927	2,283	0.27	0.26	0.26	0.26	-4	-4	-4
124	12,759,341	1,716	0.41	0.37	0.37	0.36	-10	-10	-12
125	40,254,807	6,068	0.41	0.43	0.41	0.40	5	0	-2

Domain	Tonnes	Number of Samples	Mean Composite Au Grade g/t	OK Estimated Mean Au Grade g/t	IDW Estimated Mean Au Grade g/t	NN Estimated Mean Au Grade g/t	OK Relative Difference %	IDW Relative Difference %	NN Relative Difference %
126	3,874,114	1,953	1.1	1.02	1.01	0.96	-7	-8	-13
127	94,415,832	18,702	0.67	0.63	0.62	0.62	-6	-7	-7
128	4,089,068	926	0.5	0.58	0.62	0.72	16	24	44
129	28,423,218	6,324	0.55	0.50	0.50	0.47	-9	-9	-15
130	5,678,622	1,366	0.46	0.42	0.43	0.42	-9	-7	-9
131	11,715,121	1,871	0.46	0.45	0.44	0.44	-2	-4	-4
132	783,619	353	0.32	0.33	0.31	0.28	3	-3	-13
133	147,090	87	0.39	0.52	0.49	0.48	33	26	23
134	843,313	98	0.38	0.34	0.33	0.32	-11	-13	-16
135	3,026,511	184	0.52	0.48	0.50	0.47	-8	-4	-10
136	387,753	230	0.37	0.37	0.36	0.35	0	-3	-5
137	1,132,852	212	0.35	0.39	0.37	0.36	11	6	3
138	5,620,430	872	0.71	0.64	0.65	0.59	-10	-8	-17
139	713,877	150	0.68	0.72	0.68	0.77	6	0	13
140	12,693,874	2,316	0.55	0.58	0.60	0.59	5	9	7
141	26,890,389	4,873	0.51	0.51	0.51	0.51	0	0	0
142	2,438,558	463	0.37	0.33	0.34	0.32	-11	-8	-14
143	802,799	181	0.42	0.46	0.44	0.44	10	5	5
144	4,116,442	790	0.44	0.44	0.43	0.40	0	-2	-9
145	1,042,294	219	0.53	0.49	0.47	0.40	-8	-11	-25
146	2,985,995	683	0.49	0.48	0.48	0.51	-2	-2	4
147	1,487,966	160	0.37	0.36	0.35	0.31	-3	-5	-16
148	155,321	28	0.35	0.43	0.46	0.37	23	31	6
149	852,773	333	0.7	0.57	0.56	0.52	-19	-20	-26
150	1,282,976	614	0.52	0.47	0.45	0.44	-10	-13	-15
151	1,443,716	917	0.5	0.47	0.45	0.44	-6	-10	-12
152	1,348,935	354	0.5	0.58	0.60	0.69	16	20	38
153	2,295,552	521	0.72	0.50	0.47	0.50	-31	-35	-31
154	449,497	158	0.41	0.37	0.37	0.37	-10	-10	-10
155	111,526	45	0.34	0.36	0.35	0.37	6	3	9
156	67,389	69	0.5	0.58	0.52	0.54	16	4	8
157	256,059	89	0.35	0.32	0.30	0.35	-9	-14	0
158	220,661	41	0.41	0.43	0.40	0.41	5	-2	0

Domain	Tonnes	Number of Samples	Mean Composite Au Grade g/t	OK Estimated Mean Au Grade g/t	IDW Estimated Mean Au Grade g/t	NN Estimated Mean Au Grade g/t	OK Relative Difference %	IDW Relative Difference %	NN Relative Difference %
159	121,307	68	0.6	0.53	0.54	0.55	-12	-10	-8
160	109,699	59	0.33	0.37	0.36	0.35	12	9	6
161	135,046	25	0.28	0.31	0.33	0.39	11	18	39
162	1,291,756	207	0.59	0.57	0.62	0.69	-3	5	17
163	1,312,770	181	0.31	0.31	0.32	0.35	0	3	13
164	147,289	41	0.32	0.36	0.34	0.31	13	6	-3
165	2,670,085	676	0.33	0.35	0.36	0.32	6	9	-3
166	3,033,401	937	0.42	0.39	0.38	0.37	-7	-10	-12
167	790,777	122	0.5	0.47	0.45	0.41	-6	-10	-18
168	1,996,436	194	0.39	0.40	0.40	0.39	3	3	0
169	444,574	183	0.47	0.42	0.40	0.38	-11	-15	-19
170	1,189,829	113	0.38	0.38	0.38	0.37	0	0	-3
171	615,339	26	0.48	0.56	0.55	0.61	17	15	27
172	211,403	33	0.57	0.51	0.49	0.56	-11	-14	-2
173	3927,347	641	0.49	0.37	0.38	0.38	-24	-22	-22
174	22,076,483	4,862	0.55	0.54	0.54	0.56	-2	-2	2
175	1,101,093	213	0.3	0.29	0.28	0.25	-3	-7	-17
176	335,239	82	0.34	0.35	0.33	0.33	3	-3	-3
177	824,194	96	0.26	0.36	0.33	0.33	38	27	27
178	52,315	17	0.35	0.33	0.34	0.36	-6	-3	3
179	236,181	26	0.31	0.42	0.45	0.45	35	45	45
180	26,553,055	3,949	0.45	0.47	0.47	0.46	4	4	2
181	508,275	28	0.53	0.46	0.51	0.56	-13	-4	6
182	22,331	14	0.48	0.56	0.55	0.58	17	15	21
Total	**498,730,887**	**103,962**	**0.50**	**0.48**	**0.48**	**0.47**	**-4**	**-4**	**-5**

An additional validation check was completed by comparing the average grade of the composites in a block to the associated model block grade estimate (Figure 14.4.8).

Figure 14.4.8 Validation Plot Between B2 Zone Block Grades and Average Composite Grades



The volume estimated was also checked against the reported volume of the individual mineralization domains. Estimated volumes are based on a 0.001 g/t cut-off and whole block volumes (Table 14.4.10). The results fall within acceptable limits for linear estimation.

Table 14.4.10 B2 Zone Volume Comparison

Mineralization Domains	Rock Code	Wireframe Volume k m^3	Estimated Whole Block Volume k m^3	Solid volume versus Block Volume %
MB_B1_03A	101	151	149	-1
MB_B1_06A	102	231	229	-1
MB_B1_12A	103	108	106	-2
MB_B1_13A	104	52,734	52,438	-1
MB_B1_20A	105	37,579	37,431	0
MB_B1_21A	106	121	120	-1
MB_B1_23A	107	2,085	2,074	-1
MB_B1_24A	108	345	341	-1

Mineralization Domains	Rock Code	Wireframe Volume k m³	Estimated Whole Block Volume k m³	Solid volume versus Block Volume %
MB_B1_32A	109	16,652	16,463	-1
MB_B1_36	110	851	840	-1
MB_B1_59	111	333	331	-1
MB_B1_60	112	319	309	-3
MB_B1_61A	113	35	34	-3
MB_B2_00	114	54	51	-6
MB_B2_00A	115	287	281	-2
MB_B2_00B	116	390	387	-1
MB_B2_00C	117	210	209	0
MB_B2_01	118	5,064	5,050	0
MB_B2_02	119	626	619	-1
MB_B2_03	120	193	189	-2
MB_B2_04	121	133	131	-2
MB_B2_05	122	3,183	3,140	-1
MB_B2_06	123	5,570	5,513	-1
MB_B2_07	124	5,928	5,862	-1
MB_B2_08	125	18,103	18,059	0
MB_B2_09	126	1,820	1,793	-1
MB_B2_10	127	44,540	44,281	-1
MB_B2_11	128	1,681	1,673	0
MB_B2_12	129	11,610	11,514	-1
MB_B2_13	130	2529	2,475	-2
MB_B2_14	131	5,271	5,205	-1
MB_B2_15	132	334	332	-1
MB_B2_16	133	64	61	-4
MB_B2_17	134	390	389	0
MB_B2_18	135	1,602	1,588	-1
MB_B2_HWA	136	208	204	-2
MB_B2_HWB	137	553	538	-3
MB_B2_HWC	138	2,434	2,427	0
MB_B2_HWC1	139	320	316	-1
MB_B2_HWD	140	6,414	6,367	-1
MB_B2_HWE	141	13,064	12,931	-1
MB_B2_HWE1	142	1,155	1,140	-1

Mineralization Domains	Rock Code	Wireframe Volume k m^3	Estimated Whole Block Volume k m^3	Solid volume versus Block Volume %
MB_B2_HWF	143	335	328	-2
MB_B2_HWG	144	1,843	1,820	-1
MB_B2_HWH	145	484	473	-2
MB_CFU_01	146	1,374	1,357	-1
MB_CFU_02	147	695	682	-2
MB_CFU_03	148	65	65	0
MB_CFU_04	149	360	354	-2
MB_CFU_05	150	591	583	-1
MB_CFU_06	151	684	681	-1
MB_CFU_07A	152	633	625	-1
MB_CFU_08	153	940	936	0
MB_CFU_09	154	204	201	-2
MB_CFU_10	155	52	51	-2
MB_CFU_11	156	39	39	-1
MB_CFU_12	157	111	111	0
MB_CFU_13	158	90	89	-1
MB_CFU_14	159	56	55	-3
MB_CFU_15	160	58	57	-2
MB_CFU_16A	162	599	592	-1
MB_CFU_17	163	569	562	-1
MB_CFU_18	164	64	65	1
MB_P11_01	165	1,569	1,588	1
MB_P11_02	166	1,649	1,622	-2
MB_P11_03	167	339	337	-1
MB_P11_04	168	1,607	1,599	-1
MB_P11_05	169	190	183	-4
MB_P11_06	170	545	541	-1
MB_P11_07	171	237	238	0
MB_P11_08	172	83	82	-1
MB_P11_09	173	2,354	2,333	-1
MB_P11_10	174	11,827	11,750	-1
MB_P11_11	175	442	436	-1
MB_P11_12	176	160	156	-3
MB_P11_13	177	422	423	0

Mineralization Domains	Rock Code	Wireframe Volume k m^3	Estimated Whole Block Volume k m^3	Solid volume versus Block Volume %
MB_P11_14	178	23	23	-3
MB_P11_15	179	224	224	0
MB_P11_16	180	30,961	30,829	0
MB_P11_17	181	227	227	0
MB_P11_18	182	9	8	-3
Total		**307,988**	**305,910**	**-1**

14.4.10 B2 Cut-off Sensitivity

The sensitivity of the Mineral Resource block grades to changes in cut-off grade was evaluated by calculating the total Mineral Resource inventory at various cut-offs within the Mineral Resource pit (Table 14.4.11). The values presented are intended for the sole purpose of demonstrating the sensitivity of the Mineral Resource with respect to the cut-off grade.

Table 14.4.11 B2 Grade Tonnage Sensitivity Table

OXIDE MEASURED + INDICATED				OXIDE INFERRED		
Au Cut-Off	Tonnes	Au		Au Cut-Off	Tonnes	Au
g/t	k	g/t		g/t	k	g/t
1.00	4,358.2	1.74		1.00	172.9	1.48
0.95	4,816.8	1.66		0.95	194.1	1.43
0.90	5,340.9	1.59		0.90	219.0	1.36
0.85	5,952.4	1.51		0.85	248.3	1.30
0.80	6,688.4	1.43		0.80	277.0	1.25
0.75	7,539.8	1.36		0.75	314.1	1.19
0.70	8,578.7	1.28		0.70	368.3	1.12
0.65	9,853.7	1.20		0.65	427.8	1.06
0.60	11,431.3	1.11		0.60	495.4	1.00
0.55	13,386.0	1.03		0.55	582.3	0.93
0.50	15,865.4	0.95		0.50	680.1	0.87
0.45	19,017.8	0.87		0.45	794.8	0.81
0.40	22,970.3	0.79		0.40	929.2	0.75
0.35	27,834.3	0.71		0.35	1,079.0	0.70
0.30	33,592.0	0.65		0.30	1,265.7	0.64
0.25	39,751.0	0.59		0.25	1,423.0	0.60

OXIDE MEASURED + INDICATED				OXIDE INFERRED		
Au Cut-Off	Tonnes	Au		Au Cut-Off	Tonnes	Au
g/t	k	g/t		g/t	k	g/t
0.20	45,061.0	0.54		0.20	1,529.5	0.58

SULPHIDE MEASURED + INDICATED				SULPHIDE INFERRED		
Au Cut-Off	Tonnes	Au		Au Cut-Off	Tonnes	Au
g/t	k	g/t		g/t	k	g/t
1.00	9,309.1	1.78		1.00	3,175.3	1.91
0.95	10,312.2	1.70		0.95	3,508.5	1.82
0.90	11,482.4	1.62		0.90	3,893.9	1.73
0.85	12,818.8	1.55		0.85	4,323.1	1.64
0.80	14,387.5	1.47		0.80	4,826.6	1.56
0.75	16,226.9	1.39		0.75	5,387.0	1.48
0.70	18,359.6	1.31		0.70	6,024.9	1.40
0.65	20,839.6	1.24		0.65	6,793.3	1.32
0.60	23,739.2	1.16		0.60	7,714.9	1.23
0.55	27,152.9	1.09		0.55	8,760.6	1.15
0.50	31,160.2	1.01		0.50	9,915.1	1.08
0.45	35,870.3	0.94		0.45	11,215.0	1.01
0.40	41,315.6	0.88		0.40	12,691.4	0.94
0.35	47,330.0	0.81		0.35	14,270.5	0.88
0.30	53,807.1	0.75		0.30	15,849.4	0.82
0.25	59,996.1	0.70		0.25	17,234.6	0.78
0.20	64,785.3	0.67		0.20	18,233.7	0.75

Oxide includes Regolith, Oxide and Transitional Upper units.
Sulphide includes Transitional Lower and Fresh units.
Totals may differ due to rounding.

14.5 Bomboré P11 Zone

14.5.1 Data Supplied

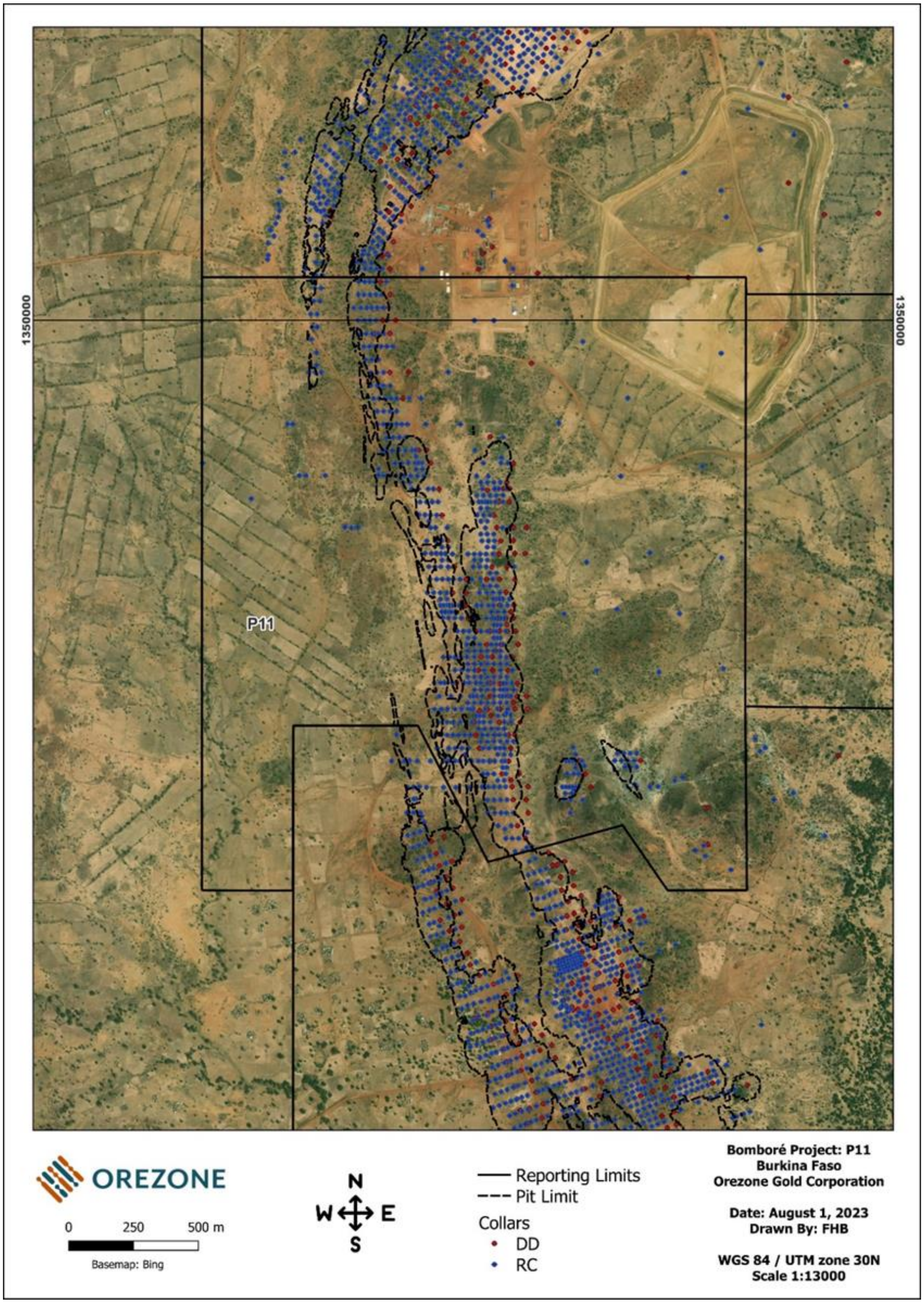
The P11 Zone model was developed by the Authors. Topography, mineralization, lithology and oxidation state three-dimensional wireframes were created by Orezone using Leapfrog Geo™ 2022.1.1. Drill hole distances are reported in metres and gold grades are reported as g/t. The drill hole collar coordinates were provided in the WGS84 Zone 30N UTM coordinate reference system.

The supplied drill hole database for the P11 Zone contains 1,670 unique drill hole collar records (Table 14.5.1 and Figure 14.5.1). Only Reverse Circulation (RC) and diamond drill (DD) holes were used for the Mineral Resource Estimate.

Table 14.5.1 P11 Zone Drill Hole Database Summary

Type	Count	Metres
Pressure Meter drill hole	1	20.5
Channel	12	19.6
Diamond Core Drill Hole (DD)	139	18,265.5
Auger Drill Hole	472	1,917.0
Rotary Air Blast Drill Hole (RAB)	206	6,017.0
Water Borehole	6	482.6
Reverse Circulation Drill Hole (RC)	637	34,136.0
Trench	197	1,916.0
Total	**1,670**	**62,774.2**

Figure 14.5.1 P11 Zone Drill Hole Collar Plot

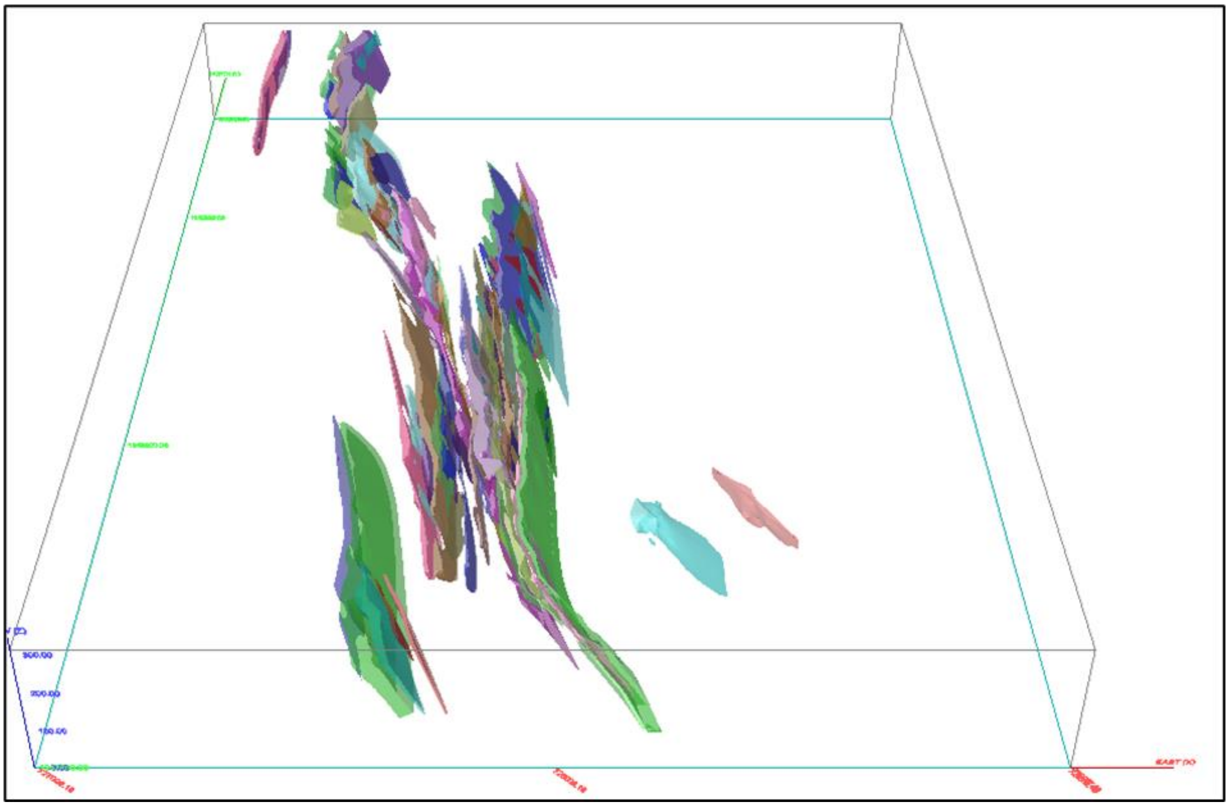


14.5.2 Exploratory Data Analysis

A total of 79 distinct mineralization domain wireframes were supplied by Orezone (Figure 14.5.2). Each domain was assigned a unique three-digit rock code.

The average nearest-neighbour collar distance for DD and RC combined is 29.1 m, the average DD length is 131.4 m and the average RC length is 53.5 m. Summary statistics for the supplied assay data are provided below (Table 14.5.2).

Figure 14.5.2 Isometric Plot of P11 Zone Mineralization Domains



View looking north. Field of view is approximately 2 km across.

Table 14.5.2 P11 Zone Assay Summary Statistics

Domain	Count	Avg Au g/t	Std Dev	CoV	Minimum Au g/t	Maximum Au g/t
101	564	0.80	1.54	1.94	0.002	14.20
102	1,454	0.87	2.00	2.28	0.001	38.18
103	135	0.80	1.94	2.43	0.004	16.85

Domain	Count	Avg Au g/t	Std Dev	CoV	Minimum Au g/t	Maximum Au g/t
104	34	0.36	0.40	1.12	0.004	1.62
105	69	0.36	0.59	1.66	0.016	3.58
106	59	0.35	0.91	2.64	0.006	5.06
107	49	0.26	0.39	1.47	0.041	2.62
108	44	1.86	7.36	3.96	0.010	48.85
109	21	0.21	0.19	0.91	0.020	0.79
110	21	0.25	0.19	0.75	0.006	0.55
111	28	0.30	0.27	0.91	0.017	1.15
112	487	0.63	1.87	2.98	0.002	36.72
113	1,432	0.56	2.45	4.40	0.001	89.94
114	335	0.47	0.69	1.48	0.001	4.92
115	1,155	0.51	1.13	2.22	0.001	27.00
116	35	0.49	0.71	1.44	0.001	4.06
117	212	0.49	0.77	1.58	0.001	6.40
118	407	0.44	0.54	1.23	0.003	4.66
119	220	0.57	2.21	3.92	0.002	32.19
120	87	0.43	0.54	1.25	0.001	2.82
121	24	0.47	0.56	1.19	0.004	2.30
122	62	0.68	1.60	2.35	0.006	12.23
123	21	0.37	0.40	1.08	0.004	1.12
124	16	0.29	0.30	1.05	0.020	1.12
125	423	0.43	0.58	1.33	0.002	5.76
126	18	0.26	0.35	1.36	0.007	1.52
127	10	0.34	0.40	1.17	0.006	1.26
128	36	0.41	0.57	1.40	0.005	2.56
129	25	0.21	0.24	1.14	0.003	0.99
130	164	0.34	0.41	1.19	0.002	3.00
131	49	0.32	0.35	1.07	0.001	1.94
132	36	0.36	0.44	1.23	0.003	1.85
133	178	0.33	0.33	1.00	0.002	2.56
134	190	0.83	0.92	1.11	0.001	5.90
135	172	0.51	0.89	1.73	0.001	8.38
136	316	0.40	0.99	2.47	0.016	15.39
137	85	0.25	0.20	0.79	0.009	1.27
138	27	0.24	0.14	0.59	0.029	0.56

Domain	Count	Avg Au g/t	Std Dev	CoV	Minimum Au g/t	Maximum Au g/t
139	137	0.72	2.48	3.46	0.017	26.50
140	152	0.24	0.24	1.01	0.001	2.09
141	39	0.27	0.24	0.92	0.010	1.13
142	140	0.45	0.56	1.26	0.022	3.42
143	767	0.53	0.98	1.86	0.003	13.15
144	230	0.56	1.45	2.57	0.015	14.54
145	117	0.85	1.50	1.77	0.012	10.41
146	50	0.41	0.44	1.06	0.006	1.78
147	200	0.35	0.53	1.54	0.032	5.90
148	130	0.34	0.88	2.57	0.018	9.92
149	246	0.43	1.13	2.61	0.006	15.09
150	101	0.38	0.51	1.35	0.016	4.28
151	28	0.38	0.85	2.22	0.031	4.65
152	71	0.31	0.41	1.32	0.014	2.13
153	45	0.42	0.66	1.56	0.024	4.20
154	44	0.44	0.93	2.10	0.037	5.29
155	677	0.46	0.75	1.63	0.011	9.36
156	196	0.32	0.66	2.05	0.009	8.22
157	40	0.30	0.31	1.03	0.035	1.29
158	54	0.42	0.53	1.26	0.012	2.57
159	132	0.41	0.49	1.20	0.004	2.89
160	32	0.30	0.26	0.85	0.014	1.26
161	25	0.62	1.26	2.02	0.018	6.44
162	69	0.29	0.30	1.01	0.037	1.53
163	71	0.36	0.76	2.11	0.013	6.45
164	26	0.32	0.39	1.24	0.021	1.87
165	275	0.37	0.93	2.50	0.020	14.86
166	178	0.34	0.49	1.42	0.003	4.95
167	51	0.71	1.76	2.50	0.008	9.52
168	35	0.32	0.29	0.90	0.026	1.38
169	13	0.26	0.32	1.21	0.015	1.19
170	125	0.40	0.51	1.29	0.028	3.07
171	72	0.34	0.53	1.56	0.021	3.80
172	145	0.31	0.35	1.10	0.023	2.11
173	105	0.28	0.28	1.00	0.022	2.17

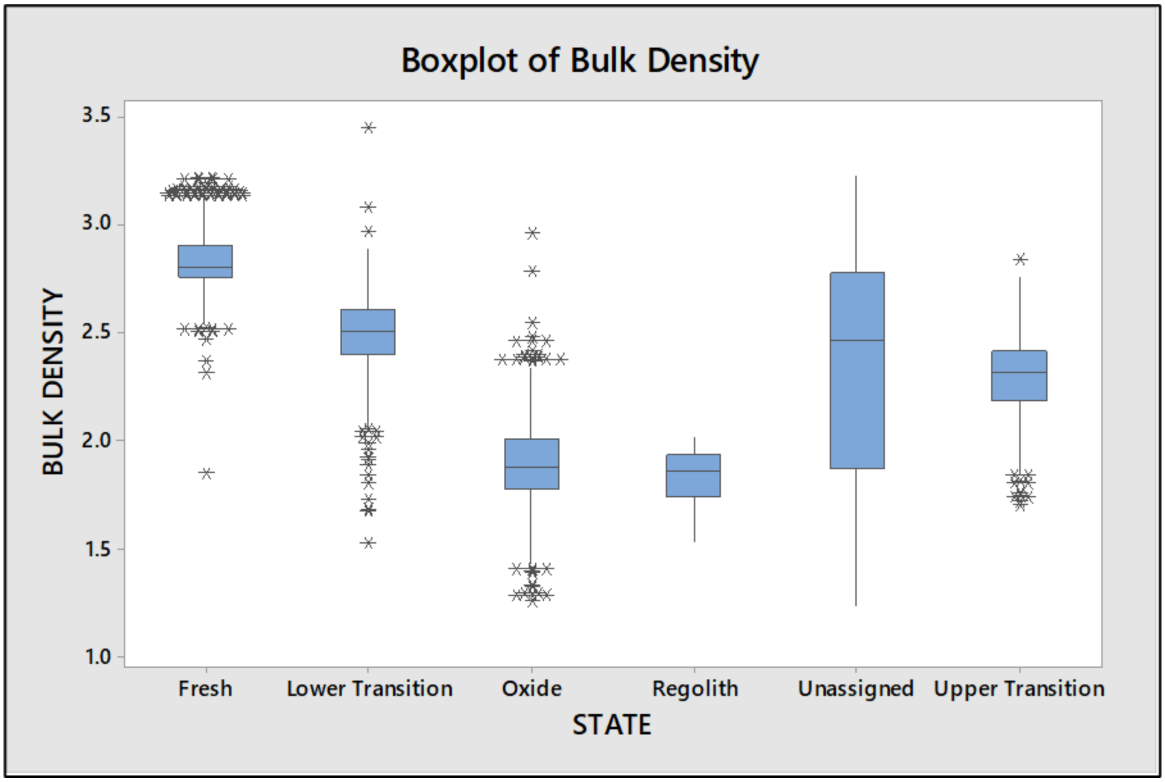
Domain	Count	Avg Au g/t	Std Dev	CoV	Minimum Au g/t	Maximum Au g/t
174	80	0.35	0.39	1.13	0.048	2.17
184	38	0.27	0.36	1.32	0.014	2.14
200	113	0.34	0.42	1.22	0.022	3.70
201	712	0.44	0.72	1.64	0.017	15.30
202	233	0.37	0.44	1.17	0.024	3.26
203	45	0.39	0.35	0.89	0.022	1.39
Total	**14,739**	**0.52**	**1.41**	**2.69**	**0.001**	**89.94**

The supplied database contains 10,145 point bulk density measurements from drill hole core, with values ranging from 1.23 to 3.44 t/m^3 (Table 14.5.3). Bulk density measurements were back-tagged by mineralization domain, lithology and oxidation state wireframes. Bulk density measurements display differing ranges of values based on oxidation state (Figure 14.5.3).

Table 14.5.3 P11 Zone Bulk Density Summary Statistics

Unit	Count	Avg. Bulk Density t/m3	Min. Bulk Density t/m3	Max. Bulk Density t/m3	Median Bulk Density t/m3	Std Dev
Unassigned	282	2.36	1.23	3.23	2.46	0.49
Regolith	26	1.82	1.52	2.01	1.86	0.13
Oxide	1,501	1.89	1.25	2.95	1.88	0.19
Upper Transition	639	2.29	1.70	2.83	2.31	0.18
Lower Transition	726	2.49	1.52	3.44	2.50	0.19
Fresh	6,971	2.83	1.84	3.21	2.80	0.10
Total	**10,145**	**2.61**	**1.23**	**3.44**	**2.76**	**0.38**

Figure 14.5.3 Boxplot of P11 Zone Bulk Density by Oxide State



14.5.3 Block Model

An orthogonal block model was established with the block model limits selected to cover the extent of the mineralized structures, the potential open pit dimensions, and to reflect the general orientation of the mineralized zones (Table 14.5.4). The block model consists of separate variables for estimated grades, rock codes, bulk density and classification attributes. Separate variables were coded with unique mineralization domain, lithology and oxide state rock codes.

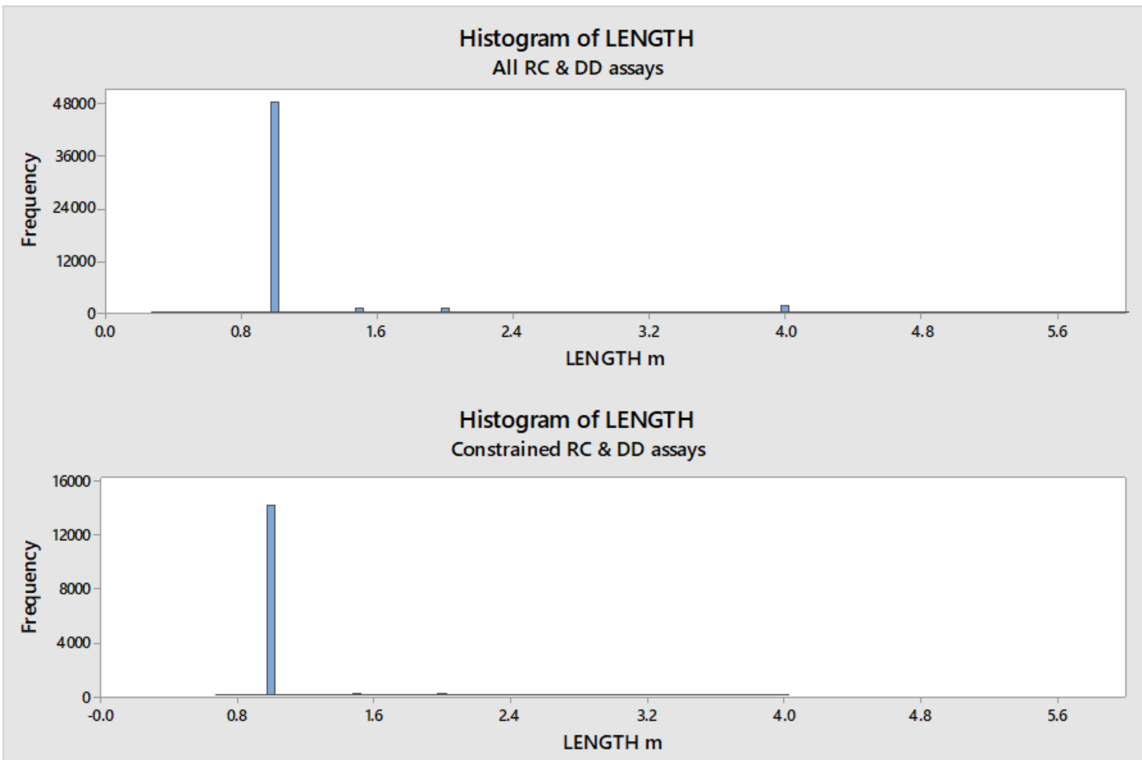
Table 14.5.4 P11 Zone Block Model Setup

Coordinate	Origin	Block Size m	Number of Blocks
Easting (X)	726,740	2.00	1,110
Northing (Y)	1,347,740	6.25	429
Elevation (max Z)	350	3.00	134

14.5.4 Compositing

Assay sample lengths range from 0.30 to 6.00 m, with an average sample length of 1.13 m (Figure 14.5.4). Constrained assay samples lengths within the defined mineralization domains range from 0.70 to 4.00 m, with an average sample length of 1.20 m and a median sample length of 1.00 m.

Figure 14.5.4 Histogram of P11 Zone Constrained Assay Sample Lengths



Based on the predominance of 1.00 m sample lengths, all constrained assay samples were composited to this length in order to ensure equal sample support. Length-weighted composites were calculated within the defined mineralization domains. Missing sample intervals in the data were assigned a nominal background grade of 0.001 g/t. The compositing process started at the first point of intersection between the drill hole and the domain wireframe intersected and halted on exit from the wireframe. Downhole residual composites that were less than half the compositing length were discarded so as to not introduce a short sample bias into the composite sample population. The wireframes that represent the interpreted mineralization domains were utilized to back-tag a rock code variable into the composite workspace. The composite data were then visually validated against the mineralization wireframes and extracted for analysis and grade estimation. Summary composite statistics are listed in Table 14.5.5.

Table 14.5.5 P11 Zone Composite Summary Statistics

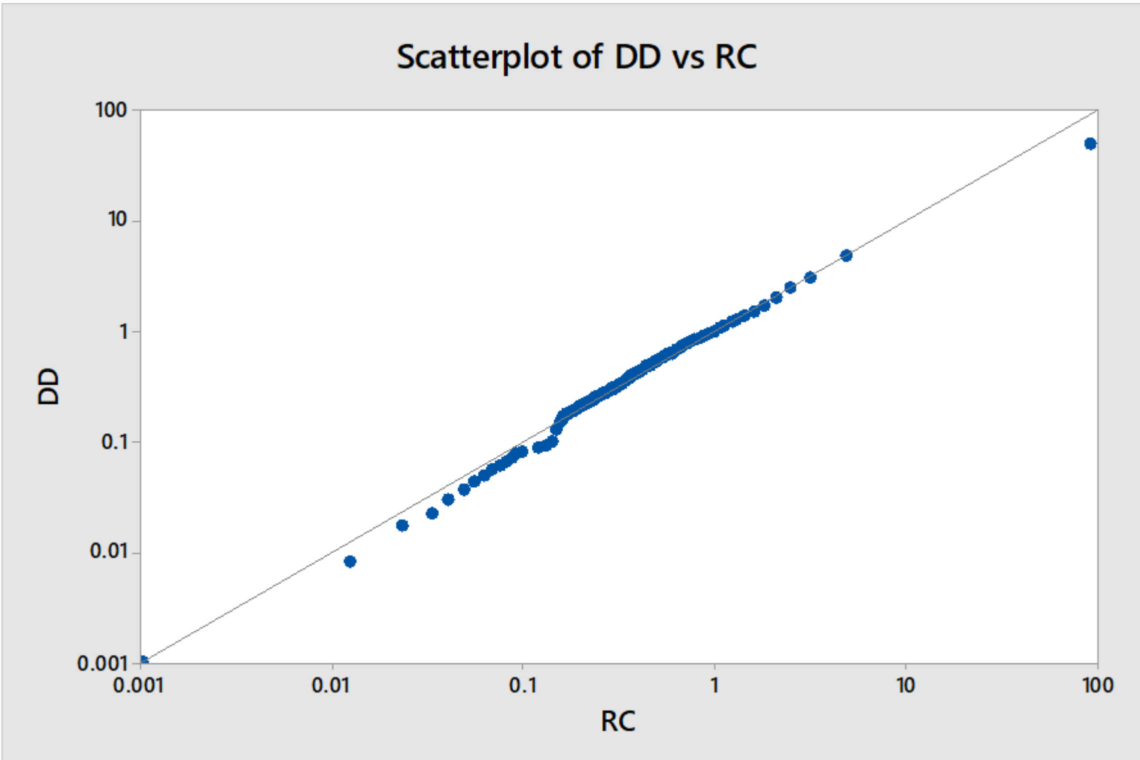
Domain	Count	Avg Au g/t	Std Dev	CoV	Minimum Au g/t	Maximum Au g/t
101	509	0.90	1.61	1.80	0.002	14.16
102	1,379	0.94	2.02	2.16	0.001	38.17
103	117	1.04	2.22	2.13	0.005	16.85
104	28	0.45	0.39	0.87	0.012	1.62
105	85	0.39	0.62	1.58	0.036	3.58
106	56	0.40	0.93	2.32	0.002	5.05
107	44	0.31	0.29	0.95	0.052	1.56
108	35	2.30	8.18	3.56	0.010	48.67
109	18	0.24	0.19	0.79	0.030	0.79
110	17	0.29	0.18	0.61	0.020	0.54
111	26	0.34	0.21	0.64	0.068	1.15
112	430	0.73	1.98	2.70	0.004	36.71
113	1,414	0.58	2.46	4.24	0.002	89.87
114	295	0.53	0.72	1.34	0.003	4.92
115	1,093	0.54	1.15	2.14	0.001	26.89
116	29	0.64	0.73	1.14	0.021	4.05
117	178	0.58	0.81	1.39	0.004	6.39
118	372	0.48	0.55	1.14	0.003	4.66
119	204	0.64	1.90	2.99	0.017	25.47
120	71	0.52	0.56	1.06	0.009	2.81
121	17	0.66	0.57	0.88	0.155	2.30
122	41	1.02	1.88	1.84	0.109	12.22
123	15	0.50	0.40	0.79	0.025	1.12
124	12	0.46	0.35	0.76	0.158	1.12
125	389	0.48	0.58	1.21	0.002	5.75
126	14	0.34	0.37	1.10	0.008	1.52
127	6	0.56	0.38	0.68	0.234	1.26
128	24	0.60	0.62	1.03	0.153	2.56
129	20	0.24	0.25	1.06	0.003	0.99
130	153	0.39	0.42	1.07	0.002	3.00
131	39	0.40	0.36	0.89	0.002	1.94
132	27	0.47	0.46	0.98	0.019	1.85
133	158	0.37	0.32	0.86	0.010	2.56

Domain	Count	Avg Au g/t	Std Dev	CoV	Minimum Au g/t	Maximum Au g/t
134	186	0.90	0.92	1.02	0.001	5.89
135	168	0.54	0.89	1.65	0.001	8.38
136	284	0.46	0.93	2.01	0.016	10.08
137	76	0.28	0.21	0.76	0.001	1.27
138	24	0.26	0.11	0.43	0.084	0.54
139	124	0.79	2.59	3.28	0.020	26.50
140	135	0.26	0.24	0.93	0.028	2.09
141	32	0.32	0.24	0.76	0.010	1.13
142	126	0.49	0.57	1.17	0.039	3.42
143	730	0.55	0.98	1.79	0.003	13.12
144	209	0.61	1.51	2.47	0.024	14.49
145	101	0.97	1.58	1.63	0.018	10.41
146	44	0.46	0.44	0.97	0.007	1.78
147	189	0.36	0.54	1.51	0.032	5.89
148	119	0.36	0.91	2.51	0.018	9.91
149	221	0.47	1.18	2.49	0.020	15.09
150	93	0.40	0.53	1.32	0.019	4.28
151	21	0.49	0.97	1.97	0.047	4.65
152	61	0.35	0.43	1.21	0.018	2.13
153	36	0.51	0.71	1.40	0.024	4.19
154	39	0.49	0.98	1.99	0.042	5.28
155	627	0.49	0.77	1.56	0.001	9.35
156	167	0.37	0.71	1.91	0.038	8.22
157	35	0.38	0.33	0.88	0.061	1.29
158	45	0.49	0.55	1.12	0.050	2.57
159	124	0.43	0.50	1.16	0.001	2.87
160	27	0.35	0.26	0.76	0.014	1.25
161	19	0.81	1.40	1.74	0.020	6.42
162	60	0.33	0.30	0.93	0.041	1.53
163	57	0.43	0.84	1.93	0.038	6.45
164	22	0.37	0.40	1.10	0.031	1.87
165	258	0.39	0.96	2.43	0.034	14.86
166	163	0.37	0.50	1.33	0.003	4.93
167	45	0.79	1.86	2.34	0.053	9.51
168	28	0.38	0.29	0.76	0.034	1.38

Domain	Count	Avg Au g/t	Std Dev	CoV	Minimum Au g/t	Maximum Au g/t
169	9	0.37	0.33	0.89	0.082	1.19
170	110	0.45	0.53	1.19	0.034	3.07
171	65	0.33	0.40	1.22	0.059	2.33
172	129	0.34	0.35	1.03	0.040	2.11
173	95	0.31	0.29	0.94	0.051	2.17
174	72	0.38	0.40	1.07	0.076	2.16
184	35	0.31	0.36	1.15	0.034	2.13
200	101	0.38	0.43	1.15	0.022	3.70
201	697	0.45	0.72	1.60	0.017	15.30
202	222	0.39	0.42	1.08	0.027	3.25
203	40	0.44	0.34	0.76	0.054	1.39
Total	**13,585**	**0.57**	**1.45**	**2.53**	**0.001**	**89.87**

Examination of the RC and DD composite grades shows no significant difference nor a bias between the drill sample populations (Figure 14.5.5).

Figure 14.5.5 QQ Plot of P11 Zone RC Composite Grades Versus DD Composite Grades



14.5.5 Treatment of Extreme Values

Capping thresholds were determined by the decomposition of the individual composite log-probability distributions, and examination of box-plots and histograms. Potential outliers are not markedly clustered in localized high-grade areas and sub-domaining is therefore not warranted. Composites are capped to the defined threshold prior to grade estimation. A total of 25 domains do not require capping (Table 14.5.6).

Table 14.5.6 P11 Zone Capping Thresholds and Summary Statistics

Domain	Count	Cap	Number Capped	Mean	Capped Mean	Change %
101	509	7	7	0.90	0.84	-6
102	1,379	14	5	0.94	0.91	-3
103	117	NA	0	1.04	1.04	0
104	28	1	3	0.45	0.42	-7
105	85	3	2	0.39	0.38	-3
106	56	1	3	0.40	0.26	-36

Domain	Count	Cap	Number Capped	Mean	Capped Mean	Change %
107	44	1	2	0.31	0.29	-5
108	35	3	3	2.30	0.84	-63
109	18	NA	0	0.24	0.24	0
110	17	NA	0	0.29	0.29	0
111	26	NA	0	0.34	0.34	0
112	430	5	4	0.73	0.64	-12
113	1,414	5	6	0.58	0.52	-11
114	295	3	6	0.53	0.51	-4
115	1,093	4	8	0.54	0.49	-9
116	29	NA	0	0.64	0.64	0
117	178	3	5	0.58	0.55	-5
118	372	2.5	6	0.48	0.47	-2
119	204	2	3	0.63	0.47	-25
120	71	NA	0	0.52	0.52	0
121	17	NA	0	0.66	0.66	0
122	41	2	2	1.02	0.75	-27
123	15	NA	0	0.50	0.50	0
124	12	NA	0	0.46	0.46	0
125	389	2	11	0.48	0.45	-5
126	14	NA	0	0.34	0.34	0
127	6	NA	0	0.56	0.56	0
128	24	NA	0	0.60	0.60	0
129	20	NA	0	0.24	0.24	0
130	153	1.4	6	0.39	0.37	-5
131	39	NA	0	0.40	0.40	0
132	27	1	3	0.47	0.41	-13
133	158	2	1	0.37	0.37	-1
134	186	3	5	0.90	0.87	-4
135	168	4	2	0.54	0.51	-5
136	284	4	6	0.46	0.42	-9
137	76	NA	0	0.28	0.28	0
138	24	NA	0	0.26	0.26	0
139	124	2	3	0.79	0.50	-37
140	135	NA	0	0.26	0.26	0
141	32	NA	0	0.32	0.32	0

Domain	Count	Cap	Number Capped	Mean	Capped Mean	Change %
142	126	3	2	0.49	0.49	-1
143	730	7	3	0.55	0.53	-4
144	209	5	3	0.61	0.51	-16
145	101	NA	0	0.97	0.97	0
146	44	1	5	0.46	0.40	-12
147	189	NA	0	0.36	0.36	0
148	119	1	3	0.36	0.28	-22
149	221	3	2	0.47	0.40	-17
150	93	2	1	0.40	0.37	-6
151	21	1	1	0.49	0.32	-35
152	61	1	4	0.35	0.30	-15
153	36	2	1	0.51	0.45	-12
154	39	1	3	0.49	0.31	-37
155	627	5	4	0.49	0.48	-2
156	167	2	2	0.37	0.32	-13
157	35	1	3	0.38	0.36	-5
158	45	2	2	0.49	0.47	-4
159	124	1	7	0.43	0.37	-14
160	27	NA	0	0.35	0.35	0
161	19	1	2	0.80	0.51	-37
162	60	1	4	0.33	0.31	-5
163	57	1	1	0.43	0.34	-22
164	22	NA	0	0.37	0.37	0
165	258	2	2	0.39	0.34	-13
166	163	2	3	0.37	0.36	-5
167	45	3	2	0.79	0.52	-34
168	28	NA	0	0.38	0.38	0
169	9	NA	0	0.37	0.37	0
170	110	1	10	0.45	0.36	-19
171	65	2	2	0.33	0.32	-2
172	129	1	9	0.34	0.31	-8
173	95	NA	0	0.31	0.31	0
174	72	1.5	4	0.38	0.36	-4
184	35	1	1	0.31	0.28	-10
200	101	2	1	0.38	0.36	-4

Domain	Count	Cap	Number Capped	Mean	Capped Mean	Change %
201	697	6	1	0.45	0.44	-3
202	222	2	4	0.39	0.38	-3
203	40	1	4	0.44	0.42	-4

14.5.6 Variography and Continuity Analysis

Three-dimensional continuity analyses (variography) were conducted on the domain-coded uncapped composite data using normal-score and 50% indicator semi-variograms. The downhole variograms were viewed at a 1.00 m lag spacing (equivalent to the composite length) to assess the nugget variance contribution.

The Authors were unable to derive acceptable semi-variograms for the individual mineralization domains, which tended towards pure-nugget effect semi-variograms or very short range semi-variograms, reflecting the relatively small number of assays and the homogeneity of mineralization within the constrained mineralization domains. For estimation purposes a best-fit isotropic semi-variogram was constructed from the combined composites (Equation 1).

$$0.60 + 0.24 \times SPH(3.0\ m) + 0.16 \times SPH(20\ m) \qquad (1)$$

14.5.7 Mineral Resource Estimate

Bulk density was estimated by Inverse Distance Squared (ID2) anisotropic linear interpolation using a minimum of three and maximum of twelve composites within a 300 m diameter search envelope. Sample selection was restricted to a maximum of four composite samples from a single drill hole within the same oxide state domain. Bulk density estimation was performed by weathering unit within the weathered portion of the deposits, and by lithological domain within the fresh Birimian portion of the deposits. These weathering and lithological units define hard boundaries for bulk density estimation. For those areas of limited sample data, the mean value of the specific lithology type in the Fresh domain or within the weathering unit above the Fresh unit was applied. Orezone applies a reduction factor of 5.5% and 2.5% to individual samples in the oxide and transition zones, respectively, for the block model dry bulk density calculations.

Bulk density block values were adjusted by these factors after estimation.

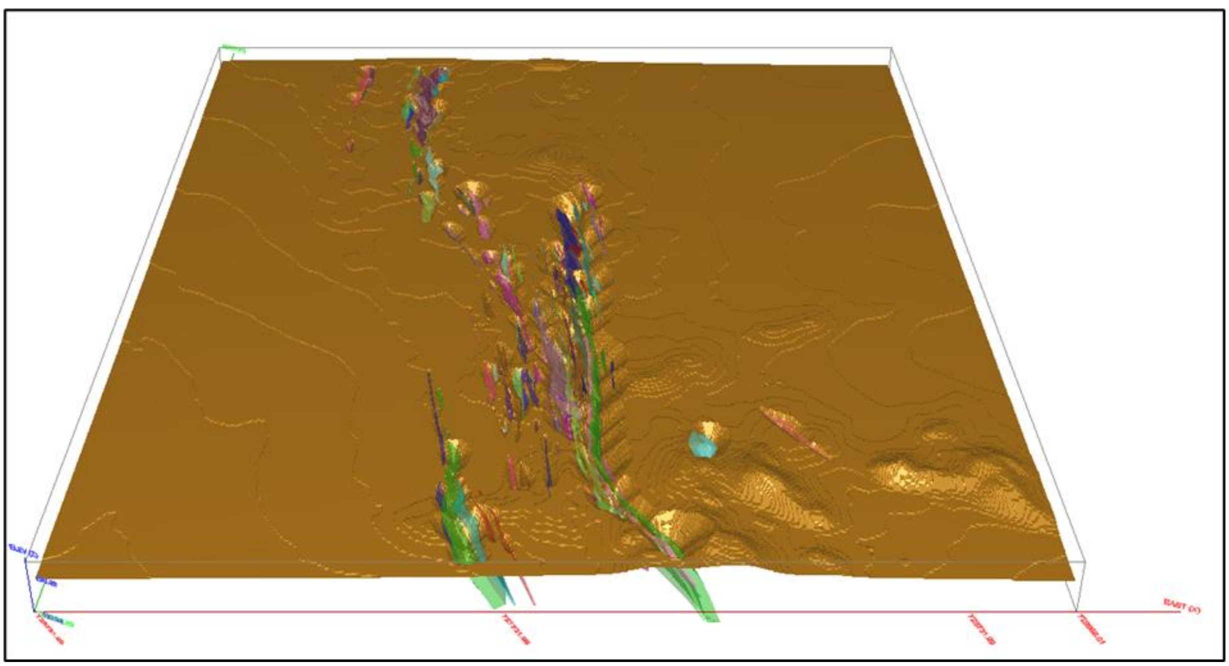
Block grades for gold were estimated by Inverse Distance Cubed (ID3) weighting of capped composites using a minimum of three and maximum of twelve composites within an extended 600 m x 300 m x 50 m search envelope. Sample selection selected the nearest composites to the block centroid and was restricted to a maximum of four composite samples from a single drill hole.

The orientations of the search ellipsoids were defined by Orezone geologists based on the local geology within each respective mineralization domain. The defined orientations were then converted to Gemcom™ format rotations. Ordinary Kriging (OK) and capped Nearest Neighbour models (NN) were also generated using the same search parameters. Only composites derived from DD and RC drill holes were used for grade estimation. Search and grade estimation were constrained by the individual mineralization domains, which define hard boundaries for grade estimation.

The parameters used to define classification limits included spatial analysis, drill hole spacing, and the observed continuity of the mineralization. Mineral Resources were classified algorithmically based on the local drill hole spacing within each mineralization domain. All blocks within 25 m of three or more drill holes were classified as Measured, blocks within 50 m of three or more drill holes were classified as Indicated, and all additional estimated blocks were classified as Inferred.

Mineral Resources reported herein have been constrained within an optimized pit (Figure 14.5.6). The results from the optimized pit shell are utilized solely for the purpose of reporting mineral resources and include Measured, Indicated and Inferred Mineral Resources. Mineral Resources are reported at a cut-off grade of 0.25 g/t Au for regolith, oxide and upper transition lithologies, and 0.45 g/t Au for lower transition and fresh lithologies (Table 14.5.7). The Authors are reporting whole block volumes for only those blocks where the volume inclusion is ≥50%. Orezone applied a 0.85 factor to the P11 Zone oxide grade to account for artisanal mining.

Figure 14.5.6 Isometric View of P1 Zone Optimized Pit-Shell



View looking north

Table 14.5.7 P11 Zone Mineral Resource Estimate*

Total	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	NA	3,125	0.79	79
Indicated	NA	2,717	0.77	68
Meas + Ind	NA	5,842	0.78	147
Inferred	NA	160	0.47	2

Regolith	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.25	0	0.00	0
Indicated	0.25	0	0.31	0
Meas + Ind	0.25	0	0.31	0
Inferred	0.25	0	0.00	0

Oxide	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.25	1,402	0.61	28
Indicated	0.25	1,161	0.47	18
Meas + Ind	0.25	2,563	0.55	45
Inferred	0.25	137	0.39	2

Trans Upper	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.25	564	0.80	15
Indicated	0.25	341	0.59	7
Meas + Ind	0.25	905	0.72	21
Inferred	0.25	17	0.41	0

Trans Lower	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.45	406	1.06	14
Indicated	0.45	146	0.93	4
Meas + Ind	0.45	552	1.02	18
Inferred	0.45	1	1.37	0

Fresh	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.45	754	0.95	23
Indicated	0.45	1,069	1.14	39
Meas + Ind	0.45	1,823	1.06	62
Inferred	0.45	5	2.19	0

*Mineral Resources are inclusive of Reserves. Totals may differ due to rounding.

14.5.8 Validation

The P11 Zone block model was validated visually by the inspection of successive vertical cross-section lines, in order to confirm that the block models correctly reflect the distribution of high-grade and low-grade values.

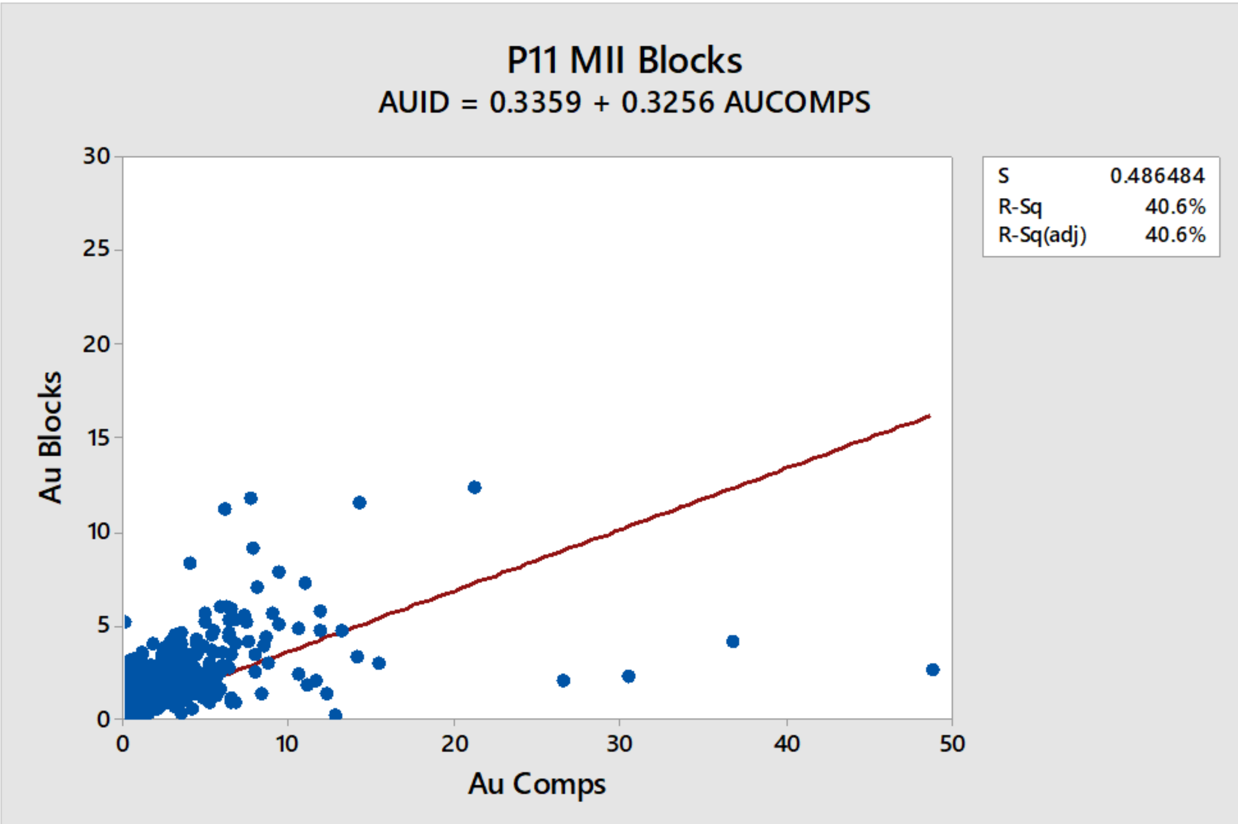
The average estimated block grades were compared to the average NN block estimate at a 0.001 g/t cut-off grouped by oxidation state domain (Table 14.5.8). The results fall within acceptable limits for grade estimation.

Table 14.5.8 Comparison of P11 Zone OK, ID3 and NN Average Block Grades

Unit	Au OK g/t	Au ID g/t	Au NN g/t	Ratio ID/NN %
Oxide / Saprolite	0.447	0.457	0.462	97
Transition Upper	0.480	0.488	0.493	97
Transition Lower	0.487	0.490	0.496	98
Fresh	0.486	0.491	0.499	97
Regolith	0.476	0.492	0.485	98
Total	0.478	0.491	0.487	98

An additional validation check was completed by comparing the average grade of the composites in a block to the associated model block grade estimate (Figure 14.5.7).

Figure 14.5.7 Validation Plot Between P11 Zone Block Grades and Average Composite Grades



The volume estimated was also checked against the reported volume of the individual mineralization domains and the estimated volumes for both the Whole Block Centroid model and Partial Block model (Table 14.5.9). The results fall within acceptable limits for linear estimation.

Table 14.5.9 P11 Zone Volume Reconciliation

Domain	Volume k m^3	Estimated Whole Block Volume k m^3	Estimated Partial Block Volume k m^3
101	1,094	1,090	1,093
102	2,263	2,234	2,257
103	138	129	137
104	41	33	41
105	318	317	316
106	81	80	80

Domain	Volume k m^3	Estimated Whole Block Volume k m^3	Estimated Partial Block Volume k m^3
107	139	137	138
108	24	21	24
109	3	3	4
110	4	3	4
111	43	42	43
112	473	456	469
113	1,568	1,564	1,565
114	154	148	153
115	1,659	1,653	1,657
116	9	7	9
117	458	450	457
118	779	767	776
119	187	182	187
120	291	287	291
121	57	53	57
122	92	80	92
123	18	13	18
124	39	21	39
125	373	358	371
126	5	4	5
127	45	41	45
128	23	15	23
129	6	6	6
130	272	261	270
131	129	126	129
132	19	16	19
133	217	200	216
134	150	144	150
135	130	122	130
136	412	406	411
137	62	57	62
138	28	22	28
139	104	100	104
140	108	103	107
141	114	110	114

Domain	Volume k m^3	Estimated Whole Block Volume k m^3	Estimated Partial Block Volume k m^3
142	152	150	152
143	691	691	690
144	180	176	177
145	388	377	389
146	36	35	36
147	324	322	324
148	175	174	175
149	647	645	647
150	85	81	85
151	51	46	51
152	172	151	172
153	13	12	13
154	65	63	64
155	882	869	879
156	253	245	250
157	82	80	82
158	41	38	41
159	107	106	106
160	18	17	18
161	12	9	12
162	105	101	105
163	302	291	301
164	4	4	4
165	961	951	961
166	375	369	375
167	76	71	76
168	23	21	23
169	35	30	35
170	230	225	230
171	65	62	65
172	570	565	570
173	174	167	173
174	158	152	159
184	192	189	192
200	234	230	234

Domain	Volume k m³	Estimated Whole Block Volume k m³	Estimated Partial Block Volume k m³
201	2,630	2,615	2,626
202	319	315	317
203	37	35	37
Total	**22,994**	**22,536**	**22,938**

14.5.9 P11 Cut-Off Sensitivity

The sensitivity of the Mineral Resource block grades to changes in cut-off grade was evaluated by calculating the total Mineral Resource inventory at various cut-offs within the Mineral Resource pit (Table 14.5.10). The values presented are intended for the sole purpose of demonstrating the sensitivity of the Mineral Resource with respect to the cut-off grade.

Table 14.5.10 P11 Grade Tonnage Sensitivity Table

OXIDE MEASURED + INDICATED				OXIDE INFERRED		
Au Cut-Off	**Tonnes**	**Au**		**Au Cut-Off**	**Tonnes**	**Au**
g/t	**k**	**g/t**		**g/t**	**k**	**g/t**
1.00	536.4	1.44		1.00	1.6	1.13
0.95	592.1	1.38		0.95	1.8	1.11
0.90	654.2	1.33		0.90	1.8	1.11
0.85	731.1	1.27		0.85	2.1	1.05
0.80	816.0	1.22		0.80	3.3	0.92
0.75	906.1	1.16		0.75	5.6	0.82
0.70	1,015.0	1.11		0.70	9.0	0.74
0.65	1,139.3	1.05		0.65	15.2	0.67
0.60	1,290.0	1.00		0.60	26.2	0.62
0.55	1,472.3	0.94		0.55	29.0	0.60
0.50	1,707.4	0.87		0.50	34.9	0.58
0.45	1,975.0	0.81		0.45	46.2	0.53
0.40	2,289.1	0.75		0.40	97.9	0.45
0.35	2,662.5	0.69		0.35	111.0	0.43
0.30	3,085.8	0.64		0.30	145.9	0.40
0.25	3,467.5	0.59		0.25	153.5	0.39
0.20	3,799.7	0.56		0.20	166.0	0.37

SULPHIDE MEASURED + INDICATED				SULPHIDE INFERRED		
Au Cut-Off	Tonnes	Au		Au Cut-Off	Tonnes	Au
g/t	k	g/t		g/t	k	g/t
1.00	887.0	1.68		1.00	5.6	2.67
0.95	970.8	1.62		0.95	5.7	2.64
0.90	1,056.8	1.56		0.90	5.9	2.59
0.85	1,151.6	1.50		0.85	5.9	2.59
0.80	1,261.5	1.44		0.80	6.0	2.57
0.75	1,385.9	1.38		0.75	6.0	2.57
0.70	1,513.3	1.33		0.70	6.0	2.57
0.65	1,657.6	1.27		0.65	6.1	2.54
0.60	1,820.0	1.21		0.60	6.1	2.54
0.55	1,988.7	1.16		0.55	6.1	2.54
0.50	2,167.1	1.11		0.50	6.1	2.54
0.45	2,374.5	1.05		0.45	6.2	2.51
0.40	2,584.6	1.00		0.40	6.4	2.45
0.35	2,793.0	0.95		0.35	6.4	2.42
0.30	3,014.6	0.91		0.30	6.4	2.42
0.25	3,238.0	0.86		0.25	6.4	2.42
0.20	3,404.7	0.83		0.20	6.4	2.42

Oxide includes Regolith, Oxide and Transitional Upper units.
Sulphide includes Transitional Lower and Fresh units.
Totals may differ due to rounding.

14.6 Bomboré Siga Zone

14.6.1 Data Supplied

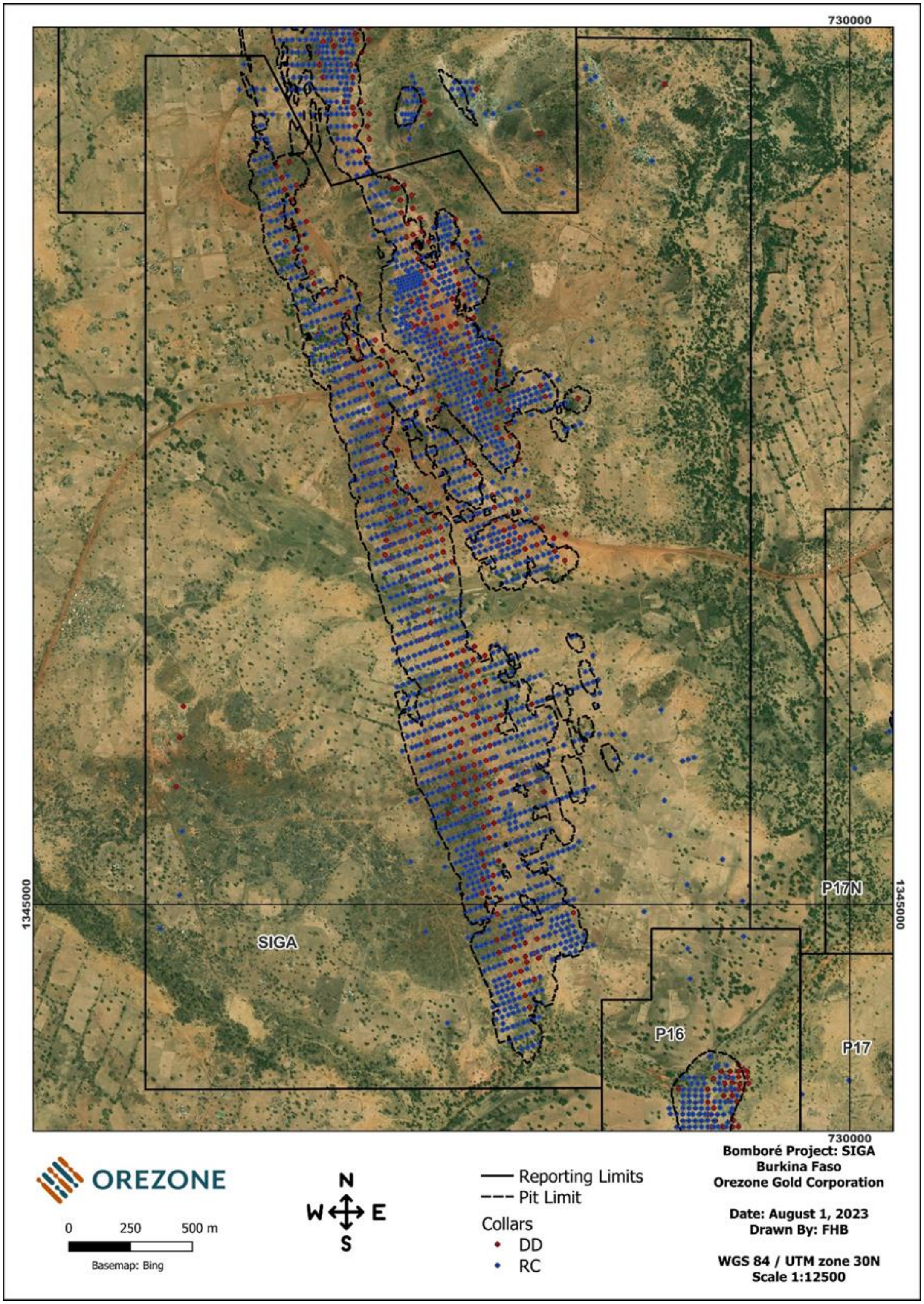
The Siga Zone model was developed by the Authors. Topography, mineralization, lithology and oxidation state three-dimensional wireframes were created by Orezone using Leapfrog Geo™ 2022.1.1. Drill hole distances are reported in metres and gold grades are reported as g/t. The collar coordinates were provided in the WGS84 Zone 30N UTM coordinate reference system.

The supplied drill hole database for the Siga Zone contains 3,276 unique drill hole collar records (Table 14.6.1 and Figure 14.6.1). Only reverse circulation (RC) holes and diamond drill (DD) drill holes were used for the Mineral Resource Estimate.

Table 14.6.1 Siga Zone Drill Hole Database Summary

Type	Count	Metres
Pressure Meter Drill Hole	1	20.50
Channel	10	13.90
Diamond Core Drill Hole (DD)	438	67,927.60
Auger Drill Hole	632	3,742.00
Rotary Air Blast Drill Hole (RAB)	278	8,189.00
Water Borehole	9	631.96
Reverse Circulation Drill Hole (RC)	1,682	90,702.00
Trench	126	1,285.60
Total	**3,276**	**172,512.56**

Figure 14.6.1 Siga Zone Drill Hole Collar Plot

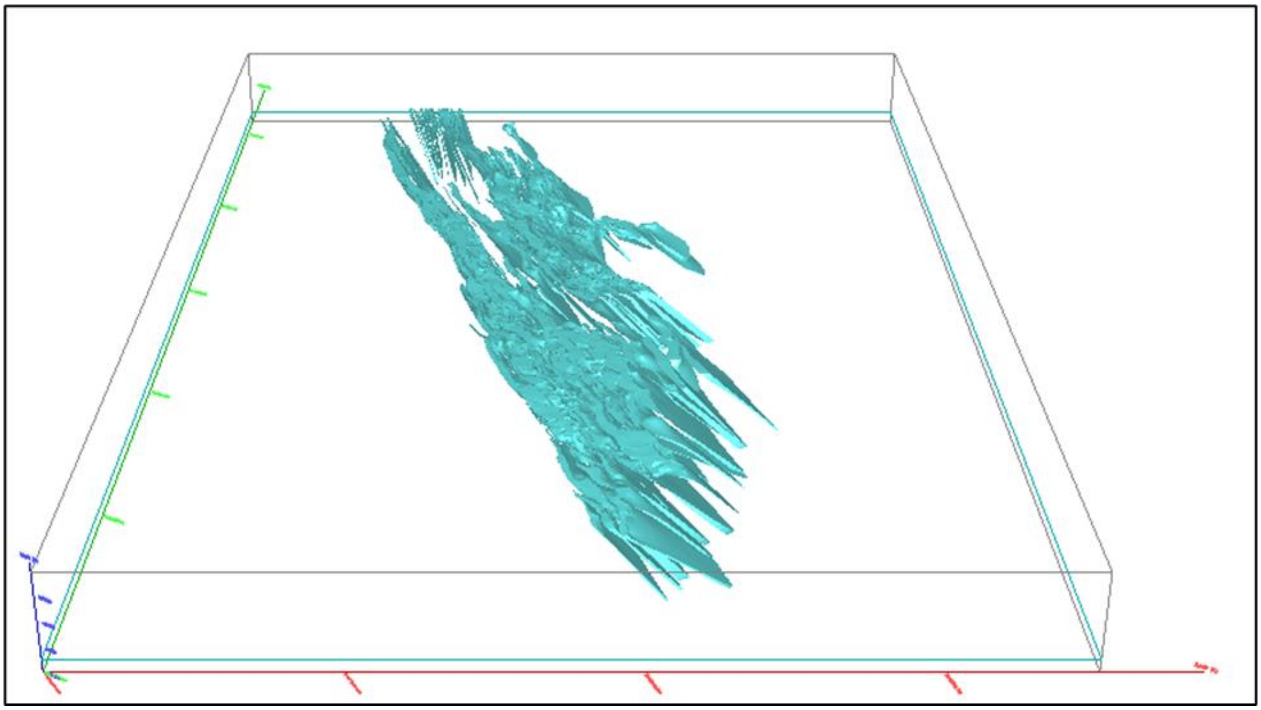


14.6.2 Exploratory Data Analysis

A total of 73 distinct mineralization domain wireframes were supplied by Orezone (Figure 14.6.2). Each domain was assigned a unique three-digit rock code.

The mean nearest-neighbour drill hole collar distance for DD and RC combined is 22.4 m, the average DD length is 155.1 m and the average RC length is 53.9 m. Summary statistics for the supplied assay data are provided below (Table 14.6.2).

Figure 14.6.2 Siga Zone Isometric Plot of Mineralization Domains



View looking north. Field of view is approximately 2 km across.

Table 14.6.2 Siga Zone Assay Summary Statistics

Domain	Count	Avg Au g/t	Std Dev	CoV	Minimum Au g/t	Maximum Au g/t
101	193	0.50	2.27	4.59	0.008	28.93
102	504	0.48	1.38	2.89	0.01	26.96
103	2,089	0.50	1.99	4.01	0.007	79.71
104	949	0.33	0.38	1.14	0.005	5.93
105	355	0.37	0.53	1.42	0.009	4.89
106	5,172	1.08	24.86	22.98	0.001	1,783.87

Domain	Count	Avg Au g/t	Std Dev	CoV	Minimum Au g/t	Maximum Au g/t
107	931	0.37	0.98	2.62	0.001	19.81
108	4,087	0.38	0.89	2.32	0.0005	44.51
109	5,330	0.41	0.82	2.00	0.0005	42.59
110	5,616	0.44	0.81	1.84	0.0005	32.40
111	5,500	0.46	0.67	1.46	0.0005	16.98
112	1,653	0.39	1.18	3.00	0.0005	22.47
113	5,090	0.47	0.98	2.09	0.0005	31.62
114	4,332	0.39	0.67	1.71	0.002	21.44
115	280	0.47	1.05	2.24	0.005	10.16
116	65	0.36	0.32	0.90	0.01	1.39
117	67	0.31	0.46	1.48	0.01	2.59
118	1,406	0.33	0.92	2.77	0.01	26.84
120	517	0.38	0.84	2.22	0.02	12.72
121	169	0.28	0.33	1.20	0.02	3.92
122	1,598	0.30	0.64	2.10	0.008	15.59
123	1,398	0.30	0.43	1.40	0.01	8.38
124	111	0.28	0.20	0.72	0.04	1.17
125	317	0.31	0.31	1.01	0.0005	2.89
126	315	0.31	0.39	1.30	0.04	5.38
127	174	0.28	0.23	0.82	0.01	1.39
128	224	0.24	0.22	0.92	0.01	2.02
129	180	0.37	0.66	1.78	0.04	5.56
130	91	0.35	0.55	1.56	0.03	3.72
131	79	0.44	1.60	3.61	0.001	14.19
132	560	0.35	1.24	3.54	0.009	28.00
133	572	0.34	0.41	1.22	0.003	4.10
134	1,436	0.36	0.75	2.08	0.002	16.15
135	1,222	0.46	2.66	5.80	0.003	73.95
136	656	0.30	0.37	1.25	0.0005	4.04
137	628	0.39	0.98	2.48	0.001	15.49
138	581	0.38	0.65	1.71	0.003	9.46
141	497	0.36	0.54	1.49	0.001	7.43
142	5,321	0.63	1.61	2.56	0.0005	67.04
143	3,633	0.63	1.83	2.91	0.0005	54.72
144	159	0.37	0.58	1.57	0.001	5.04

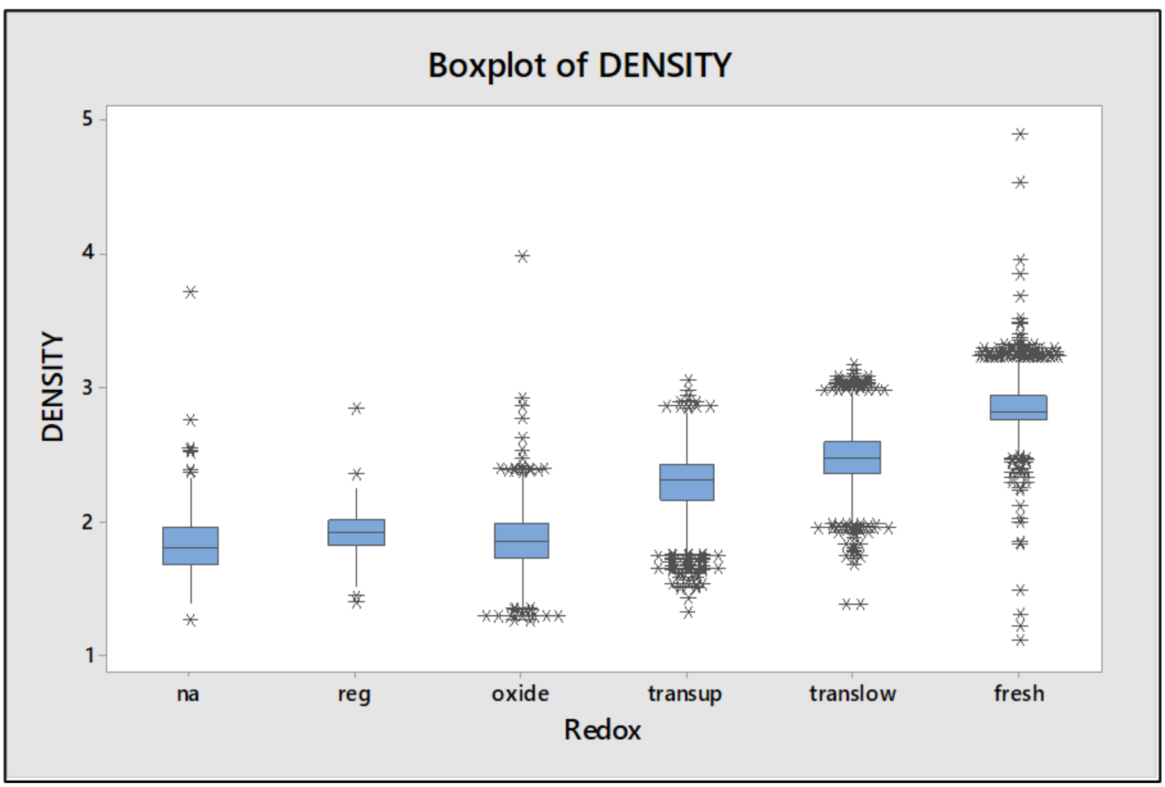
Domain	Count	Avg Au g/t	Std Dev	CoV	Minimum Au g/t	Maximum Au g/t
145	386	0.45	0.99	2.20	0.001	12.43
146	125	0.31	0.36	1.14	0.006	1.61
147	431	0.51	0.97	1.92	0.0005	11.44
148	254	0.72	1.35	1.87	0.006	11.27
149	1,356	0.52	0.97	1.88	0.0005	13.02
150	741	1.00	2.45	2.46	0.001	32.28
151	79	0.30	0.29	0.96	0.003	1.79
152	129	0.62	2.31	3.73	0.003	23.75
153	155	0.49	0.76	1.55	0.003	7.10
154	117	0.39	0.63	1.59	0.001	4.15
155	216	0.76	0.92	1.21	0.001	5.90
156	88	0.33	0.43	1.30	0.005	3.22
157	304	0.40	0.55	1.38	0.002	4.41
158	484	0.48	0.72	1.50	0.0005	10.84
159	114	0.32	0.32	1.02	0.0005	1.83
160	129	0.31	0.35	1.14	0.02	2.11
161	48	0.32	0.31	0.97	0.04	1.64
162	105	0.29	0.34	1.14	0.02	2.16
163	143	0.32	0.34	1.06	0.03	2.81
164	79	0.46	0.61	1.32	0.02	3.07
165	64	0.36	0.56	1.55	0.0005	3.79
166	23	0.27	0.26	0.99	0.006	1.19
167	65	0.35	0.38	1.08	0.02	2.01
168	39	0.79	2.00	2.53	0.008	9.51
169	78	0.31	0.42	1.34	0.01	3.10
170	95	0.35	0.61	1.74	0.002	5.29
171	72	0.39	0.62	1.58	0.001	4.10
172	691	0.45	0.63	1.38	0.001	8.80
173	200	0.39	0.51	1.32	0.002	4.01
174	456	0.41	0.56	1.36	0.002	5.75
175	300	0.47	0.57	1.22	0.0005	5.34
176	403	0.36	0.74	2.03	0.004	12.24
Total	**72,026**	**0.49**	**6.75**	**13.68**	**0.0005**	**1,783.87**

The supplied database contains 37,424 point bulk density measurements from drill hole core, with values ranging from 1.11 to 4.88 t/m^3 (Table 14.6.3). Bulk density measurements were back-tagged by mineralization domain, lithology and oxidation state wireframes. Bulk density measurements display differing ranges of values based on oxidation state (Figure 14.6.3).

Table 14.6.3 Siga Zone Bulk Density Summary Statistics

Unit	Count	Avg. Bulk Density t/m^3	Min. Bulk Density t/m^3	Max. Bulk Density t/m^3	Median Bulk Density t/m^3	Std Dev
Unassigned	336	1.83	1.25	3.71	1.80	0.23
Regolith	226	1.90	1.39	2.84	1.92	0.17
Oxide	3,081	1.85	1.25	3.97	1.85	0.20
Upper Transition	2,015	2.29	1.31	3.04	2.31	0.23
Lower Transition	2,400	2.47	1.38	3.17	2.47	0.20
Fresh	29,366	2.85	1.11	4.88	2.81	0.13
Total	**37,424**	**2.70**	**1.11**	**4.88**	**2.78**	**0.35**

Figure 14.6.3 Boxplot of Siga Zone Bulk Density by Oxide State



14.6.3 Block Model

A rotated block model was established with the block model limits selected to cover the extent of the mineralized structures, the potential open pit dimensions, and to reflect the general orientation of the mineralized zones (Table 14.6.4). The block model consists of separate variables for estimated grades, rock codes, bulk density and classification attributes. Separate variables were coded with unique mineralization domain, lithology and oxide state rock codes.

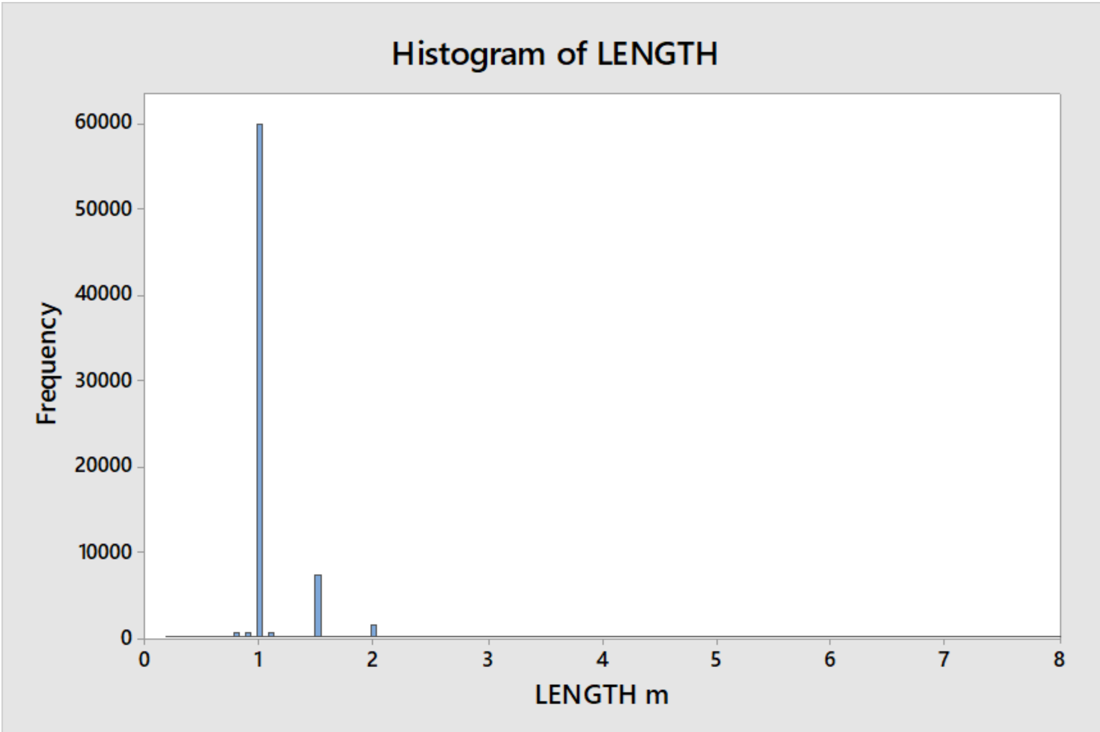
Table 14.6.4 Siga Zone Block Model Setup

Coordinate	Origin	Block Size m	Number of Blocks
Easting (X)	728,400	2.00	1,000
Northing (Y)	1,344,000	6.25	770
Elevation (max Z)	130	3.00	130
Rotation	20° anti-clockwise		

14.6.4 Compositing

Assay sample lengths range from 0.10 to 28.9 m, with an average of 1.12 m. Constrained assay samples lengths within the defined mineralization domains range from 0.20 to 8.20 m, with an average sample length of 1.08 m and a median sample length of 1.00 m (Figure 14.6.4).

Figure 14.6.4 Histogram of Suga Zone Constrained Assay Sample Lengths



Based on the predominance of 1.00 m sample lengths, all constrained assay samples were composited to this length in order to ensure equal sample support. Length-weighted composites were calculated within the defined mineralization domains. Missing sample intervals in the data were assigned a nominal background grade of 0.001 g/t. The compositing process started at the first point of intersection between the drill hole and the domain wireframe intersected and halted on exit from the wireframe. Down hole residual composites that were less than half the compositing length were discarded so as to not introduce a short sample bias into the composite sample population. The wireframes that represent the interpreted mineralization domains were utilized to back-tag a rock code variable into the composite workspace. The composite data were then visually validated against the mineralization wireframes and extracted for analysis and grade estimation. Summary composite statistics are listed in Table 14.6.5.

Table 14.6.5 Siga Zone Summary Composite Statistics

Domain	Count	Avg Au g/t	Std Dev	CoV	Minimum Au g/t	Maximum Au g/t
101	177	0.54	2.37	4.39	0.008	28.90
102	485	0.51	1.40	2.77	0.013	26.91
103	2,083	0.52	1.99	3.84	0.008	79.70

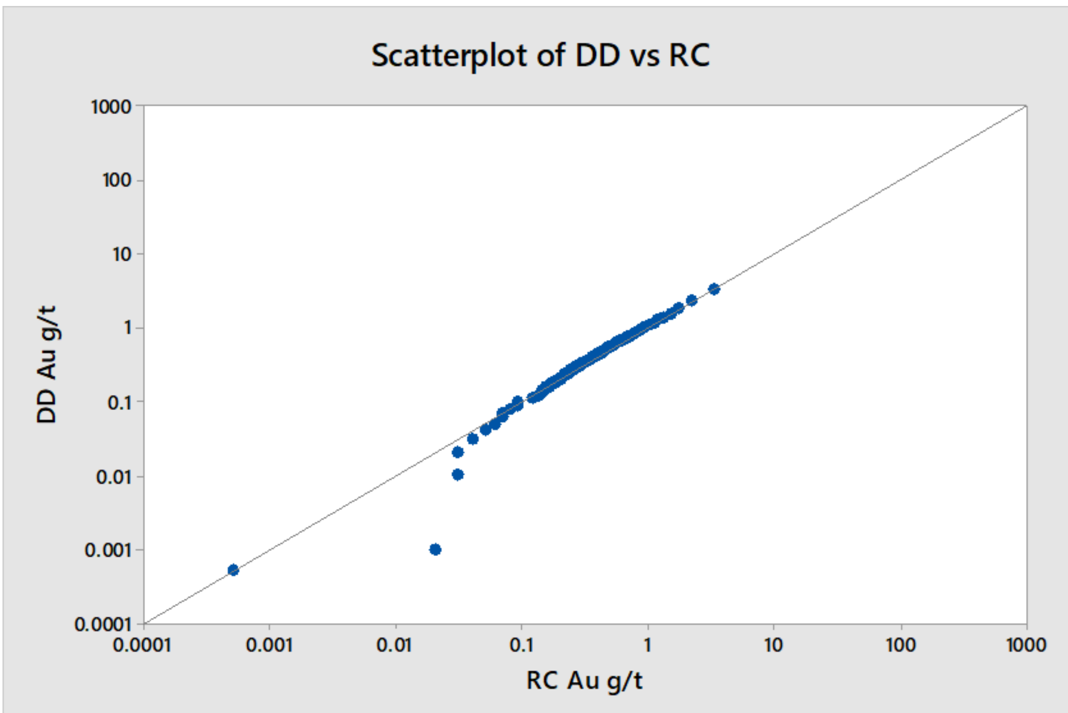
Domain	Count	Avg Au g/t	Std Dev	CoV	Minimum Au g/t	Maximum Au g/t
104	930	0.35	0.37	1.06	0.013	5.93
105	336	0.40	0.52	1.30	0.009	4.89
106	5,564	1.09	23.98	22.02	0.001	1,783.57
107	933	0.40	1.00	2.47	0.001	19.81
108	4,122	0.41	0.87	2.15	0.004	44.46
109	5,612	0.44	0.85	1.95	0.001	42.59
110	5,865	0.47	0.83	1.77	0.001	32.37
111	5,970	0.47	0.64	1.35	0.001	15.30
112	1,639	0.42	1.18	2.81	0.001	22.47
113	5,303	0.49	0.94	1.91	0.001	31.62
114	4,320	0.40	0.66	1.64	0.002	21.40
115	278	0.51	1.05	2.07	0.005	10.16
116	57	0.42	0.31	0.74	0.054	1.39
117	64	0.38	0.46	1.20	0.015	2.59
118	1,365	0.35	0.94	2.67	0.011	26.84
120	486	0.41	0.86	2.12	0.032	12.72
121	161	0.31	0.37	1.19	0.051	3.93
122	1,559	0.32	0.64	2.04	0.008	15.59
123	1,382	0.31	0.43	1.36	0.010	8.38
124	109	0.29	0.20	0.69	0.049	1.17
125	295	0.33	0.31	0.92	0.001	2.90
126	299	0.32	0.40	1.25	0.047	5.38
127	180	0.28	0.23	0.80	0.013	1.39
128	242	0.26	0.21	0.83	0.013	2.03
129	192	0.37	0.64	1.74	0.046	5.57
130	80	0.39	0.57	1.45	0.050	3.73
131	73	0.48	1.66	3.46	0.014	14.19
132	542	0.38	1.26	3.29	0.009	27.92
133	569	0.39	0.43	1.11	0.003	4.10
134	1,458	0.38	0.74	1.96	0.008	16.15
135	1,266	0.51	2.96	5.75	0.003	73.93
136	636	0.32	0.37	1.15	0.006	4.04
137	669	0.42	0.98	2.36	0.004	15.50
138	568	0.41	0.62	1.51	0.009	8.32
141	507	0.39	0.54	1.36	0.001	7.40

Domain	Count	Avg Au g/t	Std Dev	CoV	Minimum Au g/t	Maximum Au g/t
142	5,760	0.67	1.73	2.57	0.001	67.04
143	3,776	0.65	1.78	2.73	0.001	54.48
144	149	0.43	0.64	1.49	0.001	5.04
145	388	0.50	1.00	2.01	0.002	12.42
146	119	0.35	0.35	1.02	0.009	1.61
147	399	0.57	1.00	1.74	0.001	11.45
148	249	0.79	1.35	1.72	0.023	11.27
149	1,312	0.55	0.99	1.80	0.001	13.02
150	735	1.07	2.46	2.31	0.001	32.08
151	63	0.37	0.29	0.79	0.003	1.79
152	116	0.89	3.23	3.63	0.005	23.75
153	139	0.54	0.78	1.44	0.004	7.09
154	107	0.48	0.64	1.34	0.003	4.15
155	210	0.84	0.91	1.09	0.001	5.90
156	98	0.35	0.42	1.19	0.005	3.22
157	282	0.45	0.55	1.20	0.007	4.41
158	495	0.48	0.70	1.45	0.001	10.76
159	108	0.36	0.34	0.95	0.029	1.83
160	115	0.34	0.36	1.06	0.039	2.11
161	42	0.36	0.31	0.86	0.098	1.65
162	95	0.32	0.34	1.07	0.051	2.17
163	137	0.34	0.34	1.00	0.035	2.82
164	70	0.52	0.58	1.13	0.067	3.07
165	56	0.39	0.49	1.26	0.059	2.95
166	19	0.34	0.26	0.76	0.096	1.20
167	58	0.39	0.39	1.00	0.033	2.02
168	35	0.88	2.09	2.38	0.053	9.50
169	67	0.38	0.37	0.96	0.017	1.80
170	83	0.39	0.34	0.88	0.018	1.85
171	61	0.46	0.65	1.42	0.002	4.10
172	701	0.49	0.67	1.35	0.002	8.81
173	187	0.42	0.46	1.10	0.009	3.35
174	423	0.46	0.57	1.23	0.002	5.75
175	305	0.50	0.57	1.14	0.001	5.34
176	397	0.39	0.74	1.92	0.007	12.24

Domain	Count	Avg Au g/t	Std Dev	CoV	Minimum Au g/t	Maximum Au g/t
TOTAL	**73,732**	**0.52**	**6.68**	**12.76**	**0.001**	**284.00**

Examination of the RC and DD composite grades shows no significant difference nor a bias between the drill sample populations (Figure 14.6.5).

Figure 14.6.5 QQ Plot of Siga Zone RC Composite Grades Versus DD Composite Grades



14.6.5 Treatment of Extreme Values

Capping thresholds were determined by the decomposition of the individual composite log-probability distributions, and examination of box-plots and histograms. Potential outliers are not markedly clustered in localized high-grade areas and sub-domaining is therefore not warranted. Composites are capped to the defined threshold prior to grade estimation (Table 14.6.6).

Table 14.6.6 Siga Zone Capping Thresholds and Summary Statistics

Domain	Count	Cap	Number Capped	Mean	Capped Mean	Change %
101	178	4	3	0.54	0.34	-36

Domain	Count	Cap	Number Capped	Mean	Capped Mean	Change %
102	486	7	2	0.51	0.47	-8
103	2,084	28	1	0.52	0.49	-5
104	931	2	5	0.35	0.34	-2
105	337	3	4	0.40	0.39	-2
106	5,565	24	4	1.09	0.75	-31
107	934	5	7	0.40	0.37	-8
108	4,123	6	5	0.41	0.39	-3
109	5,613	10	4	0.44	0.43	-2
110	5,866	12	4	0.47	0.46	-1
111	5,971	10	3	0.47	0.47	0
112	1,640	10	9	0.42	0.40	-5
113	5,304	15	4	0.49	0.49	-1
114	4,321	10	5	0.40	0.40	-1
115	279	4	4	0.51	0.45	-11
116	58	1	5	0.41	0.40	-2
117	65	2.5	1	0.38	0.38	-2
118	1366	8	2	0.35	0.33	-5
120	487	2	9	0.41	0.35	-15
121	162	1	3	0.31	0.28	-8
122	1,560	8	4	0.32	0.31	-2
123	1,383	3	6	0.31	0.30	-3
124	110	1	2	0.29	0.28	-1
125	296	1	11	0.33	0.31	-5
126	300	2	2	0.32	0.30	-5
127	181	1	5	0.28	0.27	-2
128	243	1	2	0.26	0.25	-3
129	193	2	5	0.37	0.33	-11
130	81	2	2	0.39	0.35	-10
131	74	2	2	0.48	0.31	-36
132	543	2	5	0.38	0.32	-16
133	570	2	5	0.39	0.38	-2
134	1,459	10	2	0.38	0.37	-2
135	1,267	8	5	0.51	0.38	-25
136	637	3	1	0.32	0.32	0
137	670	8	2	0.42	0.40	-5

Domain	Count	Cap	Number Capped	Mean	Capped Mean	Change %
138	569	5	3	0.41	0.41	-2
141	508	2	12	0.39	0.38	-4
142	5,761	20	10	0.67	0.65	-3
143	3,777	14	6	0.65	0.62	-5
144	150	2	6	0.44	0.39	-11
145	389	4	6	0.50	0.45	-10
146	120	1	8	0.35	0.32	-6
147	400	4	7	0.57	0.54	-6
148	250	5	6	0.78	0.72	-8
149	1,313	10	2	0.55	0.55	-1
150	736	12	8	1.07	1.00	-6
151	64	1	2	0.37	0.35	-5
152	117	1	10	0.88	0.35	-60
153	140	4	1	0.54	0.52	-4
154	108	2	4	0.48	0.44	-8
155	211	3	6	0.83	0.80	-4
156	99	2	1	0.35	0.34	-3
157	283	3	4	0.45	0.45	-1
158	496	1	53	0.48	0.39	-18
159	109	1	7	0.36	0.33	-8
160	116	1	8	0.34	0.31	-9
161	43	1	5	0.37	0.34	-9
162	96	1	3	0.32	0.30	-8
163	138	1	5	0.34	0.32	-6
164	71	1	9	0.51	0.40	-22
165	57	1	4	0.39	0.33	-15
166	20	1	1	0.34	0.33	-2
167	59	1	4	0.40	0.35	-12
168	36	1	4	0.86	0.37	-58
169	68	1	6	0.40	0.36	-11
170	84	1	5	0.39	0.37	-6
171	62	2	2	0.45	0.42	-6
172	702	4	5	0.49	0.48	-3
173	188	2	4	0.42	0.40	-5
174	424	3	4	0.46	0.45	-2

Domain	Count	Cap	Number Capped	Mean	Capped Mean	Change %
175	306	4	1	0.50	0.49	-1
176	398	5	1	0.39	0.37	-4

14.6.6 Variography and Continuity Analysis

Three-dimensional continuity analyses (variography) were conducted on the domain-coded uncapped composite data using normal-score and 50% indicator semi-variograms. The downhole variograms were viewed at a 1.00 m lag spacing (equivalent to the composite length) to assess the nugget variance contribution.

The Authors were unable to derive semi-variograms for the individual mineralization domains, which tended towards pure-nugget effect semi-variograms or very short range semi-variograms, reflecting the relatively small number of assays and the homogeneity of mineralization within the constrained mineralization domains (see Section 5.11). For grade estimation purposes a best-fit isotropic semi-variogram was constructed from the combined composites (Equation 1).

$$0.70 + 0.20 \times SPH(10.0\ m) + 0.10 \times SPH(50\ m) \qquad (1)$$

14.6.7 Mineral Resource Estimate

Bulk density was estimated by Inverse Distance Squared (ID2) anisotropic linear interpolation using a minimum of three and maximum of twelve composites within a 300 m diameter search envelope. Sample selection was restricted to a maximum of four composite samples from a single drill hole within the same oxide state domain. Bulk density estimation was performed by weathering unit within the weathered portion of the deposits, and by lithological domain within the fresh Birimian portion of the deposits. These weathering and lithological units define hard boundaries for bulk density estimation. For those areas of limited sample data, the mean value of the specific lithology type in the Fresh domain or within the weathering unit above the Fresh unit was applied. Orezone applies a reduction factor of 5.5% and 2.5% to individual samples in the oxide and transition zones, respectively, for the block model dry bulk density calculations. Bulk density block values were adjusted by these factors after estimation.

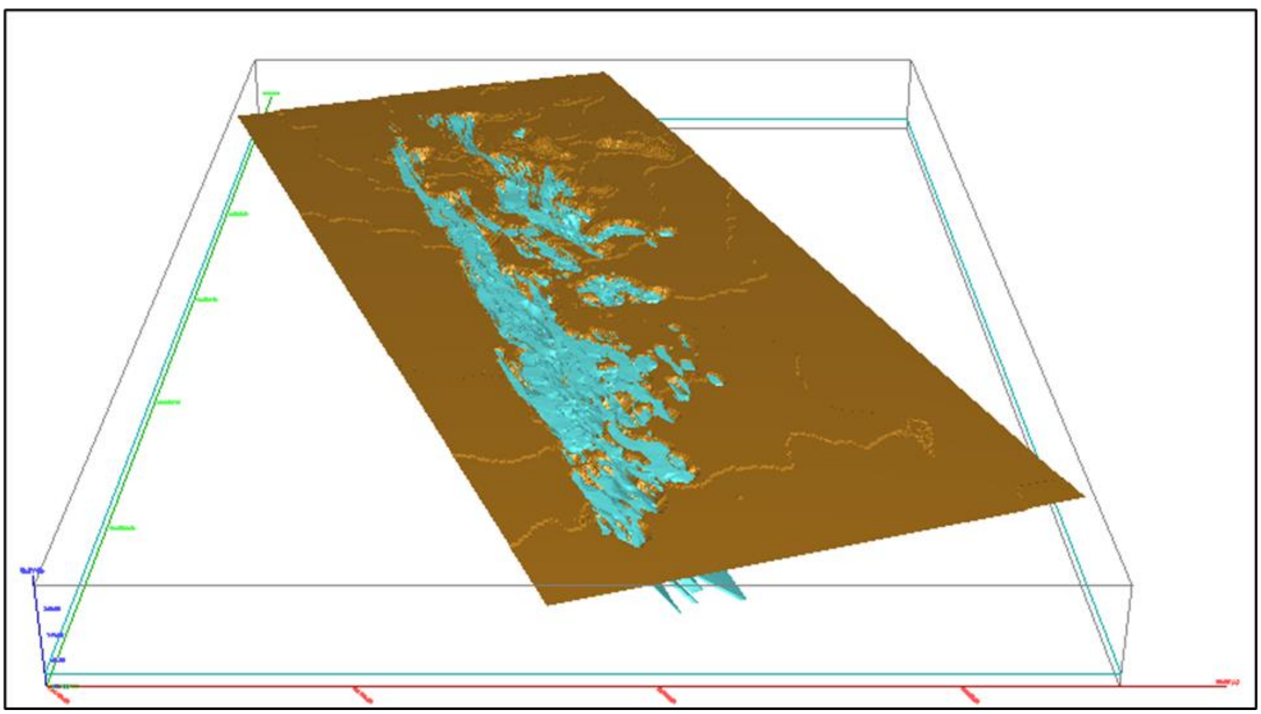
Block grades for gold were estimated by Inverse Distance Cubed (ID3) weighting of capped composites using a minimum of three and maximum of twelve composites within an extended 600 m x 300 m x 50 m search envelope. Sample selection selected the nearest composites to the block centroid and was restricted to a maximum of four composite samples from a single drill hole.

The orientations of the search ellipsoids were defined by Orezone geologists based on the local geology within each respective mineralization domain. The defined orientations were then converted to Gemcom™ format rotations. Ordinary Kriging (OK) and capped Nearest Neighbour models (NN) were also generated using the same search parameters. Only composites derived from DD and RC drill holes were used for grade estimation. Search and grade estimation were constrained by the individual mineralization domains, which define hard boundaries for grade estimation.

The parameters used to define classification limits included spatial analysis, drill hole spacing, and the observed continuity of the mineralization. Mineral Resources were classified algorithmically based on the local drill hole spacing within each mineralization domain. All blocks within 25 m of three or more drill holes were classified as Measured, blocks within 50 m of three or more drill holes were classified as Indicated, and all additional estimated blocks were classified as Inferred.

Mineral Resources reported herein have been constrained within an optimized pit (Figure 14.6.6). The results from the optimized pit shell are utilized solely for the purpose of reporting Mineral Resources and include Measured, Indicated and Inferred Mineral Resources. Mineral Resources are reported at a cut-off grade of 0.25 g/t Au for regolith, oxide and upper transition lithologies, and 0.45 g/t Au for lower transition and fresh lithologies (Table 14.6.7). The Authors are reporting whole block volumes for only those blocks where the volume inclusion is ≥50%. Orezone applies a 0.85 factor to the Siga oxide grade to account for artisanal mining.

Figure 14.6.6 Isometric View of Siga Zone Optimized Pit-Shell



View looking north.

Table 14.6.7 Siga Zone Mineral Resource Estimate*

Total	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	NA	19,831	0.72	453
Indicated	NA	37,578	0.78	944
Meas + Ind	NA	57,409	0.76	1405
Inferred	NA	3,846	0.81	100

Regolith	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.25	1	0.39	0
Indicated	0.25	1	0.52	0
Meas + Ind	0.25	2	0.44	0
Inferred	0.25	0	0.00	0

Oxide	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.25	7,356	0.59	131
Indicated	0.25	7,609	0.49	119
Meas + Ind	0.25	14,965	0.54	259
Inferred	0.25	377	0.43	5

Trans Upper	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.25	4,583	0.62	92
Indicated	0.25	4,400	0.54	77
Meas + Ind	0.25	8,983	0.58	168
Inferred	0.25	104	0.52	2

Trans Lower	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.45	2,569	0.88	73
Indicated	0.45	2,005	0.83	54
Meas + Ind	0.45	4,574	0.86	127
Inferred	0.45	46	0.80	1

Fresh	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.45	5,321	0.92	158
Indicated	0.45	23,564	0.92	694
Meas + Ind	0.45	28,885	0.92	852
Inferred	0.45	3,320	0.86	92

*Mineral Resources are inclusive of Mineral Reserves. Totals may differ due to rounding.

14.6.8 Validation

The block model was validated visually by the inspection of successive vertical cross-section lines, in order to confirm that the block models correctly reflect the distribution of high-grade and low-grade values.

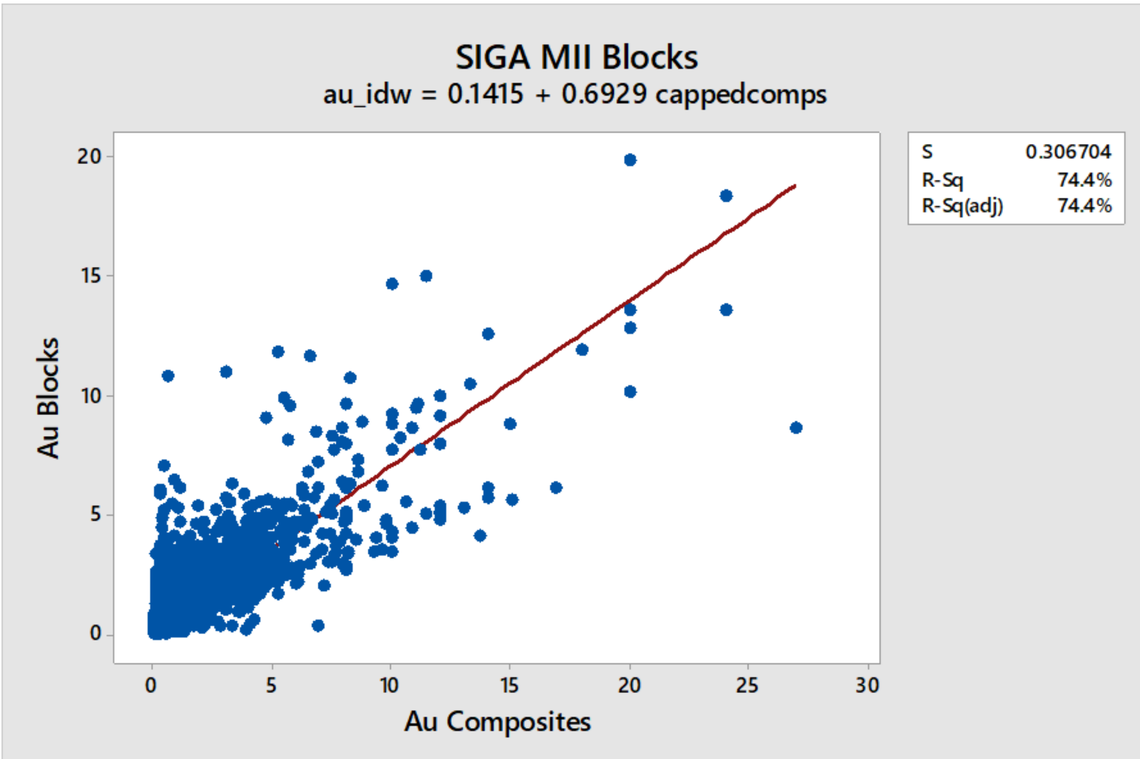
The average estimated block grades were compared to the average NN block estimate at a 0.001 g/t cut-off grouped by oxidation state domain (Table 14.6.8). The results fall within acceptable limits for linear grade estimation.

Table 14.6.8 Comparison of Siga Zone OK, ID3 and NN Average Block Grades

Unit	Au OK g/t	Au ID g/t	Au NN g/t	Ratio ID/NN %
Oxide / Saprolite	0.435	0.434	0.436	100
Transition Upper	0.445	0.442	0.445	99
Transition Lower	0.429	0.426	0.427	100
Fresh	0.431	0.429	0.431	100
Regolith	0.448	0.440	0.443	99
Total	**0.446**	**0.439**	**0.442**	**99**

An additional validation check was completed by comparing the average grade of the uncapped composites in a block to the associated model block grade estimate (Figure 14.6.7).

Figure 14.6.7 Siga Zone Regression Plot Between Block Grades and Average Composite Grades



The volume estimated was also checked against the reported volume of the individual mineralization domains and the estimated volumes for both the Whole Block Centroid model and Partial Block model (Table 14.6.9). The results fall within acceptable limits for linear estimation.

Table 14.6.9 Siga Zone Volume Reconciliation

Domain	Volume k m^3	Estimated Whole Block Volume k m^3	Estimated Partial Block Volume k m^3	Whole Block 50% Volume k m^3
S101	939	1,186	938	939
S102	1,652	1,782	1,568	1,593
S103	9,193	9,740	9,178	9,185
S104	2,560	3,121	2,557	2,552
S105	2,040	1,530	1,320	1,343
S106	15,013	19,118	14,079	14,088
S107	4,532	5,448	4,529	4,523
S108	16,570	18,660	16,566	16,594

Domain	Volume k m^3	Estimated Whole Block Volume k m^3	Estimated Partial Block Volume k m^3	Whole Block 50% Volume k m^3
S109	17,221	18,571	14,517	14,558
S110	24,127	27,285	23,789	23,901
S111	19,330	19,500	17,038	17,102
S112	11,233	11,214	10,177	10,225
S113	25,110	27,958	23,630	23,722
S114	16,160	15,301	13,104	13,118
S115	2,488	2,958	2,413	2,422
S116	119	163	120	119
S117	66	132	66	60
S118	7,473	6,412	5,752	5,764
S120	847	1,243	800	806
S121	177	251	177	176
S122	3,160	3,712	2,613	2,621
S123	11,367	8,336	7,809	7,822
S124	215	2,058	1,547	1,549
S125	900	788	606	610
S126	8,926	4,269	4,010	4,029
S127	1,342	4,054	2,489	2,489
S128	472	579	448	449
S129	2,309	4,268	1,758	1,759
S130	703	991	699	712
S131	328	501	322	328
S132	3,097	3,949	2,195	2,198
S133	4,184	4,973	4,287	4,324
S134	6,809	7,193	6,685	6,771
S135	2,185	2,231	1,867	1,874
S136	3,176	3,785	3,163	3,169
S137	3,255	3,527	3,117	3132
S138	4,844	4,483	4,043	4,038
S141	653	821	653	650
S142	8,437	9,161	8,391	8,410
S143	8,886	9,696	8,877	8,881
S144	735	857	737	738
S145	2,521	2,856	2,521	2,518

Domain	Volume k m^3	Estimated Whole Block Volume k m^3	Estimated Partial Block Volume k m^3	Whole Block 50% Volume k m^3
S146	1,752	2,046	1,746	1,743
S147	1,388	1,769	1,376	1,369
S148	235	298	233	233
S149	5,409	5,358	4,893	4,892
S150	1,115	1,451	1,063	1,058
S151	204	291	204	203
S152	130	220	130	126
S153	457	642	433	430
S154	120	188	108	104
S155	168	247	168	165
S156	783	872	782	782
S157	1,260	1,000	818	831
S158	231	247	231	231
S159	65	134	65	54
S160	1,017	1,125	1,017	1,018
S161	190	249	189	190
S162	176	220	172	177
S163	384	421	377	382
S164	176	196	174	176
S165	133	173	133	133
S166	130	149	130	130
S167	65	82	64	64
S168	72	99	72	71
S169	212	249	212	212
S170	250	328	251	251
S171	190	268	190	190
S172	706	842	706	705
S173	147	212	148	142
S174	424	543	423	420
S175	447	560	450	451
S176	957	1038	836	846
Total	**274,349**	**296,178**	**248,947**	**249,636**

14.6.9 Siga Cut-Off Sensitivity

The sensitivity of the Mineral Resource block grades to changes in cut-off grade was evaluated by calculating the total Mineral Resource inventory at various cut-offs within the Mineral Resource pit (Table 14.6.10). The values presented are intended for the sole purpose of demonstrating the sensitivity of the Mineral Resource with respect to the cut-off grade.

Table 14.6.10 Siga Grade Tonnage Sensitivity Table

OXIDE MEASURED + INDICATED				OXIDE INFERRED		
Au Cut-Off	Tonnes	Au		Au Cut-Off	Tonnes	Au
g/t	k	g/t		g/t	k	g/t
1.00	2,357.6	1.59		1.00	21.2	1.33
0.95	2,649.8	1.52		0.95	22.3	1.31
0.90	2,993.5	1.45		0.90	25.8	1.25
0.85	3,393.2	1.37		0.85	30.3	1.18
0.80	3,852.9	1.30		0.80	31.7	1.17
0.75	4,420.7	1.23		0.75	37.7	1.10
0.70	5,120.0	1.16		0.70	48.4	1.00
0.65	5,918.1	1.09		0.65	62.6	0.92
0.60	6,902.9	1.02		0.60	83.9	0.84
0.55	8,115.3	0.95		0.55	114.3	0.76
0.50	9,620.2	0.88		0.50	149.9	0.69
0.45	11,491.9	0.81		0.45	192.4	0.64
0.40	13,818.6	0.74		0.40	237.3	0.59
0.35	16,684.3	0.67		0.35	300.3	0.54
0.30	20,131.8	0.61		0.30	391.3	0.49
0.25	23,949.3	0.55		0.25	480.5	0.45
0.20	27,406.5	0.51		0.20	523.0	0.43

SULPHIDE MEASURED + INDICATED				SULPHIDE INFERRED		
Au Cut-Off	Tonnes	Au		Au Cut-Off	Tonnes	Au
g/t	k	g/t		g/t	k	g/t
1.00	8,278.5	1.24		1.00	740.8	1.62
0.95	9,245.0	1.20		0.95	819.0	1.56
0.90	10,349.2	1.17		0.90	929.9	1.48
0.85	11,690.0	1.13		0.85	1,067.7	1.41
0.80	13,277.4	1.10		0.80	1,215.1	1.34
0.75	15,063.5	1.06		0.75	1,386.7	1.27
0.70	17,179.6	1.03		0.70	1,604.4	1.19
0.65	19,568.5	1.00		0.65	1,861.6	1.12
0.60	22,339.3	0.98		0.60	2,155.6	1.05
0.55	25,535.8	0.95		0.55	2,525.4	0.98
0.50	29,245.6	0.93		0.50	2,958.2	0.92
0.45	33,459.4	0.91		0.45	3,365.9	0.86
0.40	38,097.8	0.88		0.40	3,789.4	0.81
0.35	43,322.9	0.85		0.35	4,212.6	0.77
0.30	48,981.4	0.80		0.30	4,587.5	0.73
0.25	54,767.0	0.75		0.25	4,880.1	0.71
0.20	59,585.4	0.70		0.20	5,126.8	0.68

Oxide includes Regolith, Oxide and Transitional Upper units.
Sulphide includes Transitional Lower and Fresh units.
Totals may differ due to rounding.

14.7 Bomboré P16 Zone

14.7.1 Data Supplied

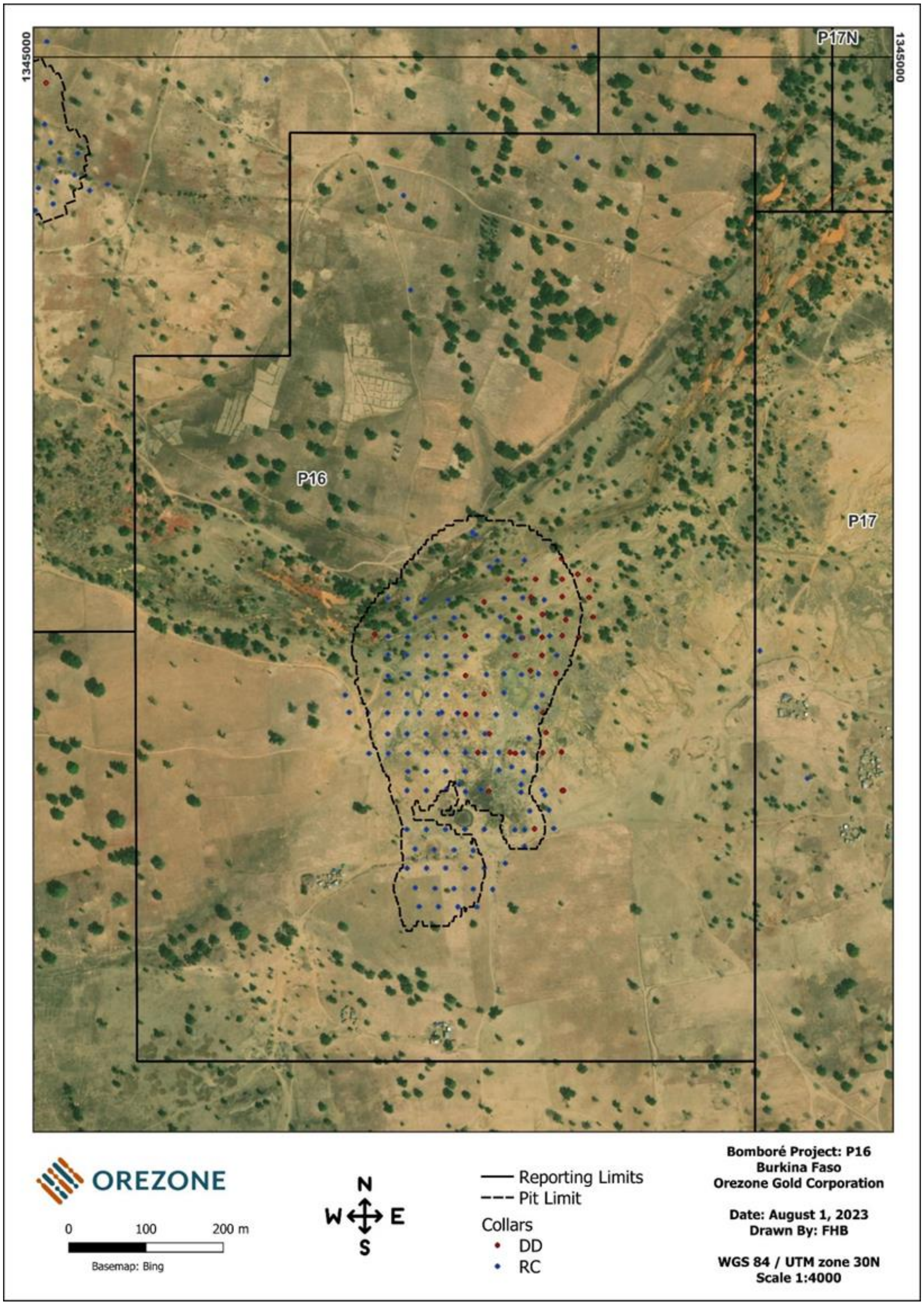
The P16 Zone model was developed by the Authors. Topography, mineralization, lithology and oxidation state three-dimensional wireframes were created by Orezone using Leapfrog Geo™ 2022.1.1. Drill hole distances are reported in metres and gold grades are reported as g/t. The drill hole collar coordinates were provided in the WGS84 Zone 30N UTM coordinate reference system.

The supplied drill hole database for the P16 Zone contains 226 unique drill hole collar records (Table 14.7.1 and Figure 14.7.1). Only Reverse Circulation (RC) and Diamond Drill Holes were used for the Mineral Resource Estimate.

Table 14.7.1 P16 Zone Drill Hole Database Summary

Type	Count	Metres
Pressure Meter Drill Hole	2	41.0
Channel	3	5.5
Diamond Core Drill Hole (DD)	39	5,992.5
Auger Drill Hole	51	241.0
Rotary Air Blast Drill Hole (RAB)	5	51.0
Reverse Circulation Drill Hole (RC)	123	6,529.0
Trench	3	12.5
Total	**226**	**12,872.5**

Figure 14.7.1 P16 Zone Drill Hole Collar Plot

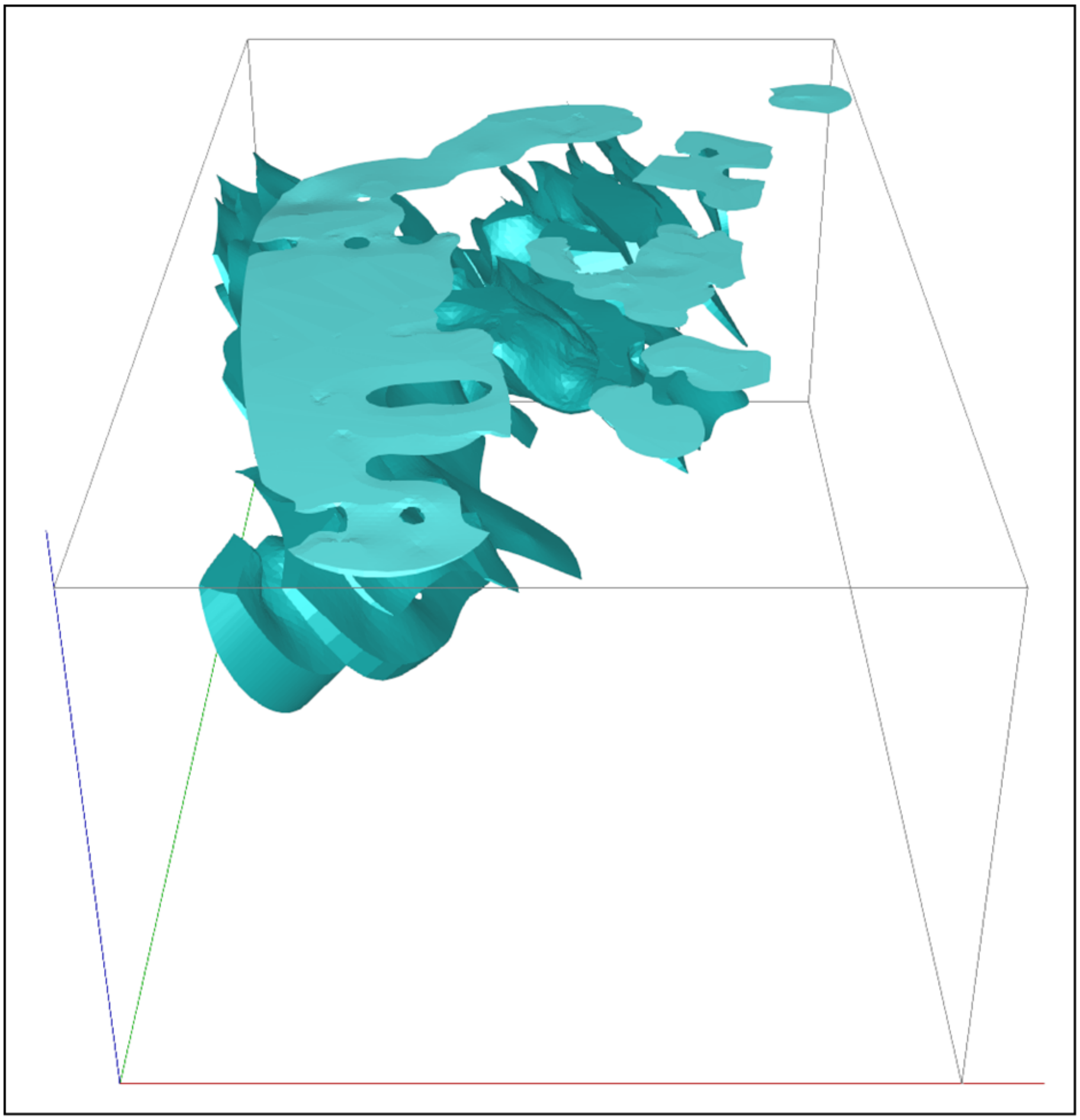


14.7.2 Exploratory Data Analysis

A total of 19 distinct mineralization domain wireframes were supplied by Orezone (Figure 14.7.2). Each domain was assigned a unique three-digit rock code.

The mean nearest neighbour collar distance (for DD and RC) is 19.2 m, the average DD length is 153.65 m and the average RC drill hole length is 53.08 m. Summary statistics for the supplied assay data are provided below (Table 14.7.2).

Figure 14.7.2 Isometric View of P16 Zone Mineralization Domains



View looking north. Field of view is approximately 300 m across.

Table 14.7.2 P16 Zone Summary Assay Statistics

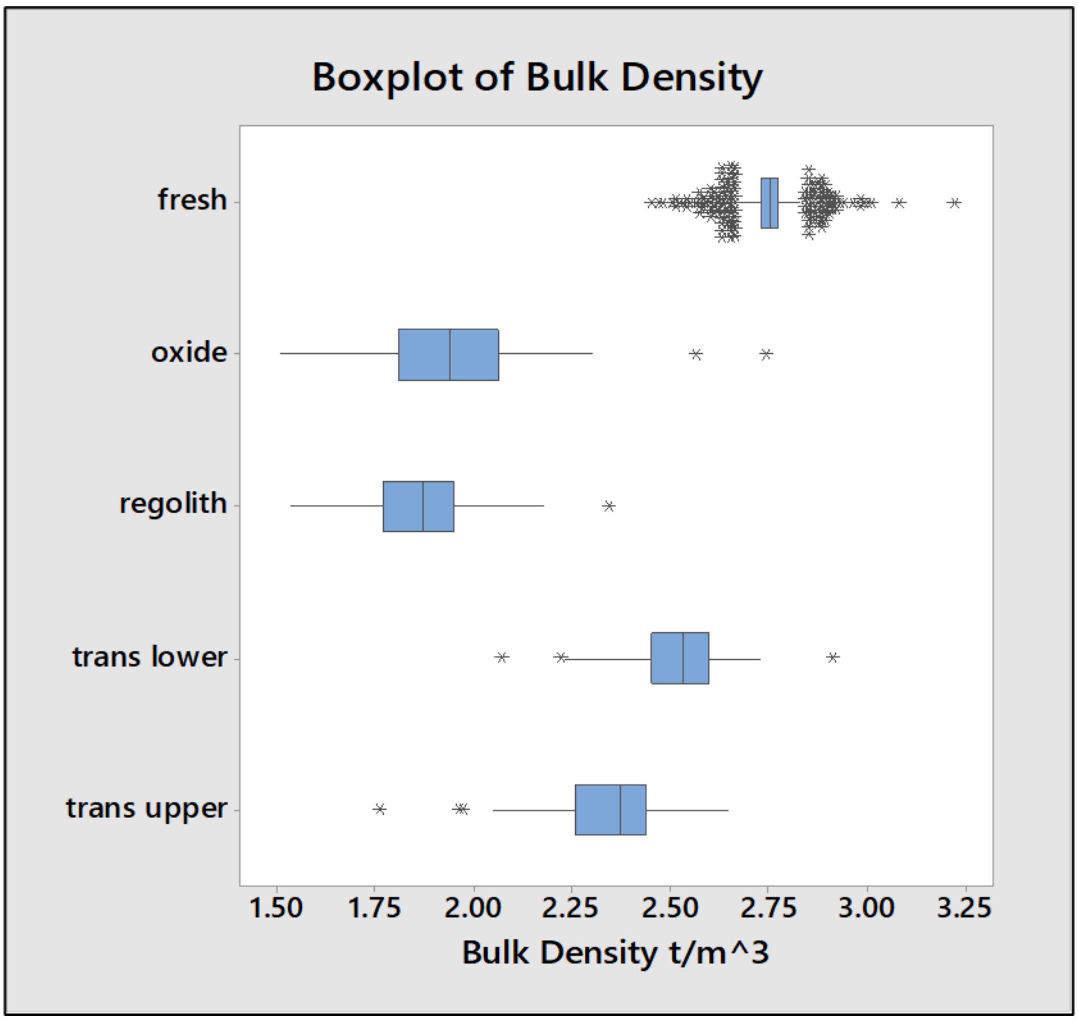
Domain	Count	Avg Length m	Min Length m	Max Length m	Avg Au g/t	Min Au g/t	Max Au g/t	Std Dev g/t	CoV
0	7,302	1.10	0.50	7.00	0.06	0.00	21.32	0.38	6.27
401	257	1.01	1.00	2.00	0.52	0.00	3.90	0.67	1.29
402	431	1.03	0.70	2.00	0.57	0.01	6.38	0.76	1.33
403	58	1.02	1.00	2.00	0.35	0.01	1.34	0.32	0.92
404	829	1.12	1.00	3.00	1.27	0.00	114.13	5.16	4.05
405	354	1.22	0.50	4.50	0.66	0.01	18.72	1.72	2.61
406	84	1.31	1.00	4.50	0.37	0.01	3.87	0.51	1.39
407	207	1.15	1.00	2.00	1.28	0.01	41.40	3.95	3.09
408	555	1.03	1.00	2.00	0.50	0.00	4.57	0.61	1.23
409	62	1.00	1.00	1.00	0.40	0.02	2.03	0.42	1.05
410	20	1.20	1.00	4.50	0.46	0.01	1.92	0.49	1.06
411	241	1.22	1.00	3.00	0.85	0.00	30.61	2.35	2.77
413	44	1.07	1.00	2.50	0.29	0.04	2.36	0.36	1.25
415	216	1.00	1.00	1.00	0.49	0.01	5.58	0.71	1.44
416	86	1.00	1.00	1.00	0.45	0.03	3.00	0.49	1.09
417	36	1.35	1.00	4.50	0.21	0.01	1.39	0.25	1.21
419	90	1.14	1.00	2.00	0.57	0.01	4.06	0.75	1.32
420	301	1.11	1.00	2.00	1.19	0.00	27.85	2.54	2.13
421	79	1.05	1.00	2.50	0.53	0.01	13.59	1.55	2.90
422	24	1.85	0.50	4.55	0.32	0.13	1.63	0.30	0.95
Total	**11,276**	**1.10**	**0.50**	**7.00**	**0.32**	**0.00**	**114.13**	**1.72**	**5.44**

The supplied database contains 3,347-point bulk density measurements from drill hole core, with values ranging from 1.51 to 3.22 t/m^3 (Table 14.7.3). Bulk density measurements were back tagged by mineralization domain, lithology and oxidation state wireframes. Bulk density measurements display a differing range of values based on oxidation state (Figure 14.7.3).

Table 14.7.3 P16 Zone Summary Bulk Density Statistics

Unity	Count	Avg. Bulk Density t/m3	Min. Bulk Density t/m3	Max. Bulk Density t/m3	Std Dev	CoV
Oxide/Saprolite	158	1.94	1.51	2.74	0.20	0.10
Transitional Upper	142	2.35	1.76	2.65	0.15	0.06
Transitional Lower	164	2.52	2.07	2.91	0.12	0.05
Fresh	2,844	2.75	2.45	3.22	0.04	0.02
Regolith	39	1.87	1.53	2.34	0.15	0.08
Total	**3,347**	**2.67**	**1.51**	**3.22**	**0.22**	**0.08**

Figure 14.7.3 Boxplot of P16 Zone Bulk Density by Oxide State



14.7.3 Block Model

An orthogonal block model was established with the block model limits selected to cover the extent of the mineralized structures, the potential open pit dimensions, and to reflect the general orientation of the mineralized zones (Table 14.7.4). The block model consists of separate variables for estimated grades, rock codes, bulk density and classification attributes.

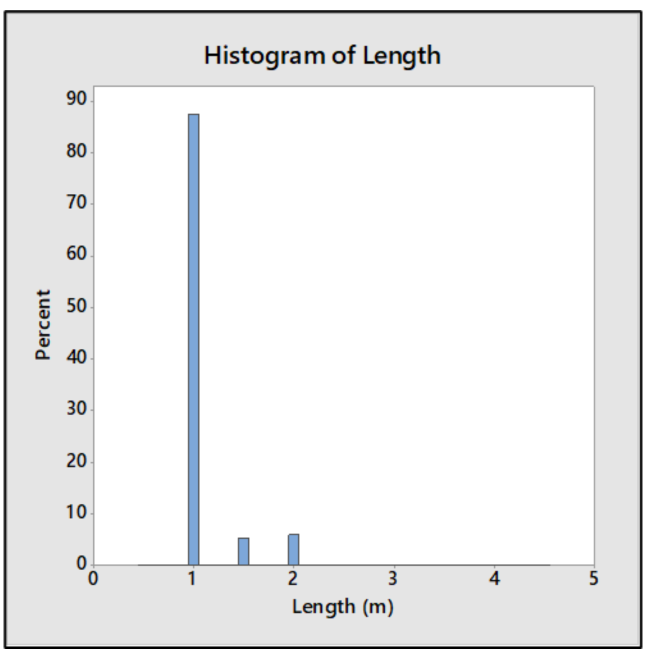
Table 14.7.4 P16 Zone Block Model Set-Up

Coordinate	Origin	Block Size m	Number of Blocks
Easting (X)	729,150	2.0	275
Northing (Y)	1,343,725	6.25	140
Elevation (max Z)	270	3.0	90

14.7.4 Compositing

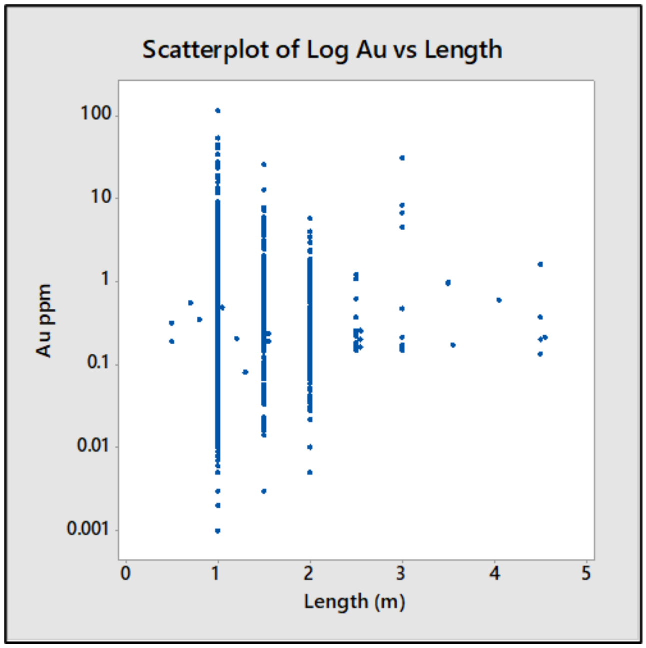
Assay sample lengths for DD and RC drill holes range from 0.50 to 7.00 m, with an average sample length of 1.10 m (Figure 14.7.4). The average sample length for DD assays is 1.13 m, and the average sample length for RC assays is 1.07 m. Constrained assay sample lengths within the defined mineralization domains range from 0.50 to 4.55 m, with an average sample length of 1.10 m. A total of 88% of the constrained assay sample lengths equal 1.00 m.

Figure 14.7.4 Histogram of P16 Zone Constrained Assay Sample Lengths



No correlation was observed between sample grade and sample length for the constrained assay samples (Figure 14.7.5).

Figure 14.7.5 Scatterplot of P16 Zone Assay Sample Lengths Versus Grade



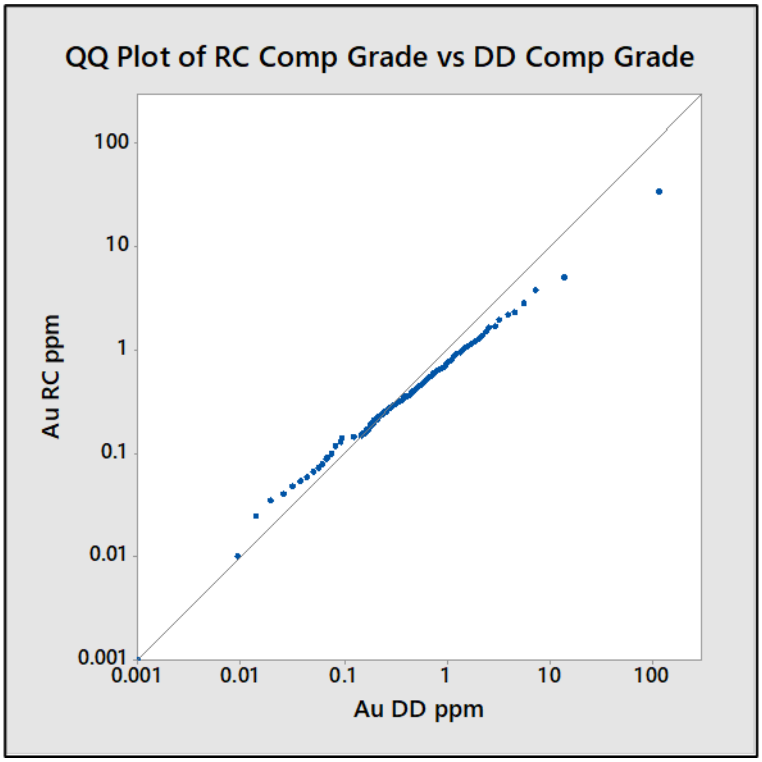
Based on the predominance of 1.00 m sample lengths, all constrained assay samples were composited to this length in order to ensure equal sample support. Length-weighted composites were calculated within the defined mineralization domains. Missing sample intervals in the data were assigned a nominal background grade of 0.001 g/t. The compositing process started at the first point of intersection between the drill hole and the domain wireframe intersected and halted on exit from the wireframe. Downhole residual composites that were less than half the compositing length were discarded so as to not introduce a short sample bias into the composite sample population. The wireframes that represent the interpreted mineralization domains were utilized to back-tag a rock code variable into the composite workspace. The composite data were then visually validated against the mineralization wireframes and extracted for analysis and grade estimation. Summary composite statistics are listed in Table 14.7.5.

Table 14.7.5 P16 Zone Summary Composite Statistics

Domain	Count	Average Au g/t	Minimum Au g/t	Maximum Au g/t	Std Dev	CoV
401	247	0.55	0.001	3.90	0.68	1.24
402	395	0.64	0.007	6.38	0.78	1.22
403	55	0.38	0.006	1.34	0.33	0.86
404	880	1.28	0.001	114.50	4.94	3.87
405	401	0.65	0.001	18.68	1.62	2.50
406	83	0.45	0.054	3.86	0.50	1.11
407	214	1.40	0.001	41.27	4.01	2.86
408	530	0.53	0.001	4.57	0.62	1.17
409	56	0.43	0.032	2.03	0.42	0.97
410	18	0.57	0.179	1.92	0.47	0.82
411	275	1.12	0.001	30.61	3.41	3.04
413	41	0.31	0.001	2.36	0.37	1.18
415	201	0.53	0.009	5.58	0.73	1.38
416	83	0.46	0.041	3.00	0.49	1.06
417	41	0.23	0.011	1.39	0.24	1.03
419	95	0.61	0.010	4.06	0.73	1.20
420	312	1.31	0.003	27.83	2.52	1.93
421	67	0.53	0.022	13.59	1.66	3.16
422	32	0.27	0.129	0.61	0.14	0.52
Total	**4,026**	**0.85**	**0.001**	**114.50**	**2.84**	**3.34**

Examination of the RC and DD composite grade distributions suggests that the RC grades slightly under-estimate the grade compared to the DD grades, above approximately 0.50 g/t (Figure 14.7.6). The RC meterage represents approximately 52% of the total drilling available but is typically much shallower than the DD drilling. A correction factor was not applied to the RC drilling results.

Figure 14.7.6 QQ Plot of P16 Zone RC Composite Grades Versus DD Composite Grades



14.7.5 Treatment of Extreme Values

Capping thresholds were determined by the decomposition of individual composite log-probability distributions, examination of box-plots and histograms, and reference to Tukey's rule of three times the interquartile range for extreme outliers (Tukey, 1977). Potential outliers are not markedly clustered in localized high-grade areas and sub-domaining is therefore not warranted. Composites are capped to the defined threshold prior to grade estimation. A total of three domains do not require capping (Table 14.7.6).

Table 14.7.6 P16 Zone Capping Thresholds and Summary Statistics

Domain	Count	Cap	Number Capped	Uncapped Mean	Capped Mean	Mean Above Cap	Change %
401	247	2.60	6	0.55	0.53	3.39	-4
402	395	4.00	4	0.64	0.63	4.83	-2
403	55	n/a	0	0.38	0.38	n/a	0
404	880	9.00	11	1.28	0.98	32.89	-31
405	401	4.00	9	0.65	0.53	9.36	-23

Domain	Count	Cap	Number Capped	Uncapped Mean	Capped Mean	Mean Above Cap	Change %
406	83	1.77	2	0.45	0.42	2.95	-7
407	214	6.00	7	1.40	0.98	18.90	-43
408	530	3.00	6	0.53	0.52	3.65	-2
409	56	1.48	2	0.43	0.42	2.02	-2
410	18	n/a	0	0.57	0.57	n/a	0
411	275	2.49	23	1.12	0.62	8.54	-81
413	41	0.80	2	0.31	0.27	1.61	-15
415	201	3.00	2	0.53	0.51	4.99	-4
416	83	1.49	3	0.46	0.43	2.33	-7
417	41	0.91	1	0.23	0.22	1.39	-5
419	95	2.49	3	0.61	0.58	3.20	-5
420	312	5.04	16	1.31	1.09	9.38	-20
421	67	0.97	4	0.53	0.30	4.79	-77
422	32	n/a	0	0.27	0.27	n/a	0
Total	**4,026**	**---**	**101**	**0.85**	**0.69**	**---**	**-23**

14.7.6 Variography and Continuity Analysis

Three-dimensional continuity analyses (variography) were conducted on the domain-coded uncapped composite data, utilizing a normal scores transformation within each domain.

The downhole variogram was viewed at a 1.0 m lag spacing (equivalent to the composite length) to assess the nugget variance contribution. Standardized isotropic spherical models were utilized to model the experimental semi-variograms in normal-score transformed space. Semi-variogram model ranges were checked and iteratively refined for each model relative to the overall nugget variance, and the back-transformed variance contributions were then calculated (Table 14.7.7).

In general, only poorly constrained semi-variograms for the individual mineralization domains were obtained, primarily due to the small number of data points per domain. Where a domain lacked sufficient points to generate an acceptable semi-variogram a combined domain semi-variogram was utilized for either the MR or MB models.

Table 14.7.7 P16 Zone Back Transformed Semi-Variograms

401	0.70 + 0.20 SPH(5.0m) + 0.10 SPH(20.0m)
402	0.60 + 0.20 SPH(4.0m) + 0.20 SPH(12.0m)
403	0.60 + 0.20 SPH(4.0m) + 0.20 SPH(20.0m)
404	0.50 + 0.40 SPH(5.0m) + 0.10 SPH(25.0m)
405	0.30 + 0.50 SPH(6.0m) + 0.20 SPH(30.0m)
406	0.60 + 0.20 SPH(4.0m) + 0.20 SPH(20.0m)
407	0.40 + 0.50 SPH(5.0m) + 0.10 SPH(30.0m)
408	0.40 + 0.40 SPH(10.0m) + 0.20 SPH(35.0m)
409	0.60 + 0.30 SPH(4.0m) + 0.10 SPH(20.0m)
410	0.60 + 0.30 SPH(4.0m) + 0.10 SPH(20.0m)
411	0.50 + 0.30 SPH(3.0m) + 0.20 SPH(20.0m)
413	0.60 + 0.30 SPH(4.0m) + 0.10 SPH(20.0m)
415	0.50 + 0.20 SPH(30.0m) + 0.30 SPH(50.0m)
416	0.60 + 0.30 SPH(4.0m) + 0.10 SPH(20.0m)
417	0.60 + 0.30 SPH(4.0m) + 0.10 SPH(20.0m)
419	0.60 + 0.30 SPH(4.0m) + 0.10 SPH(20.0m)
420	0.60 + 0.30 SPH(5.0m) + 0.10 SPH(25.0m)
421	0.50 + 0.40 SPH(28.0m) + 0.10 SPH(40.0m)
422	0.40 SPH(28.0m) + 0.10 SPH(40.0m)

14.7.7 Mineral Resource Estimate

Bulk density was estimated by Inverse Distance Squared (ID2) anisotropic linear interpolation using a minimum of three and maximum of twelve composites within a 300 m diameter search envelope. Sample selection was restricted to a maximum of four composite samples from a single drill hole within the same oxide state domain. Bulk density estimation was performed by weathering unit within the weathered portion of the deposits, and by lithological domain within the fresh Birimian portion of the deposits. These weathering and lithological units define hard boundaries for bulk density estimation. For those areas of limited sample data, the mean value of the specific lithology type in the Fresh domain or within the weathering unit above the Fresh unit was applied. Orezone applies a reduction factor of 5.5% and 2.5% to individual samples in the oxide and transition zones, respectively, for the block model dry bulk density calculations. Density block values were adjusted by these factors after estimation.

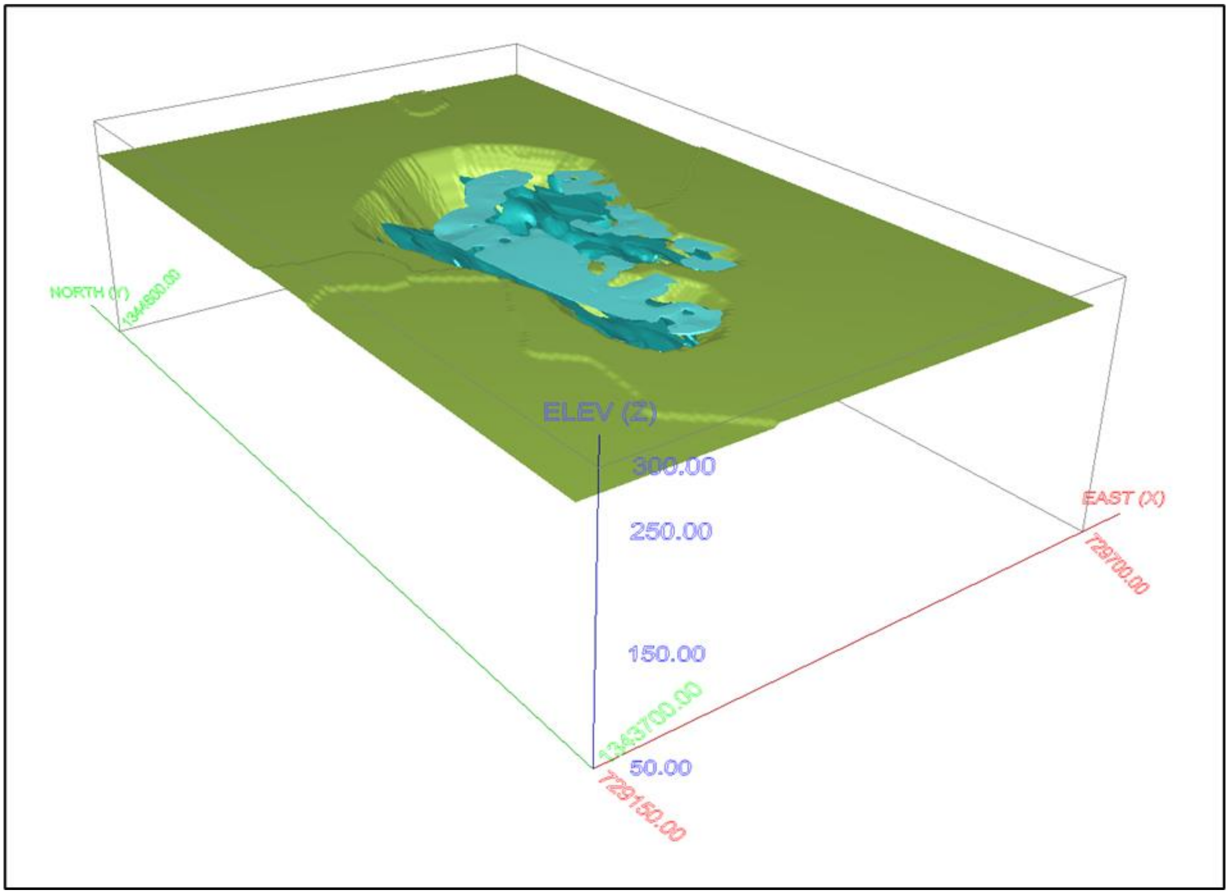
Block grades for gold were estimated by Inverse Distance Cubed (ID3) weighting of capped composites using a minimum of three and maximum of twelve composites within an extended 600 m x 300 m x 50 m search envelope. Sample selection selected the nearest composites to the block centroid and was restricted to a maximum of four composite samples from a single drill hole.

The orientations of the search ellipsoids were defined by Orezone geologists based on the local geology within each respective mineralization domain. The defined orientations were then converted to Gemcom™ format rotations. Ordinary Kriging (OK) and capped Nearest Neighbour models (NN) were also generated using the same search parameters. Only composites derived from DD and RC drill holes were used for grade estimation. Search and grade estimation were constrained by the individual mineralization domains, which define hard boundaries for grade estimation.

The parameters used to define classification limits included spatial analysis, drill hole spacing, and the observed continuity of the mineralization. Mineral Resources were classified algorithmically based on the local drill hole spacing within each mineralization domain. All blocks within 25 m of three or more drill holes were classified as Measured, blocks within 50 m of three or more drill holes were classified as Indicated, and all additional estimated blocks were classified as Inferred.

Mineral Resources reported herein have been constrained within an optimized pit (Figure 14.7.7). The results from the optimized pit shell are utilized solely for the purpose of reporting Mineral Resources and include Measured, Indicated and Inferred Mineral Resources. Mineral Resources are reported at a cut-off grade of 0.25 g/t Au for regolith, oxide and upper transition lithologies, and 0.45 g/t Au for lower transition and fresh lithologies (Table 14.7.8). The Authors are reporting whole block volumes for only those blocks where the volume inclusion is 50% or greater. Orezone applies a 0.50 factor to the P16 Zone oxide grade to account for artisanal mining.

Figure 14.7.7 Isometric View of P16 Zone Optimized Pit Shell



View looking northeast.

Table 14.7.8 P16 Zone Mineral Resource Estimate*

Total	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	NA	1770	0.94	53
Indicated	NA	750	0.83	20
Meas + Ind	NA	2519	0.91	73
Inferred	NA	23	0.72	1

Regolith	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.25	4	0.37	0
Indicated	0.25	5	0.37	0
Meas + Ind	0.25	9	0.37	0
Inferred	0.25	0	0.37	0

Oxide	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.25	552	0.40	7
Indicated	0.25	100	0.26	1
Meas + Ind	0.25	653	0.38	8
Inferred	0.25	2	0.19	0

Trans Upper	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.25	221	0.83	6
Indicated	0.25	10	0.45	0
Meas + Ind	0.25	232	0.81	6
Inferred	0.25	1	0.37	0

Trans Lower	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.45	182	1.06	6
Indicated	0.45	5	0.72	0.1
Meas + Ind	0.45	187	1.05	6
Inferred	0.45	0	0.00	0

Fresh	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.45	811	1.31	34
Indicated	0.45	629	0.93	19
Meas + Ind	0.45	1440	1.15	53
Inferred	0.45	20	0.78	1

*Resources are inclusive of Mineral Reserves. Totals may differ due to rounding.

14.7.8 Validation

The block model was validated visually by the inspection of successive vertical cross-section lines, in order to confirm that the block models correctly reflect the distribution of high-grade and low-grade values.

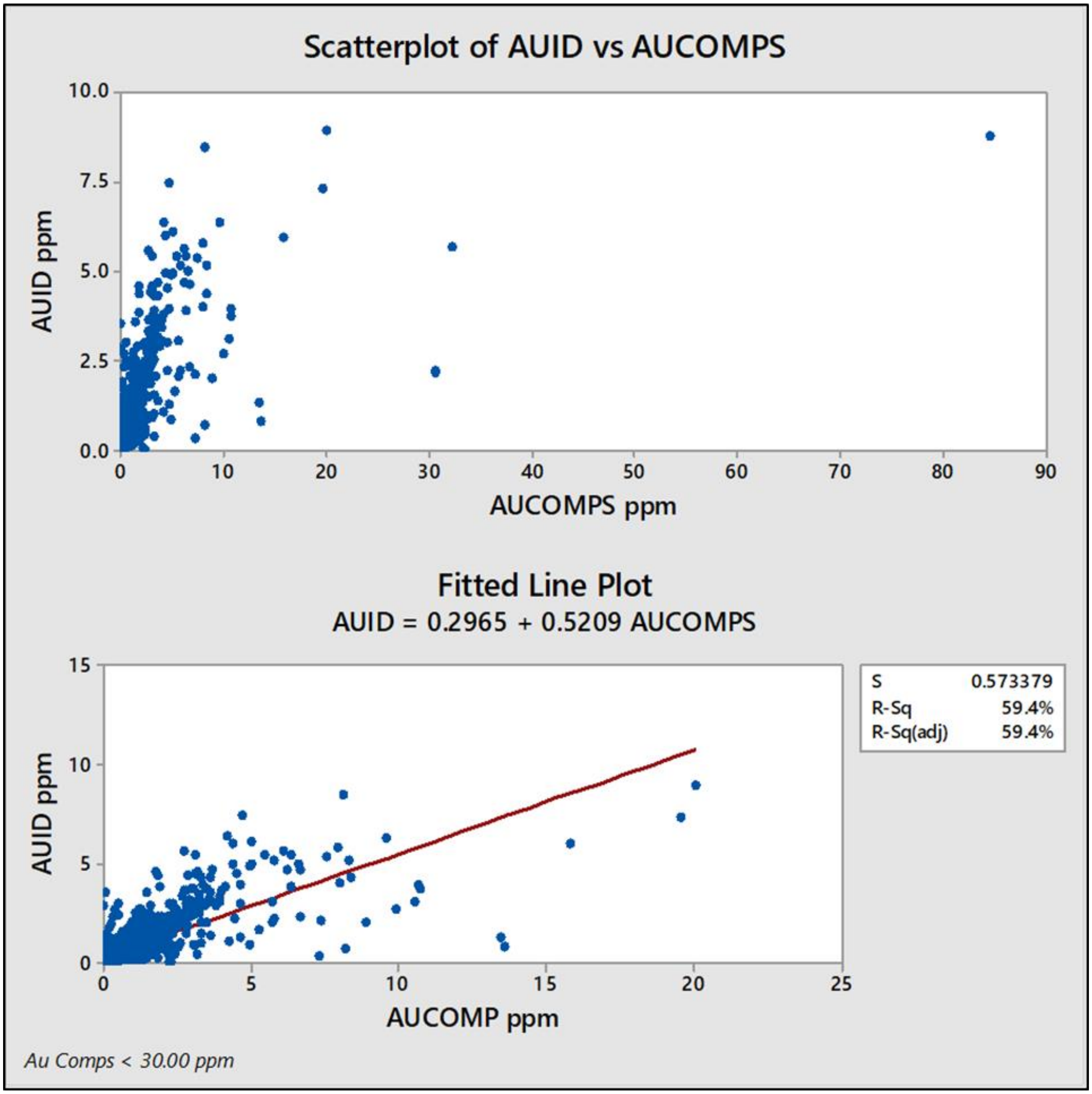
The average estimated block grades were compared to the average NN block estimate at a 0.001 g/t cut-off grouped by oxidation state domain (Table 14.7.9). The results fall within acceptable limits for grade estimation.

Table 14.7.9 P16 Zone Comparison of OK, ID3 and NN Average Block Grades

Unit	Au OK g/t	Au ID g/t	Au NN g/t	Ratio ID/NN %
Oxide / Saprolite	0.58	0.57	0.58	99
Transition Upper	0.61	0.61	0.62	99
Transition Lower	0.63	0.61	0.60	102
Fresh	0.65	0.64	0.62	103
Regolith	0.30	0.31	0.29	105
Total	**0.63**	**0.61**	**0.60**	**102**

An additional validation check was completed by comparing the average grade of the uncapped composites in a block to the associated model block grade estimate. A total of four outlier blocks with an average Au composite grade of 44.50 g/t have been reduced to an average grade of 4.72 g/t, with no other estimation issues noted. The results demonstrate a moderate amount of smoothing of higher grades and fall within acceptable limits for linear estimation (Figure 14.7.8).

Figure 14.7.8 Validation Plot Between P16 Zone Block Grades and Average Composite Grades



The volume estimated was also checked against the reported volume of the individual mineralization domains. Estimated volumes are based on a 0.001 g/t cut-off and partial block volumes (Table 14.7.10). The results fall within acceptable limits for grade estimation.

Table 14.7.10 P16 Zone Volume Reconciliation

Domain	Estimated Volume k m^3	Reported Volume k m^3	Ratio %
401	448.1	449.9	100
402	199	201.8	99
403	182.7	184.3	99
404	615.1	619.5	99
405	168.9	170.4	99
406	26.1	27.1	96
407	108.5	110.0	99
408	407.2	411.0	99
409	22.2	22.6	98
410	5.4	5.6	96
411	116.7	117.8	99
413	10.7	10.9	98
415	275.9	277.7	99
416	79	79.5	99
417	19.5	19.7	99
419	35.3	36.0	98
420	205.8	207.7	99
421	33.6	36.8	91
422	12.4	12.6	99
Total	**3,000.7**	**3,000.7**	**100**

14.7.9 Comparison

ID3 Mineral Resource Estimate were accumulated using the RPA 2017 cut-offs of 0.20 g/t for regolith, oxide, transitional upper material and lower transitional material and 0.38 g/t for fresh material. In comparison to the 2017 cut-offs, the updated ID3 mineralization demonstrates a 12% decrease in tonnage, a 7% increase in grade and a 4% increase in contained gold for the total Mineral Inventory (Table 14.7.11).

Table 14.7.11 P16 Zone Comparison Between the RPA Mineral Resource and the 2023 Update

(Total Mineral Inventory) 2023 Resources in RPA 2017 Pit and Beneath March 2023 Topographic Surface					
	Class	**Au Cut-Off g/t**	**Tonnes k**	**AuID g/t**	**AuID koz**
RPA_P16	Meas	NA	1,981	0.99	63
	Ind	NA	849	0.81	22
	Meas + Ind	NA	2,830	0.93	85
	Inferred	NA	25	0.69	1
	Total	**NA**	**2,855**	**0.93**	**85**
2017 Cut-offs in RPA 2017 Pit and Beneath March 2023 Topographic Surface					
	Class	**Au Cut-Off g/t**	**Tonnes k**	**AuID g/t**	**AuID koz**
OBSA_P16	Meas	NA	1,770	1.06	61
	Ind	NA	750	0.87	21
	Meas + Ind	NA	2,519	1.01	81
	Inferred	NA	23	0.74	1
	Total	**NA**	**2,542**	**1.00**	**82**

Totals may differ due to rounding.

14.7.10 P16 Cut-Off Sensitivity

The sensitivity of the Mineral Resource block grades to changes in cut-off grade was evaluated by calculating the total Mineral Resource inventory at various cut-offs within the Mineral Resource pit (Table 14.7.12). The values presented are intended for the sole purpose of demonstrating the sensitivity of the Mineral Resource with respect to the cut-off grade.

Table 14.7.12 P16 Grade Tonnage Sensitivity Table

OXIDE MEASURED + INDICATED				OXIDE INFERRED		
Au Cut-Off	**Tonnes**	**Au**		**Au Cut-Off**	**Tonnes**	**Au**
g/t	**k**	**g/t**		**g/t**	**k**	**g/t**
1.00	165.9	1.21		1.00	0.0	0.00
0.95	183.4	1.16		0.95	0.0	0.00
0.90	202.7	1.10		0.90	0.1	0.64
0.85	225.9	1.05		0.85	0.1	0.64
0.80	251.1	0.99		0.80	0.2	0.60
0.75	277.1	0.95		0.75	0.2	0.60

OXIDE MEASURED + INDICATED				OXIDE INFERRED		
Au Cut-Off	**Tonnes**	**Au**		**Au Cut-Off**	**Tonnes**	**Au**
g/t	**k**	**g/t**		**g/t**	**k**	**g/t**
0.70	308.3	0.90		0.70	0.2	0.60
0.65	343.8	0.85		0.65	0.2	0.60
0.60	381.8	0.81		0.60	0.2	0.54
0.55	431.0	0.76		0.55	0.3	0.51
0.50	485.9	0.71		0.50	0.3	0.51
0.45	543.8	0.66		0.45	0.4	0.47
0.40	610.0	0.62		0.40	0.4	0.47
0.35	699.1	0.57		0.35	1.1	0.34
0.30	793.3	0.53		0.30	1.6	0.30
0.25	892.7	0.49		0.25	2.6	0.26
0.20	982.2	0.46		0.20	3.1	0.25

SULPHIDE MEASURED + INDICATED				SULPHIDE INFERRED		
Au Cut-Off	**Tonnes**	**Au**		**Au Cut-Off**	**Tonnes**	**Au**
g/t	**k**	**g/t**		**g/t**	**k**	**g/t**
1.00	647.5	1.81		1.00	2.2	1.29
0.95	704.5	1.74		0.95	3.8	1.15
0.90	768.4	1.68		0.90	5.1	1.10
0.85	835.8	1.61		0.85	6.5	1.05
0.80	911.0	1.55		0.80	8.3	1.00
0.75	991.6	1.48		0.75	9.7	0.97
0.70	1,083.7	1.42		0.70	11.7	0.92
0.65	1,177.6	1.36		0.65	13.4	0.89
0.60	1,283.8	1.30		0.60	14.8	0.87
0.55	1,395.3	1.24		0.55	17.1	0.83
0.50	1,507.5	1.19		0.50	19.0	0.80
0.45	1,626.5	1.14		0.45	20.0	0.78
0.40	1,738.5	1.09		0.40	20.7	0.77
0.35	1,856.4	1.04		0.35	21.5	0.76
0.30	1,962.7	1.01		0.30	23.5	0.72
0.25	2,072.8	0.97		0.25	26.3	0.67
0.20	2,170.1	0.93		0.20	28.8	0.63

Oxide includes Regolith, Oxide and Transitional Upper units.
Sulphide includes Transitional Lower and Fresh units.
Totals may differ due to rounding.

14.8 Bomboré P17N Zone

14.8.1 Data Supplied

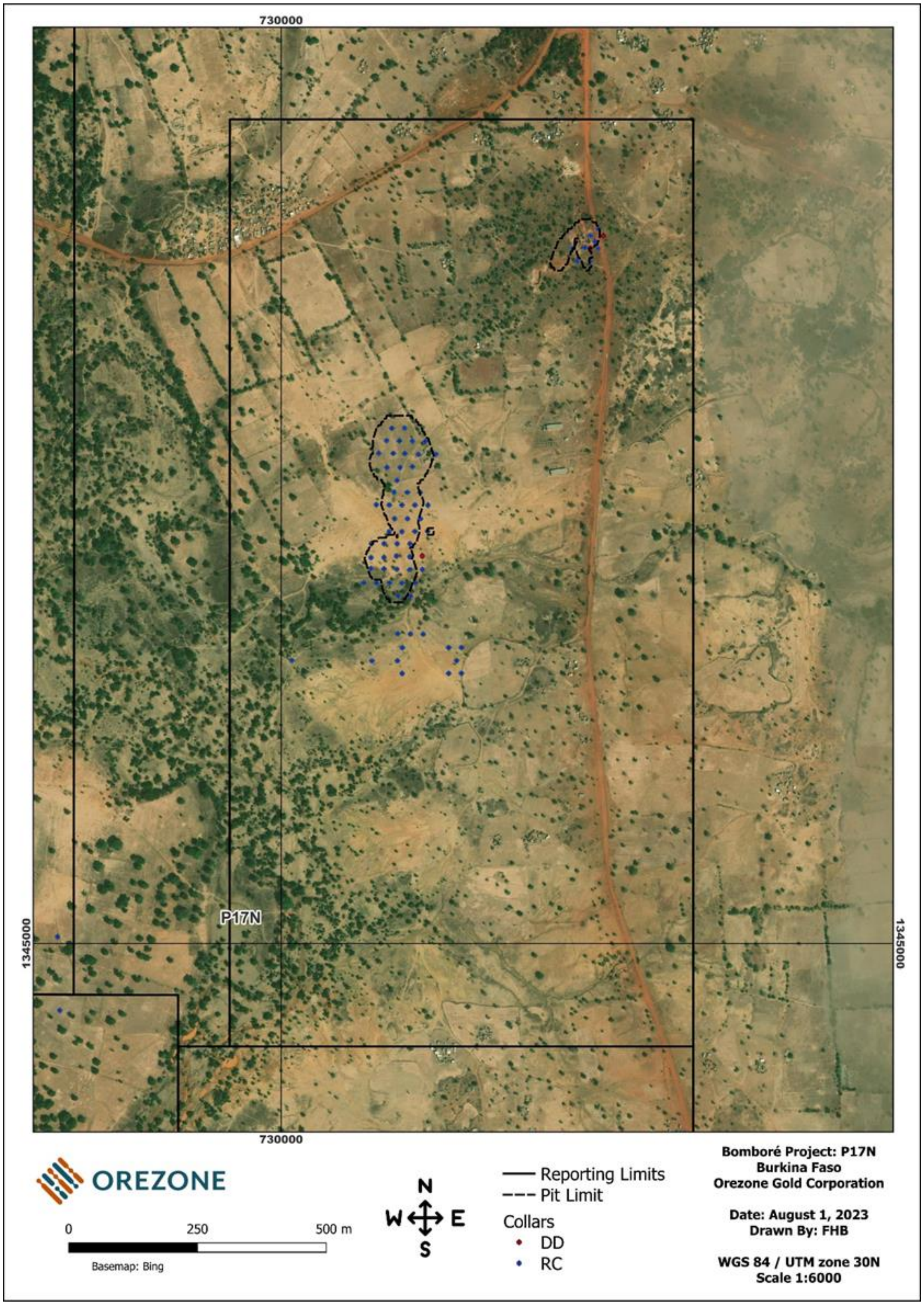
The P17N Zone model was developed by the Authors. Topography, mineralization, lithology and oxidation state three-dimensional wireframes were created by Orezone using Leapfrog Geo 2022TM.1.1. Drill hole distances are reported in metres and gold grades are reported as g/t. The drill hole collar coordinates were provided in the WGS84 Zone 30N UTM coordinate reference system.

The supplied drill hole database for the P17N Zone contains 195 unique drill hole collar records (Table 14.8.1 and Figure 14.8.1). Only Reverse Circulation (RC) and Diamond Drill Holes (DD) were used for the Mineral Resource Estimate.

Table 14.8.1 P17N Zone Drill Hole Database Summary

Type	Count	Metres
Channel	1	2.5
Diamond Core Drill Hole (DD)	3	327.0
Water Bore Hole	1	42.0
Auger Drill Hole	115	602.0
Rotary Air Blast Drill Hole (RAB)	4	88.0
Reverse Circulation Drill Hole (RC)	71	3,689.0
Total	**195**	**4,750.5**

Figure 14.8.1 P17N Zone Drill Hole Collars

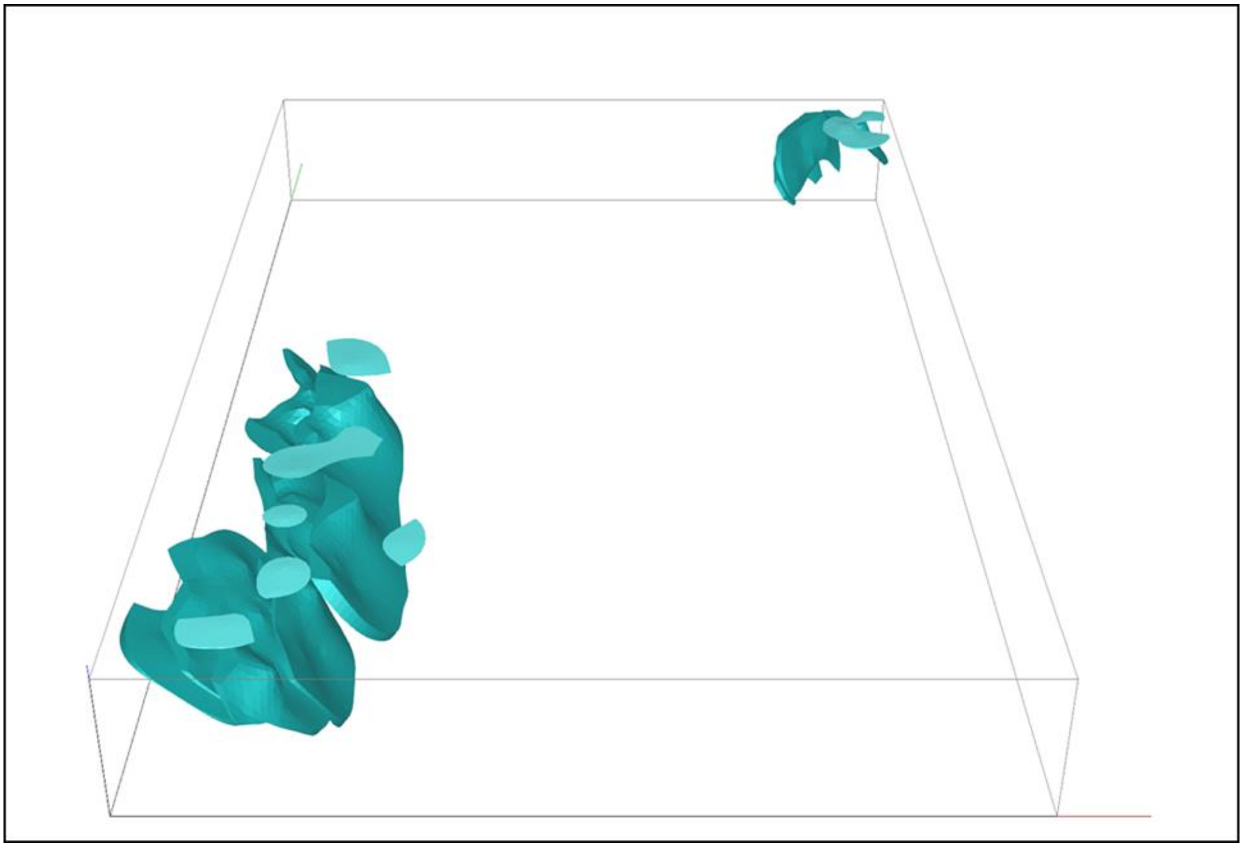


14.8.2 Exploratory Data Analysis

A total of eleven distinct mineralization domain wireframes were supplied by Orezone (Figure 14.8.2).

The mean nearest neighbour collar distance for DD and RC is 25.8 m, the average DD length is 109.0 m and the average RC drill hole length is 52.0 m. Summary statistics for the supplied assay data are provided below (Table 14.8.2).

Figure 14.8.2 Isometric View of P17N Zone Mineralization Domains



View looking north. Field of view is approximately 600 m across.

Table 14.8.2 P17N Summary Assay Statistics

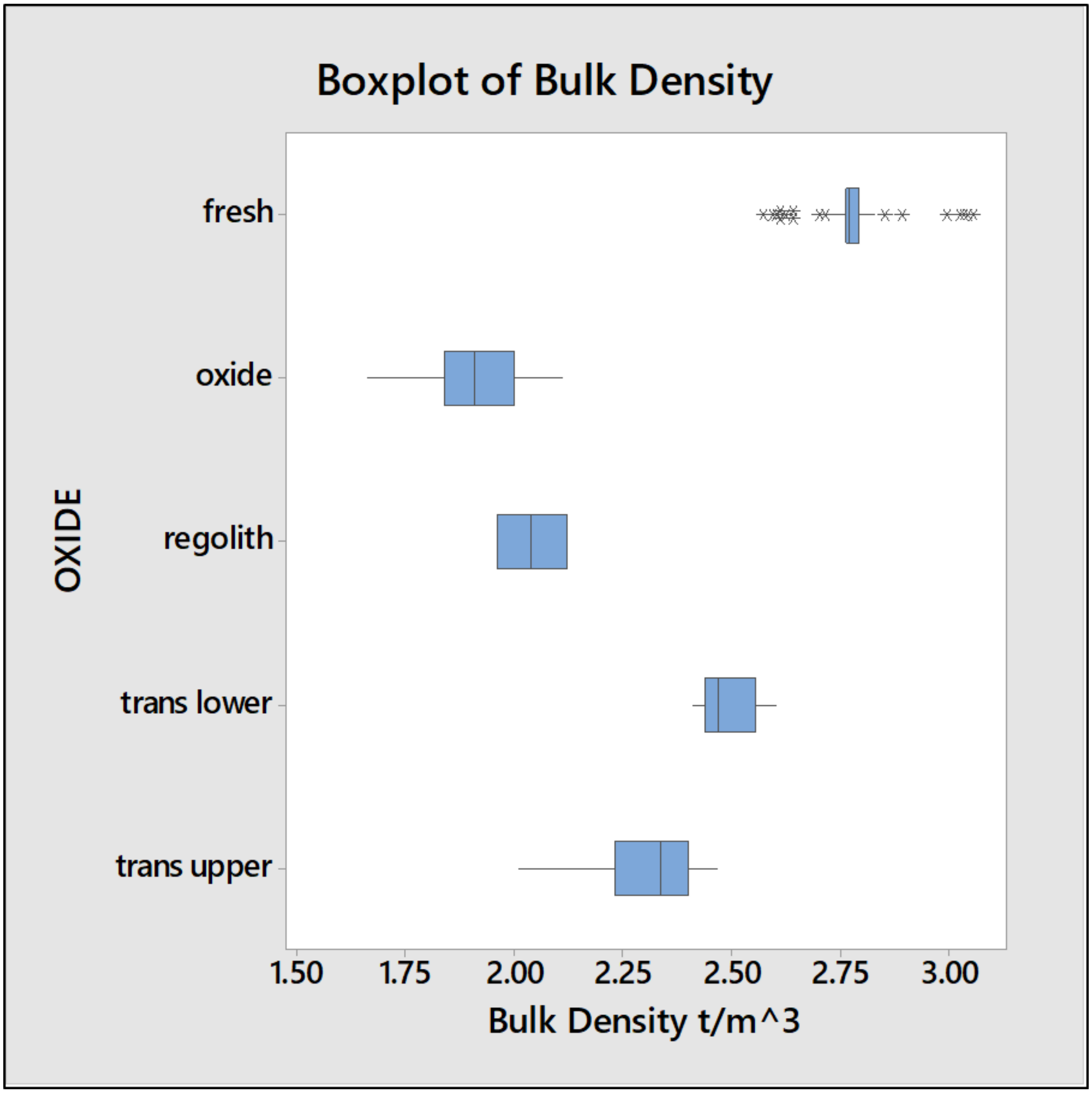
Domain	Count	Avg Length m	Min Length m	Max Length m	Avg Au g/t	Min Au g/t	Max Au g/t	Std Dev g/t	CoV
None	3,226	1.03	0.70	4.00	0.07	0.00	51.47	0.92	12.34
401	53	1.00	1.00	1.00	0.86	0.00	8.34	1.49	1.73
402	264	1.00	1.00	1.00	0.64	0.02	8.13	0.95	1.49
403	127	1.00	1.00	1.00	0.43	0.00	4.11	0.56	1.29
404	140	1.00	1.00	1.00	0.35	0.01	1.73	0.31	0.88
405	85	1.00	1.00	1.00	0.31	0.02	1.34	0.24	0.77
406	42	1.10	1.00	3.00	1.26	0.04	36.42	5.58	4.43
407	191	1.00	1.00	1.00	0.38	0.01	12.07	0.92	2.44
408	38	1.00	1.00	1.00	0.44	0.01	1.88	0.39	0.88
409	16	1.00	1.00	1.00	0.35	0.02	1.17	0.41	1.15
421	9	1.00	1.00	1.00	1.09	0.02	3.99	1.30	1.20
422	16	1.00	1.00	1.00	0.32	0.03	0.76	0.23	0.70
Total	**4,207**	**1.02**	**0.70**	**4.00**	**0.18**	**0.00**	**51.47**	**1.07**	**6.02**

The supplied database contains 177-point bulk density measurements from drill hole core, with values ranging from 1.66 to 3.05 t/m^3 (Table 14.8.3). Bulk density measurements were back-tagged by mineralization domain, lithology and oxidation state wireframes. Bulk density measurements display a range of values based on oxidation state (Figure 14.8.3).

Table 14.8.3 P17N Zone Summary Bulk Density Statistics

Unit	Count	Avg. Bulk Density t/m^3	Min. Bulk Density t/m^3	Max. Bulk Density t/m^3	Std Dev	CoV
Oxide	41	1.91	1.66	2.11	0.10	0.05
Transitional Upper	16	2.31	2.01	2.47	0.13	0.06
Transitional Lower	21	2.49	2.41	2.60	0.06	0.03
Fresh	97	2.77	2.57	3.05	0.08	0.03
Regolith	2	2.04	1.96	2.12	0.11	0.06
Total	**177**	**2.49**	**1.66**	**3.05**	**0.37**	**0.15**

Figure 14.8.3 Boxplot of P17N Zone Bulk Density by Oxide State



14.8.3 Block Model

An orthogonal block model was established with the block model limits selected to cover the extent of the mineralized structures, the potential open pit dimensions, and to reflect the general orientation of the mineralized zones (Table 14.8.4). The block model consists of separate variables for estimated grades, rock codes, bulk density, and classification attributes. Separate variables were coded with unique mineralization domain, lithology, and oxide state rock codes.

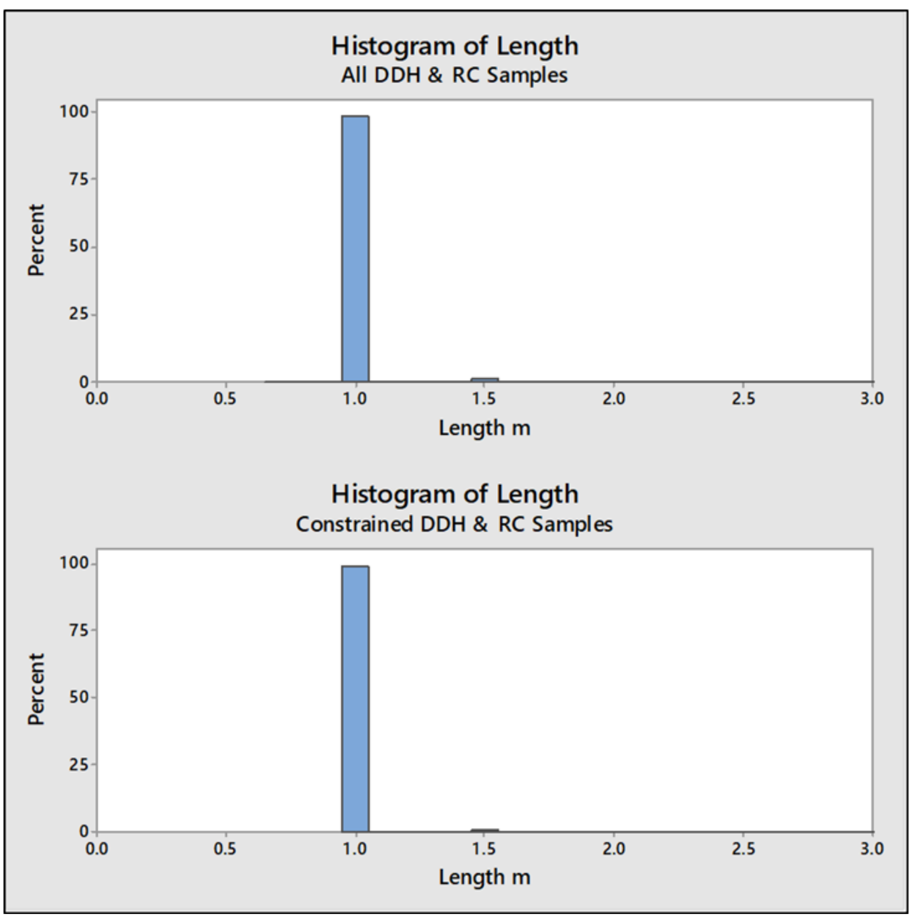
Table 14.8.4 P17N Zone Block Model Set-Up

Coordinate	Origin	Block Size m	Number of Blocks
Easting (X)	730,000	2.0	375
Northing (Y)	1,345,325	6.25	200
Elevation (max Z)	280	3.0	95

14.8.4 Compositing

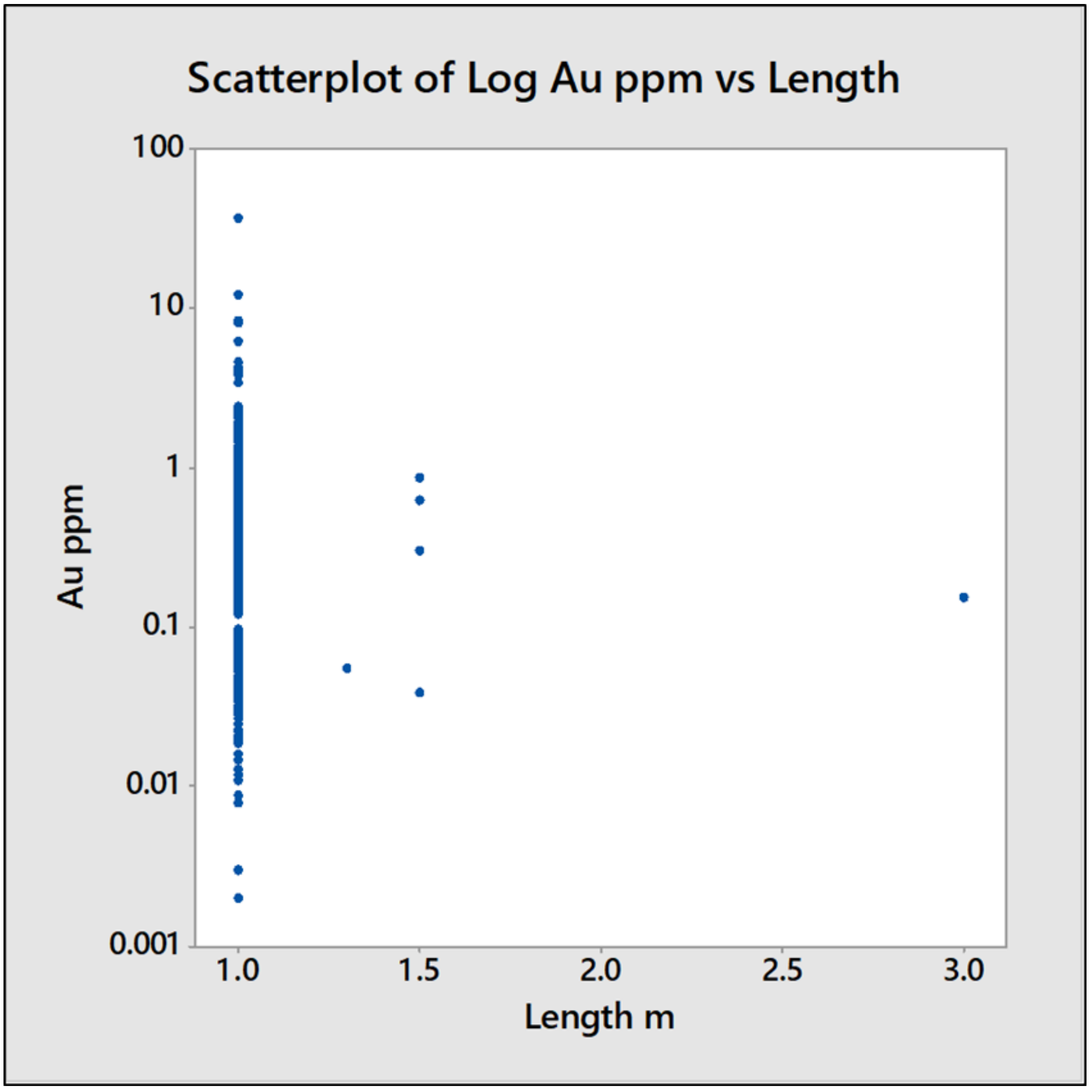
Assay sample lengths for DD and RC drill holes range from 0.70 to 4.00 m, with an average sample length of 1.01 m (Figure 14.8.4). The average sample length for DD assays is 1.13 m, and the average sample length for RC assays is 1.00 m. Constrained assay sample lengths within the defined mineralization domains range from 1.00 to 3.00 m, with an average sample length of 1.00 m. A total of 99% of the constrained assay sample lengths equal 1.00 m.

Figure 14.8.4 Histograms of P17N Zone Assay Sample Lengths



No correlation was observed between sample grade and sample length for the constrained assay samples (Figure 14.8.5).

Figure 14.8.5 Scatterplot of P17N Zone Assay Sample Lengths Versus Grade



Based on the predominance of 1.00 m sample lengths, all constrained assay samples were composited to this length in order to ensure equal sample support. Length-weighted composites were calculated within the defined mineralization domains. Missing sample intervals in the data were assigned a nominal background grade of 0.001 g/t. The compositing process started at the first point of intersection between the drill hole and the domain wireframe intersected and halted on exit from the wireframe. Downhole residual composites that were less than half the compositing length were discarded so not to introduce a short sample bias into the composite sample population. The wireframes that represent the interpreted mineralization domains were utilized to back-tag a rock code variable into the composite workspace. The composite data were then visually validated against the mineralization wireframes and extracted for analysis and grade estimation. Summary composite statistics are listed in Table 14.8.5.

Table 14.8.5 P17N Zone Summary Composite Statistics

Domain	Count	Average Au g/t	Minimum Au g/t	Maximum Au g/t	Std Dev	CoV
401	46	0.98	0.14	8.33	1.57	1.60
402	247	0.68	0.02	8.13	0.96	1.42
403	109	0.48	0.03	4.10	0.58	1.20
404	118	0.40	0.02	1.73	0.31	0.78
405	72	0.35	0.05	1.33	0.24	0.68
406	41	1.30	0.06	36.38	5.64	4.33
407	175	0.40	0.01	12.06	0.95	2.37
408	34	0.49	0.01	1.87	0.38	0.78
409	13	0.42	0.02	1.16	0.43	1.00
421	6	1.01	0.31	2.69	0.93	0.92
422	11	0.38	0.16	0.76	0.19	0.51
Total	**872**	**0.57**	**0.01**	**36.38**	**1.47**	**2.57**

14.8.5 Treatment of Extreme Values

Capping thresholds were determined by the decomposition of individual composite log-probability distributions, examination of box-plots and histograms, and reference to Tukey's rule of three times the interquartile range for extreme outliers (Tukey, 1977). Potential outliers are not markedly clustered in localized high-grade areas and sub-domaining is therefore not warranted. Composites are capped to the defined threshold prior to estimation. A total of four domains do not require capping (Table 14.8.6).

Table 14.8.6 P17N Zone Capping Thresholds and Summary Statistics

Domain	Count	Cap	Number Capped	Uncapped Mean	Capped Mean	Mean Above Cap	Change %
401	46	3.54	3	0.98	0.80	6.30	-23
402	247	2.42	6	0.68	0.61	5.41	-11
403	109	1.38	9	0.48	0.43	2.04	-12
404	118	1.44	1	0.40	0.39	1.73	-3
405	72	1.22	1	0.35	0.34	1.33	-3
406	41	1.52	2	1.30	0.41	19.91	-217
407	175	1.14	8	0.40	0.32	2.99	-25
408	34	n/a	0	0.49	0.49	n/a	0
409	13	n/a	0	0.42	0.42	n/a	0
421	6	n/a	0	1.01	1.01	n/a	0
422	11	n/a	0	0.38	0.38	n/a	0
Total	**872**	**---**	**30**	**0.57**	**0.47**	**---**	**-21**

14.8.6 Variography and Continuity Analysis

Three-dimensional continuity analyses (variography) were conducted on the domain coded uncapped composite data using a normal-scores transformation. In general, only poor semi-variograms for the individual mineralization domains were obtained, primarily due to the small number of points per domain, except for Domain 402. A semi-variogram for the southern domains (rock codes 401, 402, 403, 404, 405 and 407) was derived by consolidating the mineralization composite samples. The downhole variogram was viewed at a 1.0 m lag spacing (equivalent to the composite length) to assess the nugget variance contribution. Standardized isotropic spherical models were utilized to model the experimental semi variograms in normal score transformed space. Semi-variogram model ranges were checked and iteratively refined for each model relative to the overall nugget variance, and the back-transformed variance contributions were then calculated (Table 14.8.7).

Table 14.8.7 Back-Transformed Semi-Variograms

402	0.48 + 0.31SPH(3m) + 0.22SPH(21m)
South	0.66 + 0.22SPH(4m) + 0.12SPH(15m)

14.8.7 Mineral Resource Estimate

Bulk density was estimated by Inverse Distance Squared (ID2) anisotropic linear interpolation using a minimum of three and maximum of twelve composites within a 300 m diameter search envelope. Sample selection was restricted to a maximum of four composite samples from a single drill hole within the same oxide state domain. Bulk density estimation was performed by weathering unit within the weathered portion of the deposits, and by lithological domain within the fresh Birimian portion of the deposits. These weathering and lithological units define hard boundaries for bulk density estimation. For those areas of limited sample data, the mean value of the specific lithology type in the Fresh domain or within the weathering unit above the Fresh unit was applied. Orezone applies a reduction factor of 5.5% and 2.5% to individual samples in the oxide and transition zones, respectively, for the block model dry bulk density calculations. Bulk density block values were adjusted by these factors after estimation.

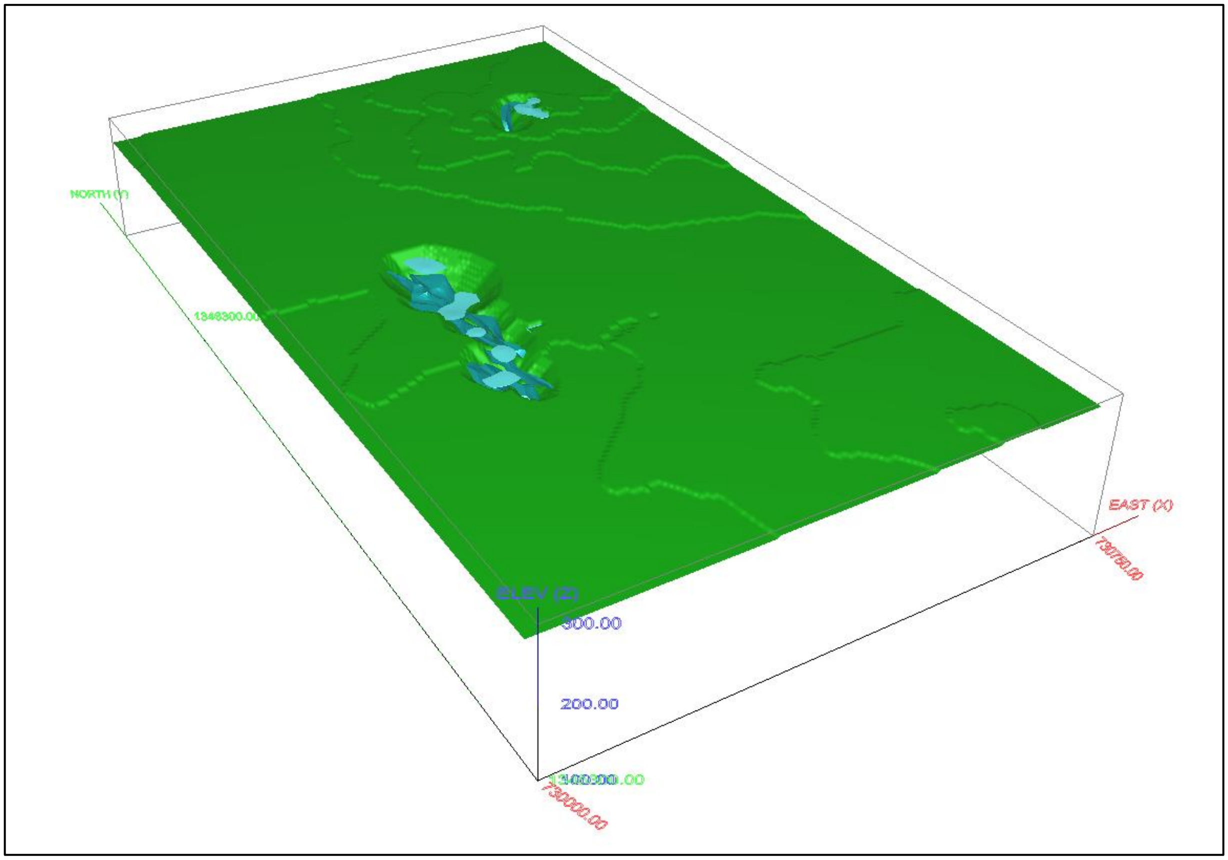
Block grades for gold were estimated by Inverse Distance Cubed (ID3) weighting of capped composites using a minimum of three and maximum of twelve composites within an extended 600 m x 300 m x 50 m search envelope. Sample selection selected the nearest composites to the block centroid and was restricted to a maximum of four composite samples from a single drill hole.

The orientations of the search ellipsoids were defined by Orezone geologists based on the local geology within each respective mineralization domain. The defined orientations were then converted to Gemcom™ format rotations. Ordinary Kriging (OK) and capped Nearest neighbour models (NN) were also generated using the same search parameters. Only composites derived from DD and RC drill holes were used for grade estimation. Search and grade estimation were constrained by the individual mineralization domains, which define hard boundaries for grade estimation.

The parameters used to define classification limits included spatial analysis, drill hole spacing, and the observed continuity of the mineralization. Mineral Resources were classified algorithmically based on the local drill hole spacing within each mineralization domain. All blocks within 25 m of three or more drill holes were classified as Measured, blocks within 50 m of three or more drill holes were classified as Indicated, and all additional estimated blocks were classified as Inferred.

Mineral Resources reported herein have been constrained within an optimized pit (Figure 14.8.6). The results from the optimized pit shell are utilized solely for the purpose of reporting Mineral Resources and include Measured, Indicated and Inferred Mineral Resources. Mineral Resources are reported at a cut-off grade of 0.25 g/t Au for regolith, oxide and upper transition lithologies, and 0.45 g/t Au for lower transition and fresh lithologies (Table 14.8.8). The Authors are reporting whole block volumes for only those blocks where the volume inclusion is ≥50%. Orezone applies a 0.95 factor to the P17N Zone oxide grade to account for artisanal mining.

Figure 14.8.6 Isometric View of Optimized P17N ZonePit Shell



View looking northeast.

Table 14.8.8 P17N Zone Mineral Resource Estimate*

Total	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	NA	305	0.66	7
Indicated	NA	150	0.78	4
Meas + Ind	NA	456	0.70	10
Inferred	NA	2	1.06	0

Regolith	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.25	0	0.45	0
Indicated	0.25	2	1.08	0
Meas + Ind	0.25	2	0.95	0
Inferred	0.25	0	0.00	0

Oxide	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.25	148	0.59	3
Indicated	0.25	45	0.51	1
Meas + Ind	0.25	192	0.57	4
Inferred	0.25	0	0.00	0
Trans Upper	**Au Cut-off g/t**	**Tonnes k**	**Au g/t**	**Au koz**
Measured	0.25	86	0.62	2
Indicated	0.25	11	0.65	0
Meas + Ind	0.25	97	0.62	2
Inferred	0.25	0	0.00	0
Trans Lower	**Au Cut-off g/t**	**Tonnes k**	**Au g/t**	**Au koz**
Measured	0.45	44	0.82	1
Indicated	0.45	13	0.74	0
Meas + Ind	0.45	57	0.80	2
Inferred	0.45	0	0.00	0
Fresh	**Au Cut-off g/t**	**Tonnes k**	**Au g/t**	**Au koz**
Measured	0.45	27	0.76	1
Indicated	0.45	80	0.93	2
Meas + Ind	0.45	107	0.89	3
Inferred	0.45	2	1.06	0

*Mineral Resources are inclusive of Reserves. Totals may differ due to rounding.

14.8.8 Validation

The P17N Zone block model was validated visually by the inspection of successive vertical cross-section lines in order to confirm that the block models correctly reflect the distribution of high grade and low-grade values.

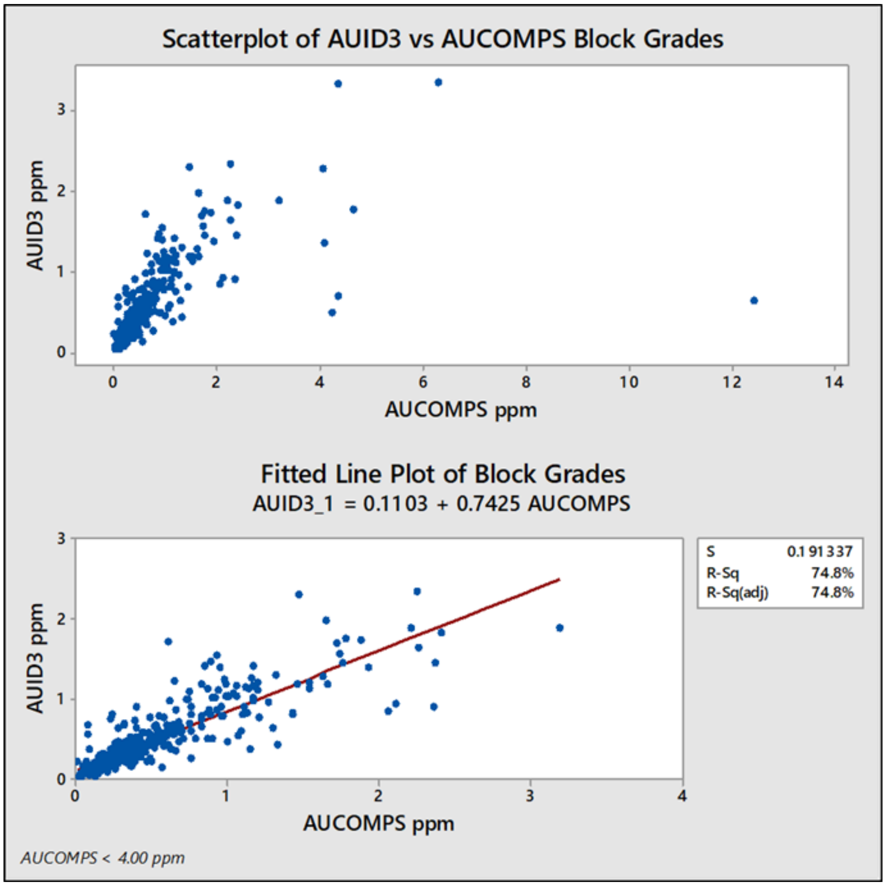
The average estimated block grades were compared to the average NN block estimate at a 0.001 g/t Au cut-off grouped by oxidation state domain (Table 14.8.9). The results fall within acceptable limits for linear estimation.

Table 14.8.9 Comparison of P17N Zone ID3 and NN Grade

Unit	Au ID g/t	Au NN g/t	Ratio ID/NN %
Oxide / Saprolite	0.48	0.48	0
Transition Upper	0.47	0.47	0
Transition Lower	0.44	0.43	2
Fresh	0.46	0.44	5
Regolith	0.57	0.55	4
Total	**0.46**	**0.45**	**2**

An additional validation check was completed by comparing the average grade of the composites in a block to the associated model block grade estimate. A total of eight outlier blocks with an average gold composite grade of 5.56 g/t Au have been reduced to an average grade of 1.74 g/t Au, with no other grade estimation issues noted. The results fall within acceptable limits for grade estimation (Figure 14.8.7).

Figure 14.8.7 Validation Plot Between P17N Zone Block Grades and Average Composite Grades



The volume estimated was also checked against the reported volume of the individual mineralization domains. Estimated volumes are based on a 0.001 g/t cut-off and partial block volumes (Table 14.8.10). The results fall within acceptable limits for estimation.

Table 14.8.10 P17N Zone Volume Reconciliation

Domain	Estimated Volume m^3	Reported Volume m^3	Ratio %
401	18,400	18,800	102
402	178,400	180,200	101
403	88,400	89,500	101
404	84,200	85,200	101
405	39,000	39,400	101

Domain	Estimated Volume m³	Reported Volume m³	Ratio %
406	15,000	15,200	101
407	143,000	144,300	101
408	23,200	23,500	101
409	5,100	5,200	102
421	2,000	2,100	105
422	3,600	3,800	106
Total	**600,300**	**607,100**	**101**

14.8.9 P17N Cut-Off Sensitivity

The sensitivity of the Mineral Resource block grades to changes in cut-off grade was evaluated by calculating the total Mineral Resource inventory at various cut-offs within the Mineral Resource pit (Table 14.8.11). The values presented are intended for the sole purpose of demonstrating the sensitivity of the Mineral Resource with respect to the cut-off grade.

Table 14.8.11 P17N Grade Tonnage Sensitivity Table

OXIDE MEASURED + INDICATED				OXIDE INFERRED		
Au Cut-Off g/t	Tonnes k	Au g/t		Au Cut-Off g/t	Tonnes k	Au g/t
1.00	28.7	1.69		1.00	0	0
0.95	32.3	1.62		0.95	0	0
0.90	36.5	1.54		0.90	0	0
0.85	41.3	1.46		0.85	0	0
0.80	47.0	1.38		0.80	0	0
0.75	53.9	1.31		0.75	0	0
0.70	62.4	1.23		0.70	0	0
0.65	72.1	1.15		0.65	0	0
0.60	84.1	1.08		0.60	0	0
0.55	98.9	1.00		0.55	0	0
0.50	117.2	0.93		0.50	0	0
0.45	140.0	0.86		0.45	0	0
0.40	168.4	0.78		0.40	0	0
0.35	203.3	0.71		0.35	0	0
0.30	245.3	0.65		0.30	0	0
0.25	291.8	0.59		0.25	0	0

OXIDE MEASURED + INDICATED				**OXIDE INFERRED**		
Au Cut-Off	**Tonnes**	**Au**		**Au Cut-Off**	**Tonnes**	**Au**
g/t	**k**	**g/t**		**g/t**	**k**	**g/t**
0.20	334.0	0.54		0.20	0	0

SULPHIDE MEASURED + INDICATED				**SULPHIDE INFERRED**		
Au Cut-Off	**Tonnes**	**Au**		**Au Cut-Off**	**Tonnes**	**Au**
g/t	**k**	**g/t**		**g/t**	**k**	**g/t**
1.00	40.5	1.59		1.00	0.3	1.99
0.95	45.2	1.52		0.95	0.4	1.91
0.90	50.7	1.45		0.90	0.4	1.82
0.85	57.2	1.38		0.85	0.5	1.72
0.80	65.0	1.31		0.80	0.6	1.64
0.75	73.7	1.24		0.75	0.6	1.55
0.70	84.1	1.17		0.70	0.7	1.46
0.65	95.8	1.11		0.65	0.9	1.37
0.60	109.3	1.04		0.60	1.0	1.29
0.55	125.0	0.98		0.55	1.2	1.21
0.50	143.1	0.92		0.50	1.4	1.12
0.45	163.8	0.86		0.45	1.6	1.06
0.40	186.5	0.80		0.40	1.8	1.00
0.35	212.0	0.75		0.35	1.9	0.94
0.30	239.7	0.70		0.30	2.1	0.90
0.25	268.0	0.65		0.25	2.3	0.87
0.20	291.6	0.62		0.20	2.4	0.84

Oxide includes Regolith, Oxide and Transitional Upper units.
Sulphide includes Transitional Lower and Fresh units.
Totals may differ due to rounding.

14.9 Bomboré P17 Zone

14.9.1 Data Supplied

The P17 Zone model was developed by Orezone and reviewed by the Authors. Topography, mineralization, lithology and oxidation state three-dimensional wireframes were created using Leapfrog Geo™ 2022.1.1. Drill hole distances are reported in metres and gold grades are reported as g/t. The drill hole collar coordinates were provided in the WGS84 Zone 30 UTM coordinate reference system. The supplied drill hole database for the P17 Zone contains 227 unique drill hole collar records (Table 14.9.1 and Figure 14.9.1).

Table 14.9.1 P17 Zone Drill Hole Database Summary

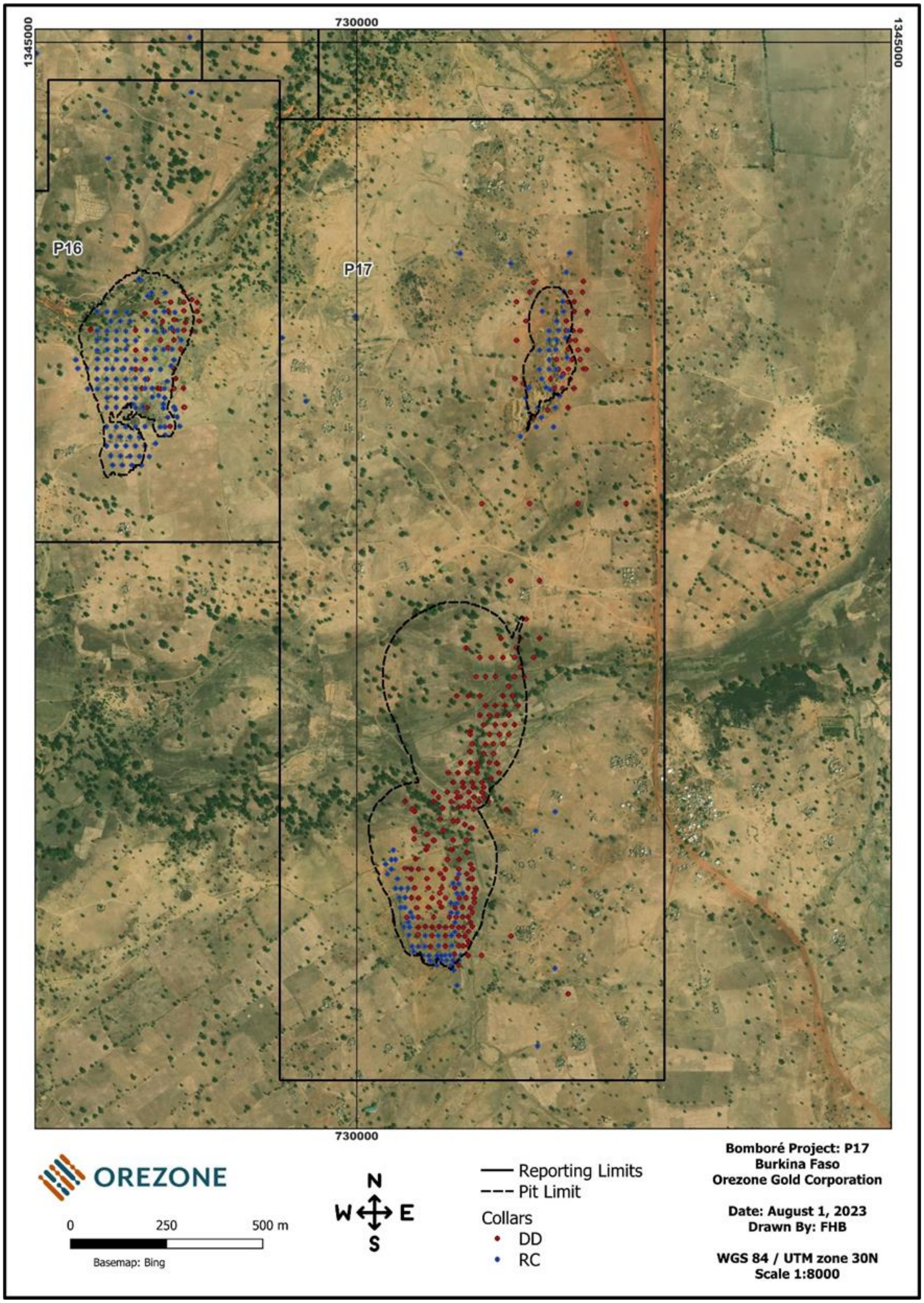
Type	Count	Metres
Auger	174	781.0
Channel	11	28.4
Diamond Core Drill Hole (DD)	242	38,910.5
Rotary-Air-Blast (RAB)	35	860
Trench	5	25.3
Reverse Circulation Drill Hole (RC)	108	4,927.0
Total	**578**	**45,532.2**

Industry standard validation checks were carried out on the supplied databases, and minor corrections made where necessary. The Authors typically validate a Mineral Resource database by checking for inconsistencies in naming conventions or analytical units, duplicate entries, interval, length, or distance values less than or equal to zero, blank or zero-value assay results, out-of-sequence intervals, intervals or distances greater than the reported drill hole length, inappropriate collar locations, and missing interval and coordinate fields.

No significant errors were noted with the supplied databases. The Authors consider that the drill hole database supplied is suitable for Mineral Resource estimation.

The drill hole data were imported into a SurpacTM format Access database, and imported wireframes were assigned a unique rock code. The supplied wireframes were used to back-tag the assay, bulk density and composite tables with mineralization domain, oxide state and lithology codes.

Figure 14.9.1 P17 Zone Drill Hole Plan View

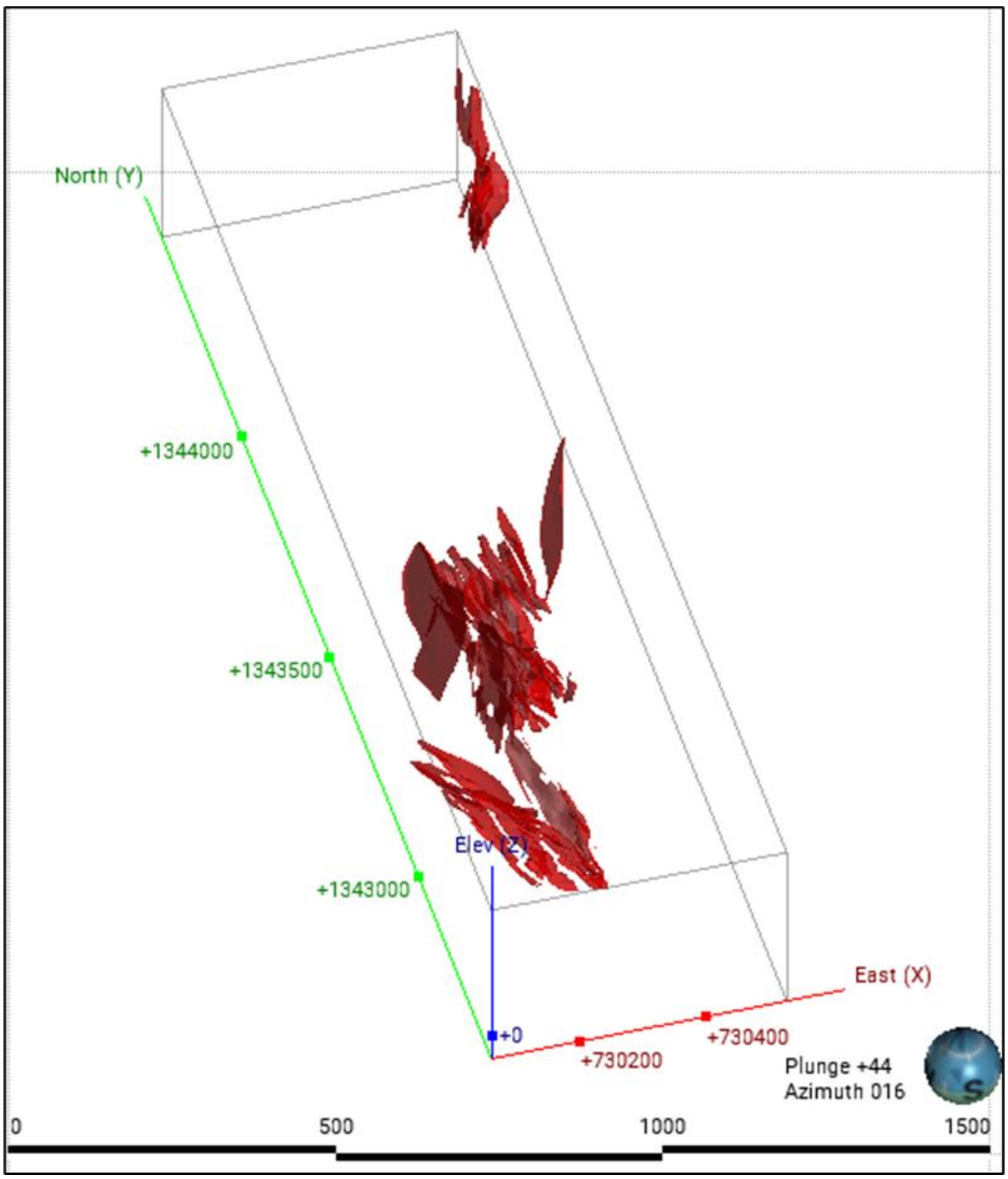


14.9.2 Exploratory Data Analysis

A total of 30 distinct mineralization domain wireframes were supplied (Figure 14.9.2).

The average nearest-neighbour collar distance (for DD and RC) is 25.3 m, the average DD length is 160.8 m, and the average RC drill hole length is 45.6 m. Summary statistics for the supplied assay data are provided below (Table 14.9.2).

Figure 14.9.2 Isometric View of P17 Zone Mineralization Domains



View looking along azimuth 16 degrees.

Table 14.9.2 P17 Zone Assay Summary Statistics

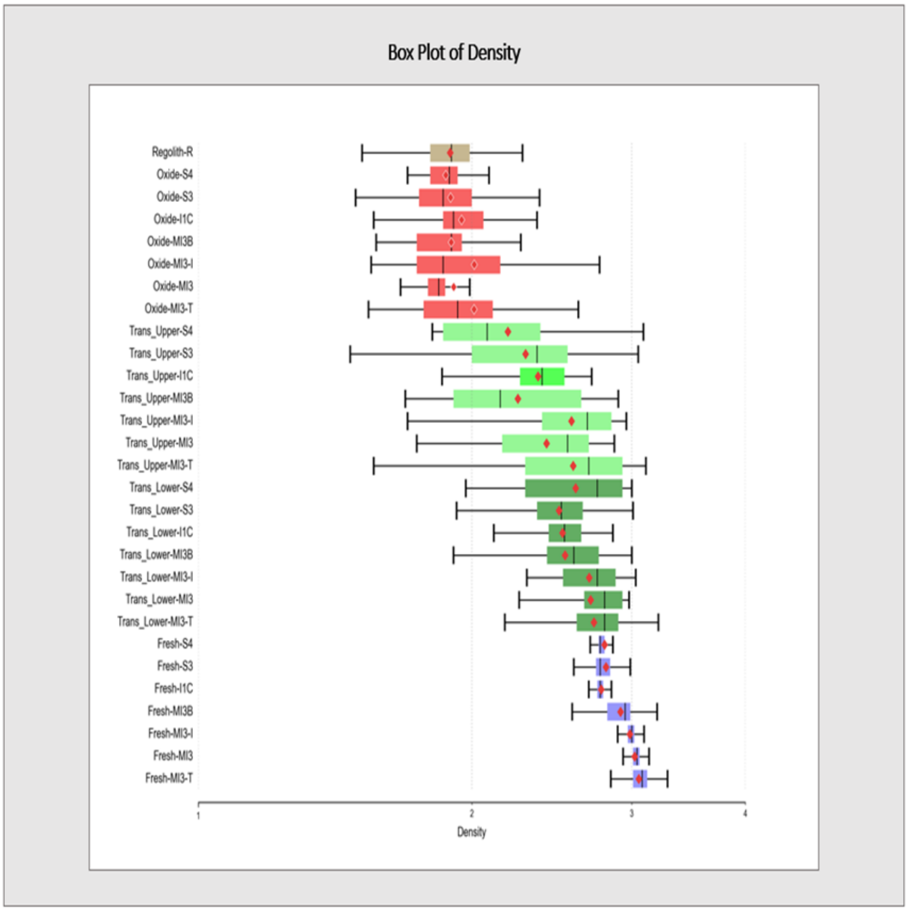
Domain	Count	Avg Length m	Min Length m	Max Length m	Avg Au g/t	Min Au g/t	Max Au g/t	Std Dev g/t	CoV
None	36,440	1.04	0.10	6.50	0.03	0.00	26.81	0.27	0.07
MB_01	35	1.01	0.85	1.30	0.91	0.02	3.51	0.91	1.00
MB_02	289	1.02	0.70	1.60	2.35	0.00	17.55	2.41	1.03
MB_03	507	1.02	0.70	2.00	2.41	0.00	38.35	3.30	1.37
MB_04	174	1.01	0.75	1.50	2.89	0.00	14.10	2.98	1.03
MB_05	290	1.03	0.70	1.55	2.43	0.00	17.98	2.76	1.14
MB_06	76	1.03	0.70	1.60	1.56	0.01	5.84	1.49	0.96
MB_07	306	1.03	0.65	2.00	1.59	0.00	19.75	2.28	1.44
MB_08	53	1.06	0.95	1.70	0.83	0.02	5.19	0.84	1.01
MB_09	194	1.01	0.65	1.50	0.91	0.00	8.88	1.43	1.57
MB_10	55	1.00	0.80	1.10	0.62	0.00	2.32	0.68	1.10
MB_11	223	1.01	0.70	1.55	0.92	0.00	5.75	1.13	1.23
MB_12	308	1.01	0.70	1.40	1.55	0.00	16.59	2.37	1.53
MB_13	321	1.00	0.50	1.55	1.55	0.00	90.92	5.34	3.44
MB_14	342	1.01	0.75	1.60	1.08	0.00	13.10	1.54	1.43
MB_16	805	1.01	0.70	3.00	1.44	0.00	62.25	2.83	1.97
MB_17	178	1.02	0.80	1.80	1.07	0.00	23.85	2.11	1.97
MB_18	23	1.02	1.00	1.40	0.53	0.02	1.45	0.44	0.83
MB_19	21	1.10	1.00	1.50	1.02	0.01	3.37	1.04	1.02
MB_20	69	1.01	0.80	1.25	0.87	0.00	4.71	1.00	1.14
MB_21	99	1.01	0.90	1.50	1.31	0.00	25.59	2.90	2.21
MB_22	254	1.00	0.80	1.30	1.34	0.00	25.15	2.66	1.98
MB_23	429	1.02	0.50	3.00	0.97	0.00	17.82	1.80	1.86
MB_24	180	1.00	0.80	1.25	0.76	0.00	7.42	1.03	1.37
MB_25	212	1.01	0.75	2.00	0.67	0.00	17.99	1.48	2.21
MB_26	56	1.03	0.80	1.50	0.49	0.01	4.09	0.68	1.40
MB_27	65	1.03	0.75	1.50	0.46	0.00	3.46	0.65	1.41
MB_31	10	1.00	1.00	1.00	1.19	0.02	6.45	1.93	1.61
MB_15	346	1.00	0.80	1.50	1.55	0.00	41.63	3.03	1.96
MB_29	57	1.00	1.00	1.00	1.15	0.00	7.05	1.69	1.46
MB_30	813	1.05	0.80	2.50	0.72	0.00	14.94	1.45	2.00

The supplied database contains 24,576 point bulk density measurements from drill hole core, with values ranging from 1.31 to 3.99 t/m^3 (Table 14.9.3). Bulk density measurements were back-tagged by mineralization domain, lithology and oxidation state wireframes. Bulk density measurements display a differing range of values based on oxidation state (Figure 14.9.3).

Table 14.9.3 P17 Zone Bulk Density Summary Statistics

Unit	Count	Avg Bulk Density t/m^3	Min Bulk Density t/m^3	Max Bulk Density t/m^3	Std Dev	CoV
Regolith	395	1.89	1.50	2.26	0.14	0.02
Oxide	340	1.93	1.49	2.97	0.26	0.02
Transitional Upper	364	2.42	1.47	3.11	0.37	0.14
Transitional Lower	542	2.60	1.79	3.21	0.26	0.07
Fresh	22,935	2.92	2.25	3.57	0.16	0.02
Total	**24,576**	**2.88**	**1.47**	**3.57**	**0.25**	**0.06**

Figure 14.9.3 Boxplot of P17 Zone Bulk Density (t/m³) by Oxide / Lithology State



14.9.3 Block Model

An orthogonal block model was established using SurpacTM 2022 software with the block model limits selected so as to cover the extent of the mineralized structures, the potential open pit dimensions, and to reflect the general orientation of the mineralized zones (Table 14.9.4).

The block model consists of separate variables for estimated grades, volume percent inclusion, rock codes, bulk density, and classification attributes. Separate variables were coded with unique mineralization domain, lithology, and oxide state rock codes.

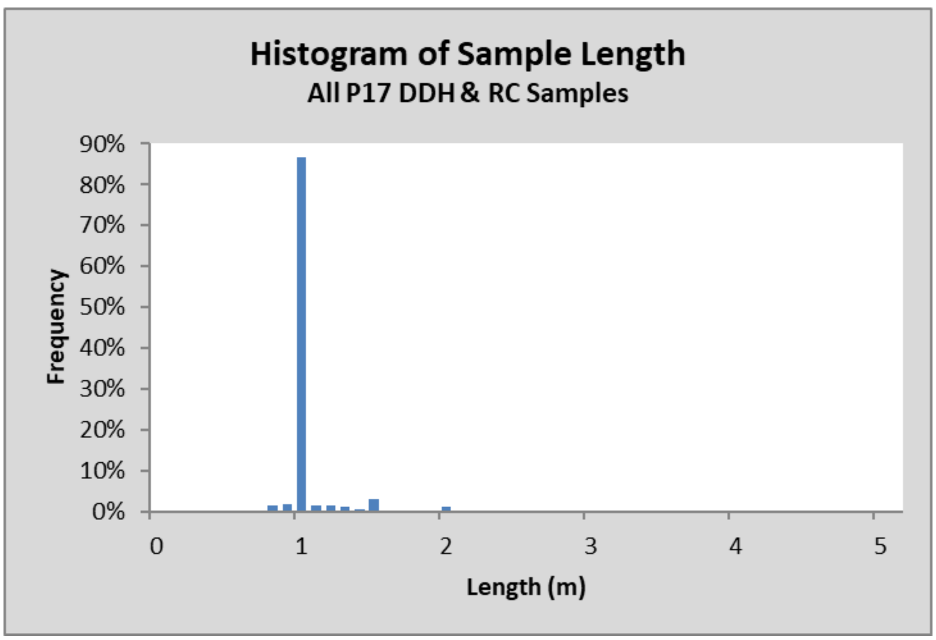
Table 14.9.4 P17 Zone Block Model Setup

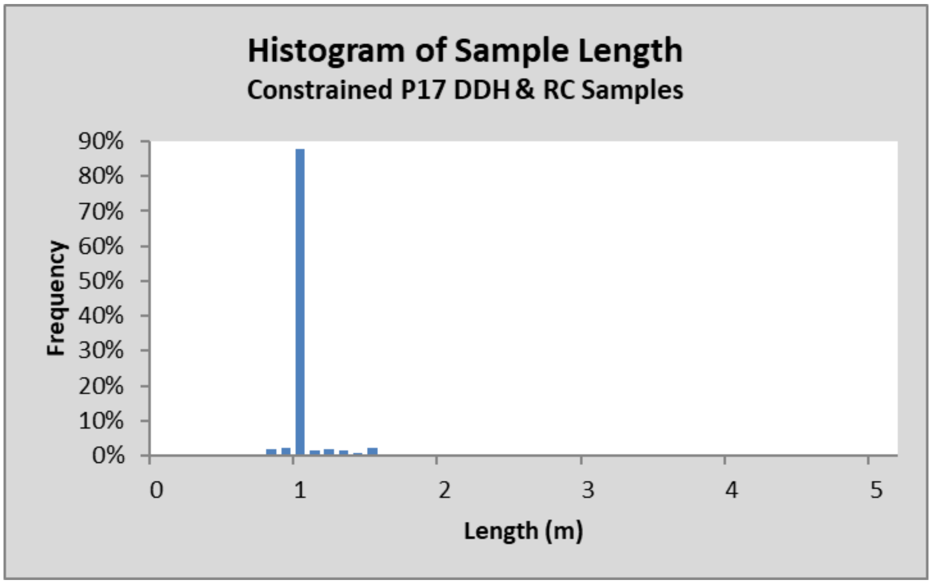
Coordinate	Minimum	Maximum	Block Size m	Number of Blocks
Easting (X)	729,800	730,800.5	1.500	667
Northing (Y)	1,342,300	1,344,800	3.125	800
Elevation (min Z)	-165	285	3.000	150

14.9.4 Compositing

Assay sample lengths range from 0.10 to 6.50 m, with an average sample length of 1.04 m (Figure 14.9.4). Constrained assay samples lengths within the defined mineralization domains range from 0.50 to 3.00 m, with an average sample length of 1.02 m. A total of 88% of the constrained assay sample lengths equal 1.00 m.

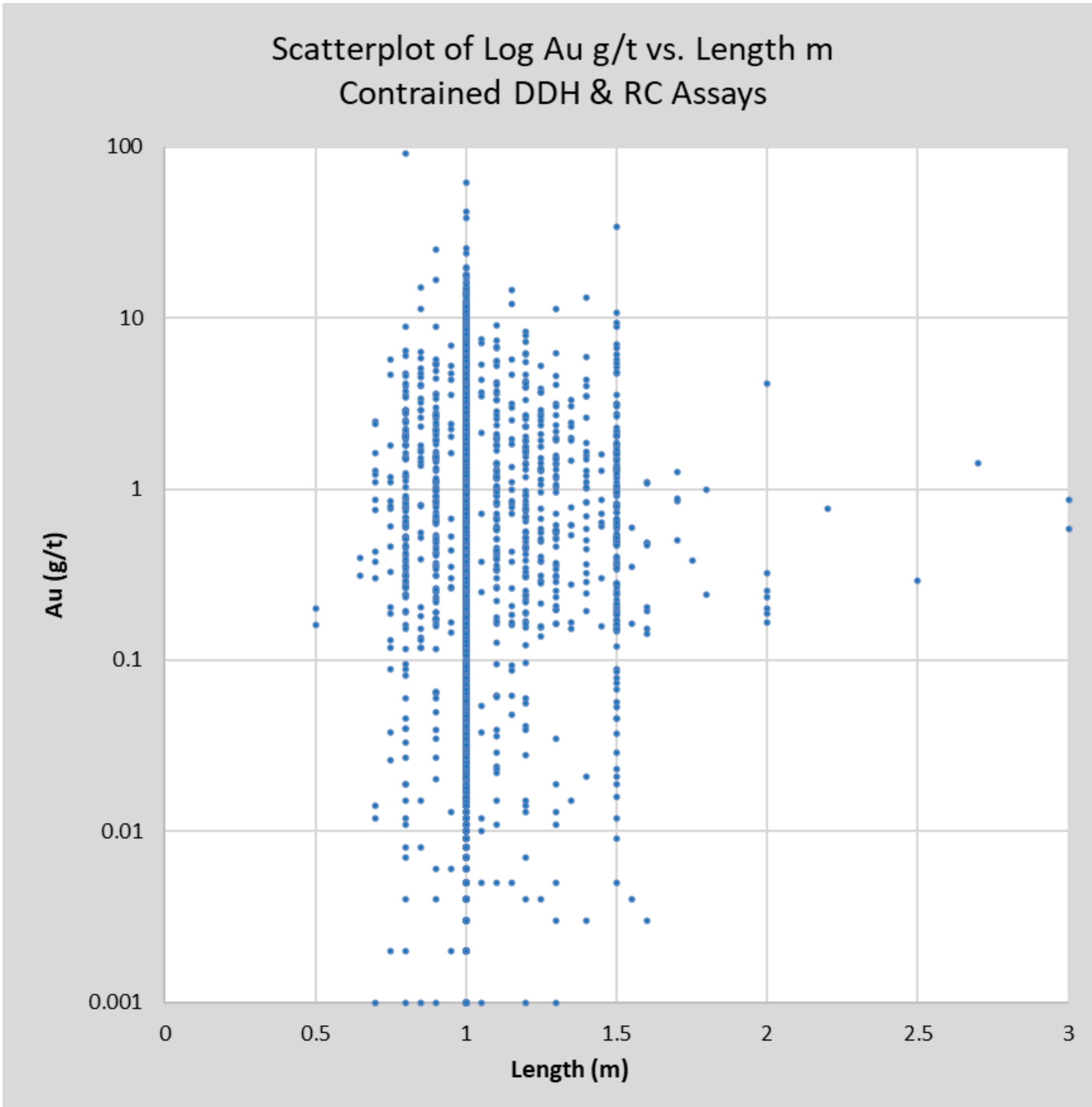
Figure 14.9.4 Histograms of P17 Zone Assay Sample Lengths





No correlation was observed between sample grade and sample length for the constrained assay samples (Figure 14.9.5).

Figure 14.9.5 Scatterplot of P17 Zone Assay Sample Lengths vs Grade



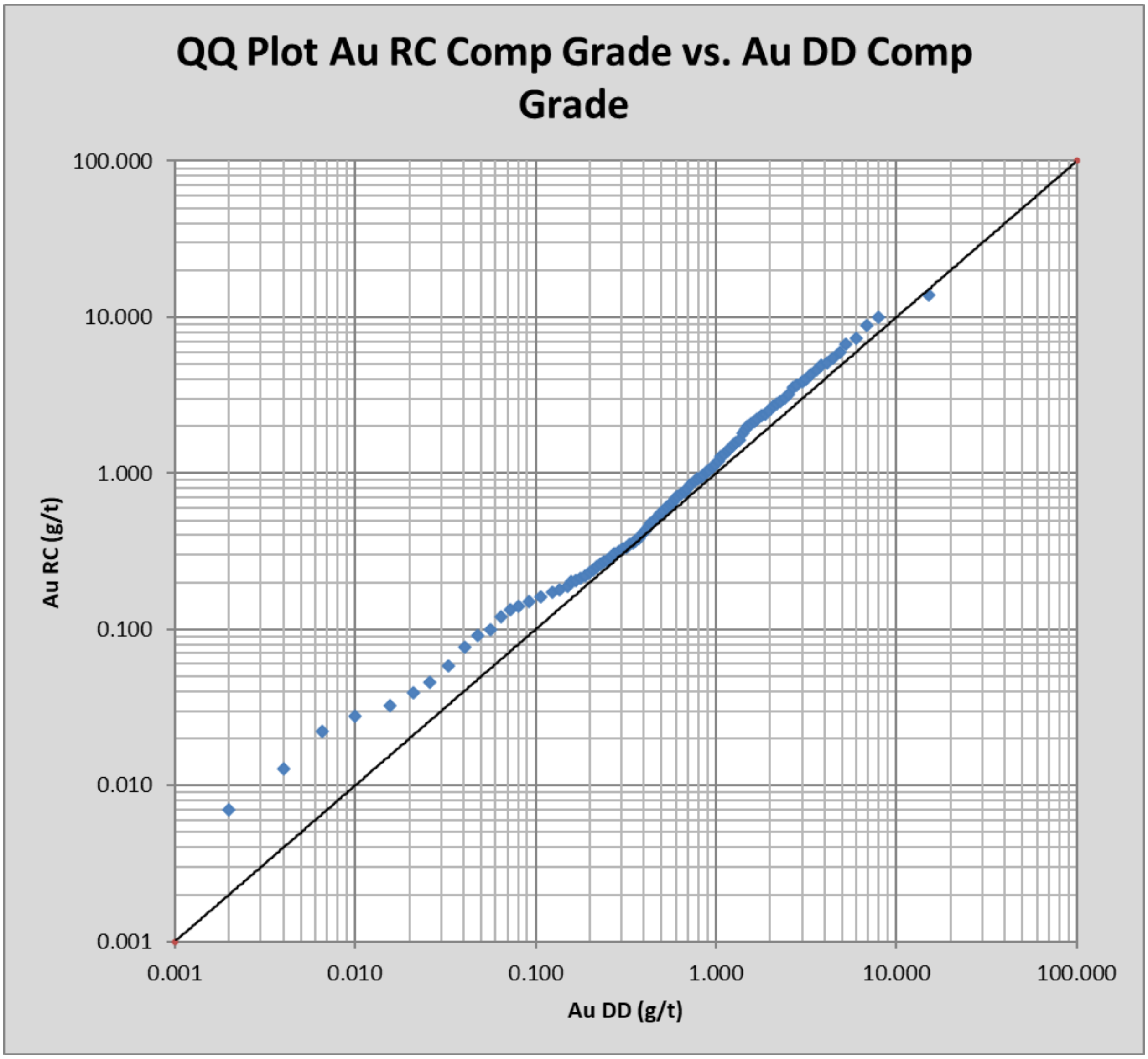
Based on the predominance of 1.00 m sample lengths, Orezone selected 1.0 m downhole composites. The downhole compositing process used a 'best fit' approach that results in composites of slightly variable length, but of equal contiguous length within a given drill hole intersection, ensuring the composite length is as close as possible to the nominated 1 m composite length. Length-weighted composites were calculated within the defined mineralization domains. Missing sample intervals in the data were ignored and treated as null values during the compositing process. The compositing process started at the first point of intersection between the drill hole and the domain wireframe intersected and halted upon exit from the wireframe. The wireframes that represent the interpreted mineralization domains were used to back-tag a rock code variable into the composite workspace. The composite data were then visually validated against the mineralization wireframes and extracted for analysis and grade estimation. Summary composite statistics are listed in Table 14.9.5.

Table 14.9.5 P17 Zone Composite Summary Statistics

Domain	Count	Average Au g/t	Minimum Au g/t	Maximum Au g/t	Std Dev	CoV
101	80	1.426	0.004	7.227	1.803	1.264
102	123	1.803	0.002	90.648	8.327	4.618
103	245	1.017	0.001	10.870	1.572	1.546
104	58	0.706	0.002	4.358	0.924	1.309
105	54	0.841	0.001	6.082	1.135	1.350
106	124	0.931	0.001	11.775	1.643	1.765
107	78	0.959	0.001	4.365	1.033	1.077
108	62	0.584	0.001	4.080	0.845	1.447
109	27	0.303	0.009	1.491	0.310	1.023
110	12	1.226	0.217	3.080	0.880	0.718
111	37	1.698	0.001	23.828	3.993	2.352
201	41	2.286	0.010	11.397	2.518	1.101
202	339	2.086	0.001	12.574	2.215	1.062
203	368	2.442	0.001	34.218	2.904	1.189
204	68	1.083	0.060	5.191	0.953	0.880
205	286	1.596	0.002	16.962	2.158	1.352
206	235	2.394	0.001	13.111	2.714	1.134
207	273	2.420	0.001	15.496	2.644	1.093
209	69	1.723	0.204	7.121	1.527	0.886
211	21	0.705	0.179	2.067	0.483	0.685
301	24	0.339	0.001	1.163	0.267	0.786
302	11	0.204	0.001	0.352	0.125	0.613
Total	**2,635**	**1.738**	**0.001**	**90.648**	**2.875**	**1.654**

Examination of the RC and DD composite grade distributions indicates that the P17 Zone RC grades are higher than the DD grades, mostly below 0.2 g/t and above 1.3 g/t (Figure 14.9.6). The RC meterage represents approximately 16% of the total constrained composites available but is primarily concentrated around the southernmost and northernmost portions of the model, which may introduce a bias in the data. A correction factor was not applied to the RC drilling results.

Figure 14.9.6 QQ plot of P17 Zone RC Composite Grades Versus DD Composite Grades



14.9.5 Treatment of Extreme Values

Capping thresholds were determined for each of the mineralized domains prior to compositing of the assay data using histograms with cumulative frequency, normal distribution, log-probability plots, and graphical inspection of the spatial grade distribution. Potential outliers are not markedly clustered in localized high-grade areas and sub-domaining is therefore not warranted. Raw assays are capped to the defined threshold prior to compositing within the mineralization domains. A total of four domains do not require capping (Table 14.9.6). The average capped assay grade is 5% lower than the raw assay grade.

Table 14.9.6 P17 Zone Capping Thresholds and Summary Statistics

Domain	Count	Cap g/t	Number Capped	Uncapped Mean g/t	Capped Mean g/t	Mean Above Cap g/t	Change %
MB_01	35	3.51	1	0.912	0.912	3.512	0
MB_02	289	12.00	1	2.346	2.327	17.553	-1
MB_03	507	15.00	3	2.408	2.315	30.660	-4
MB_04	174	14.10	0	2.889	2.889		0
MB_05	290	14.00	1	2.430	2.416	17.975	-1
MB_06	76	5.84	0	1.559	1.559		0
MB_07	306	8.00	6	1.586	1.480	13.399	-7
MB_08	53	2.50	1	0.830	0.779	5.194	-6
MB_09	194	6.00	3	0.910	0.878	8.125	-4
MB_10	55	2.32	0	0.617	0.617		0
MB_11	223	5.75	1	0.921	0.921	5.750	0
MB_12	308	10.00	6	1.548	1.479	13.543	-4
MB_13	321	10.00	3	1.554	1.271	40.237	-18
MB_14	342	7.00	4	1.078	1.040	10.195	-3
MB_16	805	14.00	1	1.440	1.380	62.254	-4
MB_17	178	5.00	3	1.069	0.931	13.152	-13
MB_18	23	1.45	1	0.530	0.530	1.454	0
MB_19	21	3.37	0	1.018	1.018		0
MB_20	69	3.00	3	0.874	0.822	4.185	-6
MB_21	99	7.00	2	1.312	1.102	17.390	-16
MB_22	254	8.00	6	1.344	1.171	15.343	-13
MB_23	429	7.00	6	0.967	0.894	12.230	-8
MB_24	180	4.50	1	0.755	0.739	7.423	-2
MB_25	212	5.00	3	0.670	0.598	10.108	-11

Domain	Count	Cap g/t	Number Capped	Uncapped Mean g/t	Capped Mean g/t	Mean Above Cap g/t	Change %
MB_26	56	2.00	2	0.488	0.446	3.165	-9
MB_27	65	3.46	1	0.458	0.458	3.463	0
MB_31	10	1.50	1	1.194	0.699	6.448	-41
MB_15	346	12.00	5	1.546	1.447	18.833	-6
MB_29	57	5.00	3	1.153	1.077	6.427	-7
MB_30	813	6.00	11	0.725	0.669	10.160	-8
Total	**6,790**		**79**	**1.386**	**1.312**	**13.729**	**-5**

14.9.6 Variography & Continuity Analysis

Orezone performed three-dimensional continuity analyses (variography) within the mineralized sub-domains to determine appropriate estimation inputs to the interpolation process. The variogram modelling process followed by Orezone involves the following steps:

- Calculate and model the downhole variogram to characterize the Nugget Effect.
- Calculate a fan of variograms within the plane of greatest continuity to identify the direction of maximum continuity within the plane.
- Model the variogram in the direction of maximum continuity and the orthogonal directions.

Variogram parameters were derived for the better sampled mineralization sub-domains that is, MB_03, MB_16, and MB_30. Acceptable variogram modelling for the sparsely sampled sub-domains was not possible, and the variography used to estimate these sub-domains has been adopted from adjacent mineralized domains, but the directions and anisotropy ratios have been tailored to best suit the geometry of the each of the sub-domains.

Better variogram ranges from the above-mentioned sampled mineralized domains were respectively 34, 42 and 55 m, indicating that the maximum spatial continuity is greater than the average drill hole spacing.

14.9.7 Estimation & Classification

The bulk density data were extracted from the drill hole database and were grouped according to their host rock lithology within the Fresh weathering unit, or within their host weathering unit above the Fresh weathering unit. Separate capping values were determined for each lithology type and oxidation state. The bulk density values for each block were then estimated using the capped bulk density values and the Inverse Distance Squared (ID2) anisotropic linear interpolation using a minimum of three and maximum of 15 composites within a 100 m diameter search envelope. Sample selection was restricted to a maximum of three composite samples from a single drill hole. Bulk density estimation was performed by weathering unit within the weathered portion of the deposits, and by lithological domain within the fresh Birimian portion of the deposits. These weathering and lithological units define hard boundaries for bulk density estimation. For those areas of limited sample data, the mean value of the specific lithology type in the Fresh domain or within the weathering unit above the Fresh unit was applied.

Orezone applies a reduction factor of 5.5% and 2.5% to individual samples in the oxide and transition zones, respectively, for the dry block model bulk density calculations. Bulk density block values were adjusted by these factors after estimation.

Block grades for gold were estimated by Inverse Distance Cubed (ID3) weighting of capped composites using a minimum of three and maximum of 15 composites. Sample selection was restricted to a maximum of three composite samples from a single drill hole.

The orientation of the search ellipsoids was defined by Orezone geologists based on the local geology within each respective mineralization domain. The defined orientations were then converted to SurpacTM format rotations. Ordinary Kriging (OK) and capped Nearest Neighbour models (NN) were also generated using the same search parameters. Only composites derived from DD and RC drill holes were used for grade estimation. Search and grade estimation were constrained by the individual mineralization domains, which define hard boundaries for grade estimation.

The parameters used to define classification limits included spatial analysis, drill hole spacing distance from nearest grade composite, and the observed continuity of the mineralization. Mineral Resources were classified algorithmically as follows:

Measured Mineral Resources are defined by:

- A drill spacing of less than 25 m x 25 m.
- Average True Distance to Samples: 0 to 25 m.
- Three or more drill holes.
- Estimation Pass 1.

Indicated Mineral Resources are defined by:

- A drill spacing of less than 50 m x 50 m.
- Average True Distance to Samples: 25 to 50 m.
- Two or more drill holes.

Inferred Mineral Resources include all estimated mineralization defined by:

- A drill spacing greater than 50 m x 50 m.
- Average True Distance to samples: 50 to 100 m.
- Two or more drill holes.

All other mineralization remained unclassified.

14.9.8 Mineral Resource Estimate

Mineral Resources reported herein have been constrained within an optimized pit shell (Figure 14.9.7). The results from the optimized pit shell are used solely for the purpose of reporting Mineral Resources and include Measured, Indicated and Inferred Mineral Resources. Orezone reports whole block volumes for only those blocks where the block centroid lies within the controlling wireframe. Orezone applies a 0.66 factor to the P17 oxide grade to account for artisanal mining.

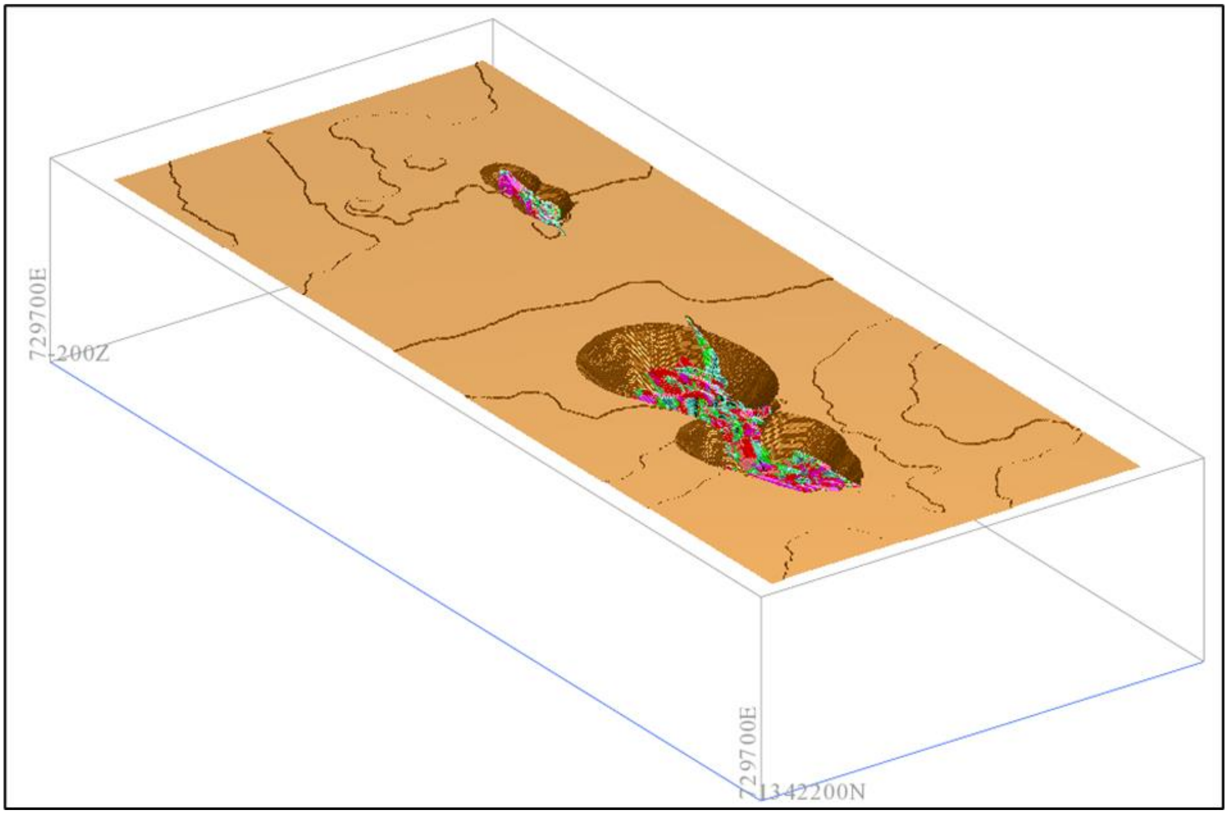
Mineral Resources are reported based on the cut-offs listed in Table 14.9.7 and a gold price of USD 1,700 per ounce.

Table 14.9.7 Cut-Offs

Unit	Au g/t
Regolith	0.250
Oxide	0.250
Trans Upper	0.250
Trans Lower	0.450
Fresh	0.450

The Mineral Resources have an effective date of 28 March (Table 14.9.8).

Figure 14.9.7 Isometric Plot of Pa7 Zone Optimized Pit Shell



View looking northeast.

Table 14.9.8 P17 Zone Mineral Resource Estimate*

Total	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	NA	988	0.00	79
Indicated	NA	4637	1.51	225
Meas + Ind	NA	5625	1.68	304
Inferred	NA	279	1.48	13

Regolith	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.25	0	0.00	0
Indicated	0.25	0	0.00	0
Meas + Ind	0.25	0	0.00	0
Inferred	0.25	0	0.00	0

Oxide	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.25	1	0.57	0
Indicated	0.25	85	0.85	2
Meas + Ind	0.25	85	0.85	2
Inferred	0.25	6	0.72	0

Trans Upper	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.25	4	0.86	0
Indicated	0.25	82	1.27	3
Meas + Ind	0.25	86	1.26	4
Inferred	0.25	4	1.36	0

Trans Lower	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.45	12	1.77	1
Indicated	0.45	91	1.43	4
Meas + Ind	0.45	103	1.47	5
Inferred	0.45	11	1.32	1

Fresh	Au Cut-off g/t	Tonnes k	Au g/t	Au koz
Measured	0.45	971	2.51	78
Indicated	0.45	4,380	1.53	215
Meas + Ind	0.45	5,351	1.71	294
Inferred	0.45	257	1.51	13

*Mineral Resources are inclusive of Mineral Reserves. Totals may differ due to rounding.

14.9.9 Validation

The block model was validated visually by the inspection of successive vertical cross-section lines and plan views in order to confirm that the block models correctly reflect the distribution of high-grade and low-grade values.

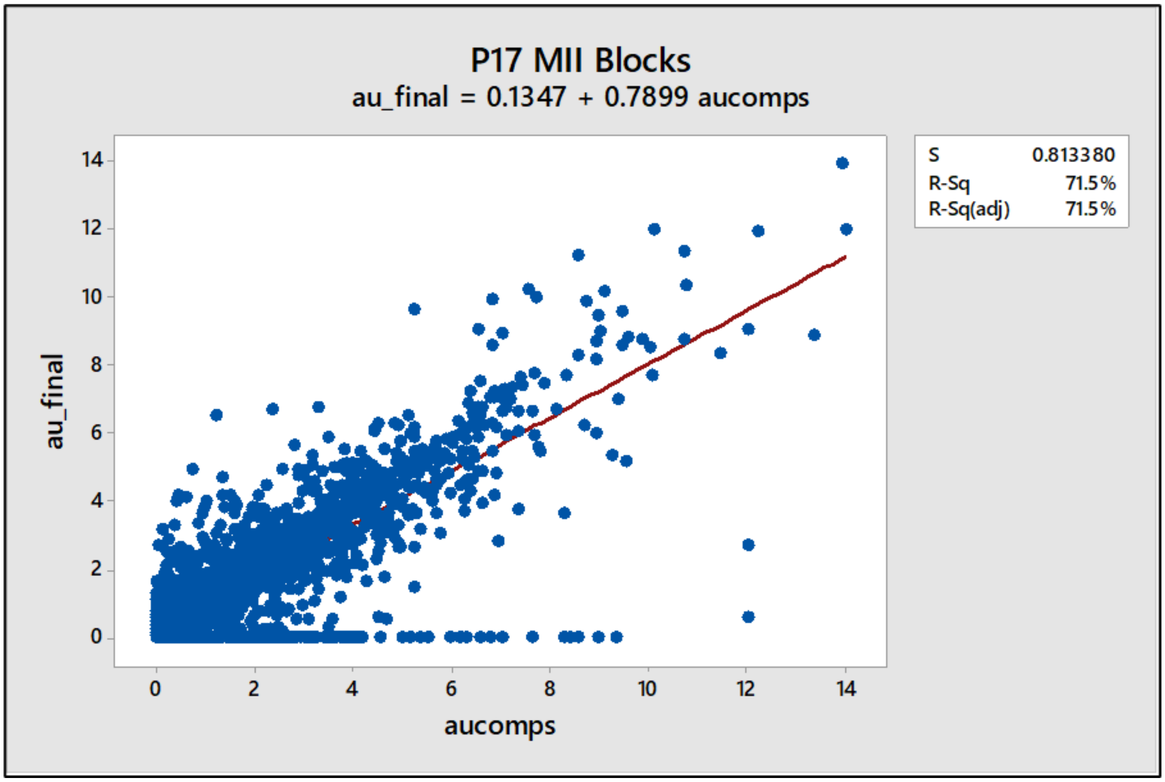
The mean block grade estimated for each sub-domain was then compared to the mean composite grade of the corresponding sub-domain (Table 14.9.9). Although these two parameters are not strictly comparable due to data clustering and volume influences, they do provide a useful validation tool in detecting any major biases and allow the comparison between input composite grade and the estimated block grade.

Table 14.9.9 P17 Zone Gold Grade Block Estimate Compared to Gold Composite Mean Grade

Domain	Tonnes	Number of Samples	Mean Composite Au Grade g/t	OK Estimated Mean Au Grade g/t	IDW Estimated Mean Au Grade g/t	NN Estimated Mean Au Grade g/t	OK Relative Difference %	IDW Relative Difference %	NN Relative Difference %
1	38,551	36	0.9	1.050	1.006	1.097	17	12	22
2	434,125	297	2.3	2.067	2.129	2.170	-10	-7	-6
3	1,481,404	519	2.32	2.033	2.150	2.350	-12	-7	1
4	156,050	178	2.87	2.682	2.741	2.479	-7	-4	-14
5	287,042	298	2.44	2.580	2.573	2.419	6	5	-1
6	50,554	79	1.55	1.358	1.377	1.312	-12	-11	-15
7	312,678	318	1.47	1.422	1.406	1.426	-3	-4	-3
8	35,926	57	0.79	0.811	0.831	0.836	3	5	6
9	908,896	196	0.87	1.062	0.968	1.181	22	11	36
10	79,707	55	0.62	0.573	0.549	0.581	-8	-11	-6
11	274,342	228	0.92	0.821	0.790	0.758	-11	-14	-18
12	524,762	310	1.47	1.581	1.562	1.482	8	6	1
13	360,753	322	1.27	1.296	1.289	1.299	2	1	2
14	420,153	343	1.04	1.038	1.039	1.048	0	0	1
15	395,845	368	1.4	1.489	1.506	1.486	6	8	6
16	1,214,752	817	1.37	1.369	1.298	1.238	0	-5	-10
17	299,960	183	0.94	0.912	0.897	0.853	-3	-5	-9
18	14,066	23	0.53	0.494	0.467	0.474	-7	-12	-11
19	14,170	24	1.04	1.139	1.207	1.452	10	16	40
20	428,178	70	0.82	0.913	0.890	1.056	11	9	29
21	554,841	100	1.1	1.067	0.979	1.300	-3	-11	18
22	1,290,433	255	1.17	0.954	0.976	1.036	-18	-17	-11
23	932,767	437	0.89	0.861	0.812	0.793	-3	-9	-11
24	626,743	180	0.74	0.732	0.726	0.776	-1	-2	5
25	762,127	216	0.59	0.484	0.500	0.546	-18	-15	-7
26	260,882	57	0.44	0.411	0.430	0.465	-7	-2	6
27	169,206	69	0.46	0.429	0.413	0.522	-7	-10	13
29	95,333	83	0.86	0.984	0.912	0.911	14	6	6
30	1,090,642	980	0.74	0.712	0.717	0.736	-4	-3	-1
31	17,501	10	0.7	0.768	0.794	0.795	10	13	14

An additional validation check was completed by comparing the average grade of the composites in a block to the associated model block grade estimate (Figure 14.9.8).

Figure 14.9.8 Validation plot between P17 Zone block grades and average composite grades



The volume estimated was also checked against the reported volume of the individual mineralization domains. Estimated volumes are based on a 0.001 g/t cut-off and partial block volumes (Table 14.9.10). The results fall within acceptable limits for estimation.

Table 14.9.10 P17 Zone Volume Reconciliation

Domain	Estimated Volume k m³	Reported Volume k m³	Ratio %
MB_01	13,233	13,291	0
MB_02	152,859	153,657	-1
MB_03	517,936	518,123	0
MB_04	54,816	54,973	0
MB_05	99,366	99,826	0
MB_06	17,438	17,536	-1
MB_07	107,128	107,272	0

Domain	Estimated Volume k m³	Reported Volume k m³	Ratio %
MB_08	12,797	13,033	-2
MB_09	319,134	318,992	0
MB_10	26,114	25,506	2
MB_11	94,809	94,708	0
MB_12	186,413	186,751	0
MB_13	130,275	131,051	-1
MB_14	151,833	151,773	0
MB_15	150,427	150,064	0
MB_16	439,088	439,091	0
MB_17	106,678	106,740	0
MB_18	4,964	4,969	0
MB_19	5,020	4,929	2
MB_20	148,416	148,815	0
MB_21	196,763	197,054	0
MB_22	458,648	458,552	0
MB_23	336,319	336,586	0
MB_24	221,344	221,141	0
MB_25	271,969	272,320	0
MB_26	89,916	90,475	-1
MB_27	60,891	60,428	1
MB_29	34,763	34,608	0
MB_30	402,455	402,853	0
MB_31	6,033	6,068	-1
Total	**4,817,845**	**4,821,185**	**0**

14.9.10 P17 Cut-Off Sensitivity

The sensitivity of the Mineral Resource block grades to changes in cut-off grade was evaluated by calculating the total Mineral Resource inventory at various cut-offs within the Mineral Resource pit (Table 14.9.11). The values presented are intended for the sole purpose of demonstrating the sensitivity of the Mineral Resource with respect to the cut-off grade.

Table 14.9.11 P17 Grade Tonnage Sensitivity Table

OXIDE MEASURED + INDICATED				OXIDE INFERRED		
Au Cut-Off	**Tonnes**	**Au**		**Au Cut-Off**	**Tonnes**	**Au**
g/t	**k**	**g/t**		**g/t**	**k**	**g/t**
1.00	68.6	1.95		1.00	5.2	1.44
0.95	72.1	1.89		0.95	5.4	1.42
0.90	76.0	1.83		0.90	5.5	1.40
0.85	80.2	1.77		0.85	5.8	1.37
0.80	84.9	1.71		0.80	6.2	1.33
0.75	89.8	1.66		0.75	6.8	1.28
0.70	94.8	1.60		0.70	7.0	1.25
0.65	101.1	1.53		0.65	7.4	1.22
0.60	107.1	1.48		0.60	7.8	1.19
0.55	115.1	1.41		0.55	8.0	1.17
0.50	124.7	1.33		0.50	8.1	1.16
0.45	133.3	1.27		0.45	8.6	1.12
0.40	141.3	1.22		0.40	9.3	1.06
0.35	150.2	1.17		0.35	9.8	1.02
0.30	159.3	1.11		0.30	10.4	0.98
0.25	170.9	1.05		0.25	10.5	0.97
0.20	181.7	1.00		0.20	10.6	0.96
SULPHIDE MEASURED + INDICATED				**SULPHIDE INFERRED**		
Au Cut-Off	**Tonnes**	**Au**		**Au Cut-Off**	**Tonnes**	**Au**
g/t	**k**	**g/t**		**g/t**	**k**	**g/t**
1.00	3,382.7	2.31		1.00	179.1	1.92
0.95	3,528.5	2.26		0.95	187.1	1.87
0.90	3,679.4	2.20		0.90	195.6	1.83
0.85	3,842.8	2.15		0.85	204.5	1.79
0.80	4,009.6	2.09		0.80	212.9	1.74
0.75	4,190.8	2.04		0.75	219.3	1.71
0.70	4,374.8	1.98		0.70	226.3	1.68
0.65	4,569.6	1.93		0.65	234.1	1.65
0.60	4,792.3	1.87		0.60	242.5	1.61
0.55	5,011.4	1.81		0.55	250.7	1.57
0.50	5,236.8	1.75		0.50	259.9	1.54
0.45	5,454.0	1.70		0.45	267.9	1.50

0.40	5,683.0	1.65		0.40	274.1	1.48
0.35	5,910.6	1.60		0.35	282.5	1.44
0.30	6,142.6	1.55		0.30	291.9	1.41
0.25	6,355.6	1.51		0.25	301.2	1.37
0.20	6,542.6	1.47		0.20	306.4	1.35

Oxide includes Regolith, Oxide and Transitional Upper units.
Sulphide includes Transitional Lower and Fresh units.
Totals may differ due to rounding.

15.0 MINERAL RESERVE ESTIMATE

Mineral Resources and Mineral Reserves are reported in accordance with National Instrument 43 101 Standards of Disclosure for Mineral Projects (NI 43-101). CIM (CIM, 2014) definitions were followed for Mineral Reserves.

Mineral Reserves of the Bomboré property were estimated by AMC using a gold price of US$1,500/oz with an effective date of 28 March 2023. Independent QP Mr D. Warren, P.Eng., is responsible for these estimates.

15.1 Mineral Reserves Summary

The Bomboré Mineral Reserves are estimated to contain 103.5 Mt at a grade of 0.72 g/t Au containing 2,403 koz Au. Mineral Reserves are composed of open pit Mineral Reserves of 95.7 Mt at an average grade of 0.75 g/t Au containing 2,301 koz Au and oxide stockpiles of 7.9 Mt at an average grade of 0.40 g/t Au containing 102 koz Au. The Mineral Reserves are summarized in Table 15.1.1.

Table 15.1.1 Bomboré Mineral Reserve Estimate as of 28 March 2023

Classification	Proven			Probable			Proven & Probable		
	Tonnes Mt	Gold Grade g/t Au	Contained Gold koz Au	Tonnes Mt	Gold Grade g/t Au	Contained Gold koz Au	Tonnes Mt	Gold Grade g/t Au	Contained Gold koz Au
Material type									
Oxide	6.2	0.62	124	50.5	0.55	897	56.7	0.56	1,020
Hard Rock	3.3	1.29	137	35.6	1.00	1,144	38.9	1.02	1,281
Total open pit	**9.5**	**0.86**	**261**	**86.2**	**0.74**	**2,041**	**95.7**	**0.75**	**2,301**
Oxide stockpiles				7.9	0.40	102	7.9	0.40	102
Total	**9.5**	**0.86**	**261**	**94.0**	**0.71**	**2,143**	**103.5**	**0.72**	**2,403**

1. CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) were used for reporting of mineral reserves.
2. Mineral Reserves are estimated using a long-term gold price of $1,500 per troy oz for all mining areas.
3. Mineral Reserves are stated in terms of delivered tonnes and grade before process recovery.
4. "Oxide" includes Regolith, Oxide, and Upper Transition material. Hard Rock includes Lower Transition and Fresh material.
5. Mineral Reserves are based on modified re-blocked mine models with variable internal dilution and mining recoveries.
6. Mineral Reserves for Block 1 (Maga), Block 2 (CFU and P8P9), Block 3 (P11), and Block 4 (Siga) are based on marginal cut-off grades that range from 0.252 to 0.270 g/t Au for Oxides, and 0.464 to 0.516 g/t Au for Hard Rock.
7. Mineral Reserves for mining blocks Block 5 (P16) and Block 6 (P17) are based on polygons developed by Orezone delimiting oxide material averaging above 0.30 g/t Au and fresh rock above 0.50 g/t Au.

8. The Mineral Reserve estimates include oxide grade reduction factors applied by Orezone based on recent mine to mill reconciliation data.
9. Tonnage and grade measurements are in metric units. Contained Au is reported as troy ounces.
10. Processing recovery varies by weathering unit and location.
11. Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability.
12. Mineral Reserves are reported effective 28 March 2023.
13. Rounding of some figures might lead to minor discrepancies in totals.

15.2 Mineral Reserves Estimation Method

The process through which the Mineral Reserves were estimated is as follows:

1. The resource block models were re-blocked by Orezone to account for dilution and losses generating diluted mine block models.

2. Geotechnical slope regions and pit optimization inputs, including mining and processing operating costs were added to the diluted block models by AMC.

3. Pit optimization was undertaken by AMC using Whittle and pit shells were selected from the results to form the basis of pit design.

4. Pit designs were created by AMC in Datamine Studio OP and Deswik based on the selected pit shells and geotechnical and operational design criteria. Ultimate pit designs were split into practical mining phases, including intermediate phase designs where appropriate.

5. Pit phase inventories were defined and imported into Minemax by AMC to generate the life of mine schedule.

6. Appropriate modifying factors for conversion of Mineral Resources to Mineral Reserves were applied.

15.3 Diluted Mining Block Models

The mining Reserve block models were derived from six geology Resource block models. The Resource block models are described in Section 14.

Prior to pit optimization, the Resource block models were re-blocked into larger block sizes to account for dilution and losses as summarized in Table 15.3.1. The estimated dilution and ore loss amounts were calculated by comparing the Reserve block model to the Resource block model.

Table 15.3.1 Dilution and Ore Loss Factors

	Block 1	Block 2	Block 3	Block 4	Block 5	Block 6
	Maga	**P8, P9, CFU**	**P11**	**Siga**	**P16**	**P17**
Resource Model File	b1_resources_model_june2023_rv1.mdl	b2_Resources_Model_juin2023.mdl	P11_20230518.csv	SIGA.BPR, .BLK	p16_rsc_2022r0.mdl	p17s_rsc_april2023_final.mdl
Resource Model Block Dimensions (Y*X*Z)	6.25*2*3	6.25*2*3	6.25*2*3	6.25*2*3	6.25*2*3	3.125*1.5*3
Reserve Model File	bombore_nord_b1_rsv_23fjun00r0.mdl	bombore_nord_b2_rsv_23gjul00r0.mdl	bombore_p11_rsv_23fjun00r0.mdl	bombore_siga_rsv_23gjul00r0.mdl	bombore_p16_rsv_23isep00r0.mdl	bombore_p17s_rsv_23dapr00r0.mdl
Created By	Orezone	Orezone	Orezone	Orezone	Orezone	Orezone
Reserve Model Block Dimensions (Y*X*Z)	12.5*4*6	12.5*4*6	12.5*4*6	12.5*4*6	6.25*2*6	3.125*3*6
Estimated Model Dilution	22%	23%	22%	16%	15%	30%
Estimated Model Ore Loss	19%	25%	19%	17%	15%	11%

Source: Orezone 2023

Siga and P16 dilution is lower than average because the mineralized zones are in general wider, while P17 dilution is higher as the mineralized zones are generally narrower than the other deposits.

Small additional ore loss factors were applied during optimization as shown in Table 15.4.1.

To define inventory after pit optimization, additional oxide reduction factors (as outlined in Section 15.6) were applied to all deposits. Additionally, polygon factors were applied to P16 and P17 only. The polygons represent mineable dig blocks for ore and are digitized in plan view for each bench. All re-blocks with a centroid inside of a polygon report as ore for processing regardless of gold grade.

15.4 Open Pit Optimization

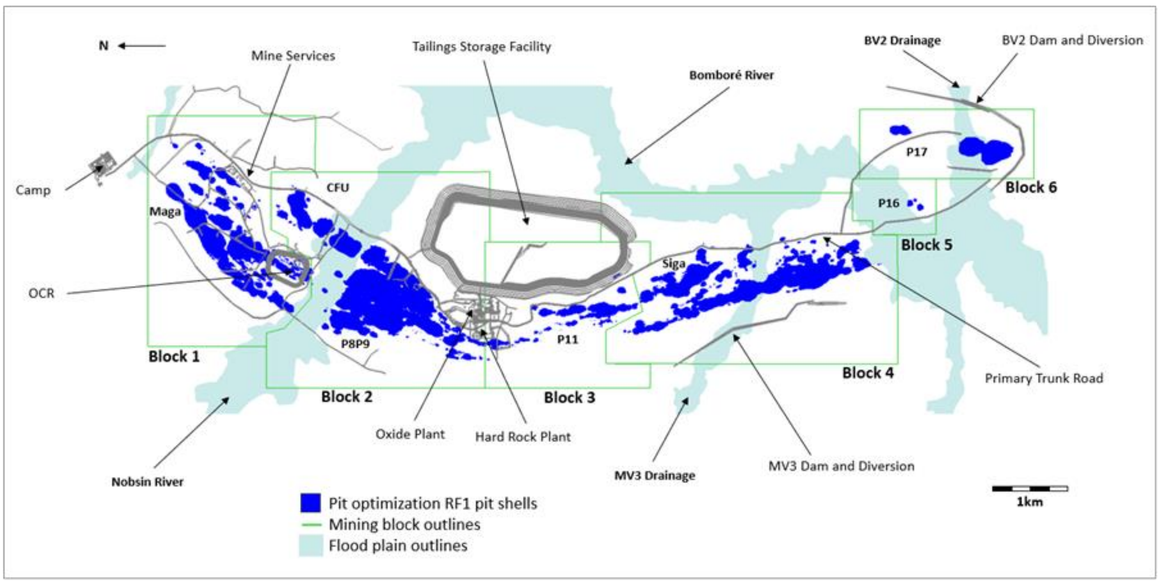
Pit optimization was undertaken by AMC in Whittle. Six pit optimizations were completed:

- Block 1 – The Maga deposit north of the Nobsin River.
- Block 2 – The CFU deposit north and P8P9 deposit south of the Nobsin River.

- Block 3 – The P11 deposit.
- Block 4 – The Siga deposit.
- Block 5 – The P16 deposit.
- Block 6 – The P17 deposit.

A layout showing the location and extent of the six optimization areas is shown in Figure 15.4.1.

Figure 15.4.1 Mining Block Model Boundaries



Pit optimization was constrained so as to not mine through the main Bomboré River flood plain which impacts the southern end of Siga and the northern end of P16. Mining through the flood plains of tributaries was allowed based on the following assumptions:

- The pits within the Nobsin River will be mined and backfilled during one dry season only. A 50 m internal buffer from the flood zone was applied to ensure that pits do not connect through to the main CFU and P8P9 pits.
- The MV3 Drainage will have an upstream dam and diversion constructed by November 2025 allowing the Siga pit to mine through the MV3 Drainage after this date. A downstream dam will also be constructed to prevent flooding from the Bomboré River.

- The BV2 Drainage will have an upstream dam and diversion constructed by November 2025 allowing P17S to extend into the flood plain after this date. A downstream dam will also be constructed to prevent flooding from the Bomboré River.

Mining under the existing Off Channel Reservoir (OCR) was allowed as this material will be available for mining when the OCR is no longer required towards the end of mine life.

The Block 1 and Block 2 block models were depleted to account for OBSA mining that had taken place up to the end of March 2023.

15.4.1 Pit Optimization Inputs

The key pit optimization inputs are summarized in Table 15.4.1.

Table 15.4.1 Pit Optimization Inputs

Parameter	Units	Block 1 Maga	Block 2 CFU and P8P9	Block 3 P11	Block 4 Siga	Block 5 P16	Block 6 P17
Global Inputs							
Gold price	US$/oz	1,500	1,500	1,500	1,500	1,500	1,500
Payable gold	%	99.95	99.95	99.95	99.95	99.95	99.95
Total Royalties	%	6.0	6.0	6.0	6.0	6.0	6.0
Off-site charges	US$/oz	3.00	3.00	3.00	3.00	3.00	3.00
Discount rate	%	5.0	5.0	5.0	5.0	5.0	5.0
Oxide plant throughput	Mtpa	5.7	5.7	5.7	5.7	5.7	5.7
Hard Rock plant throughput	Mtpa	4.4	4.4	4.4	4.4	4.4	4.4
Mining Inputs							
Mining recovery oxide and upper transition	%	98	98	98	98	100	100
Mining recovery fresh and lower transition	%	95	95	95	95	100	100
Mining base cost oxide	US$/t	2.45	2.45	2.45	2.45	2.45	2.45
Mining base cost fresh	US$/t	2.85	2.85	2.85	2.85	2.85	2.85
Ore incremental cost	US$/6 m	0.025	0.025	0.025	0.025	0.025	0.025
Reference bench elevation	m	280	280	280	268	264	264
Process Inputs Oxide and Upper Transition							
Incremental ore mining cost (oxide / upper transition)	US$/t	0.53	0.15	0.15	0.56	1.10	1.17
Process cost	US$/t	6.00	6.00	6.00	6.00	6.00	6.00
Process sustaining cost	US$/t	1.50	1.50	1.50	1.50	1.50	1.50
General & Administration cost	US$/t	2.50	2.50	2.50	2.50	2.50	2.50

Parameter	Units	Block 1 Maga	Block 2 CFU and P8P9	Block 3 P11	Block 4 Siga	Block 5 P16	Block 6 P17
Ore rehandle cost	US$/t	0.15	0.15	0.15	0.15	0.15	0.15
Closure cost	US$/t	0.15	0.15	0.15	0.15	0.15	0.15
Subtotal Process cost for Whittle	US$/t	10.83	10.45	10.45	10.86	11.40	11.47
Recovery oxide	%	91.8	91.8	91.8	91.8	91.8	95.0
Recovery upper transition	%	89.0	89.0	89.0	89.0	89.0	95.0
Process Inputs Lower Transition and Fresh							
Incremental ore mining cost (lower transition / fresh)	US$/t	0.65	0.27	0.00	0.38	0.92	0.99
Process cost	US$/t	13.75	13.75	13.75	13.75	13.75	13.75
Process sustaining cost	US$/t	1.50	1.50	1.50	1.50	1.50	1.50
General & Administration cost	US$/t	2.50	2.50	2.50	2.50	2.50	2.50
Ore rehandle cost	US$/t	0.15	0.15	0.15	0.15	0.15	0.15
Closure cost	US$/t	0.15	0.15	0.15	0.15	0.15	0.15
Subtotal Process cost for Whittle	US$/t	18.70	18.32	18.05	18.43	18.97	19.04
Recovery lower transition	%	86.0	86.0	86.0	86.0	86.0	95.0
Recovery fresh	%	81.7	84.0	81.7	81.7	81.7	95.0
Pit Rim Cut-off Grade							
Oxide	g/t Au	0.261	0.252	0.252	0.262	0.275	0.267
Upper transition	g/t Au	0.269	0.260	0.260	0.270	0.283	0.267
Lower transition	g/t Au	0.481	0.471	0.464	0.474	0.488	0.443
Fresh	g/t Au	0.506	0.482	0.489	0.499	0.513	0.443

Source: AMC and Orezone, 2023

A gold price of US$1,500/oz Au, provided by Orezone and verified by the QP as reasonable, was used as the basis for cut-off grade calculations and to determine the economic viability of the Mineral Reserves.

Gold royalties in Burkina Faso are calculated as follows:

- Spot price less than US$1,000/oz Au: 3.0% of the NSR + 1.0% Local Development Mining Fund (FMDL) tax.
- Spot price equal to or greater than US$1,000/oz Au and less than or equal to US$1,300/oz Au: 4.0% of the NSR + 1.0% FMDL tax.
- Spot price greater than US$1,300/oz Au: 5.0% of the NSR + 1.0% FMDL tax.

Therefore, based on a gold price of US$1,500/oz Au, a 6.0% total royalty was applied to the gold produced.

Mining costs are based on current site contractor rates and quotations. Process incremental ore costs include grade control and additional haulage requirements for each mining block to the process plant.

Oxide processing costs and recoveries were provided by Orezone and are based on actual on-site operating data.

Hard rock processing costs and recoveries were provided by Orezone and are based on study work conducted by Lycopodium (see Sections 13 and 21).

15.4.2 Geotechnical Inputs

The geotechnical pit slope design recommendations for Bomboré were initially developed by Golder in 2014 and updated for 6 m and 12 m benches in 2018 (Golder, 2018). In 2019, AMC reviewed the Golder reports and data and provided updated slope recommendations for P17 and the lower transition and fresh units (AMC, 2019).

The following sections summarize the slope recommendations and the overall slope angles used in the pit optimization.

Oxide and Upper Transition Geotechnical Recommendations

The Golder slope recommendations for oxide and upper transition units in Blocks 1 to 5 are summarized in Table 15.4.2.

Table 15.4.2 Golder Slope Recommendations for Oxide and Upper Transition Units (Excluding P17)

Maximum Slope Height m	Bench Height m	10% Saturation Zone			33% Saturation Zone		
		Bench Face Angle °	Catch Bench Width m	Inter-Ramp Slope Angle °	Bench Face Angle °	Catch Bench Width m	Inter-Ramp Slope Angle °
6	6	63.4	-	63.4	63.4	-	63.4
12	6	63.4	2.0	50	63.4	2.0	50
18	6	63.4	2.0	50	63.4	2.2	49
24	6	63.4	2.0	50	63.4	3.4	43
30	6	63.4	2.8	46	63.4	4.7	38
36	6	63.4	3.7	42	63.4	5.6	35
42	6	63.4	4.1	40	63.4	6.6	32

Maximum Slope Height m	Bench Height m	10% Saturation Zone			33% Saturation Zone		
		Bench Face Angle °	Catch Bench Width m	Inter-Ramp Slope Angle °	Bench Face Angle °	Catch Bench Width m	Inter-Ramp Slope Angle °
48	6	63.4	4.4	39	63.4	7.8	29
54	6	63.4	4.9	37	63.4	8.3	28
60	6	63.4	5.3	36	63.4	8.8	27
66	6	63.4	5.6	35	63.4	9.3	26
72	6	63.4	5.9	34	63.4	9.9	25
78	6	63.4	6.2	33	63.4	9.9	25
84	6	63.4	6.2	33	63.4	9.9	25
90	6	63.4	6.6	32	63.4	10.5	24
96	6	63.4	6.6	32	63.4	10.5	24

Source: Golder, 2018

Golder provided recommendations for the 10% saturation zone (normal drained conditions) and the 33% saturation zone (within and directly adjacent to flood plains with no dewatering).

The AMC slope recommendations for oxide and upper transition units in P17 are summarized in Table 15.4.3.

Table 15.4.3 AMC Slope Recommendations for Oxide and Upper Transition Units in P17

Location	Maximum Slope Height m	Operating Practices	Bench Configuration and Height m	Catch Berm Width m	Bench Face Angle °	Design Inter-Ramp Slope Angle °
All	12	No blasting within 15 m of final pit wall	Double bench 2 x 6 m; 12 m between benches	3	63.4	34

Source: AMC, 2019

Lower Transition and Fresh Geotechnical Recommendations

The Golder slope recommendations for oxide and upper transition units in Blocks 1 to 5 are summarized in Table 15.4.4.

The AMC slope recommendations for the lower transition and fresh units are summarized in Table 15.4.4.

Table 15.4.4 Lower Transition and Fresh Slope Recommendations

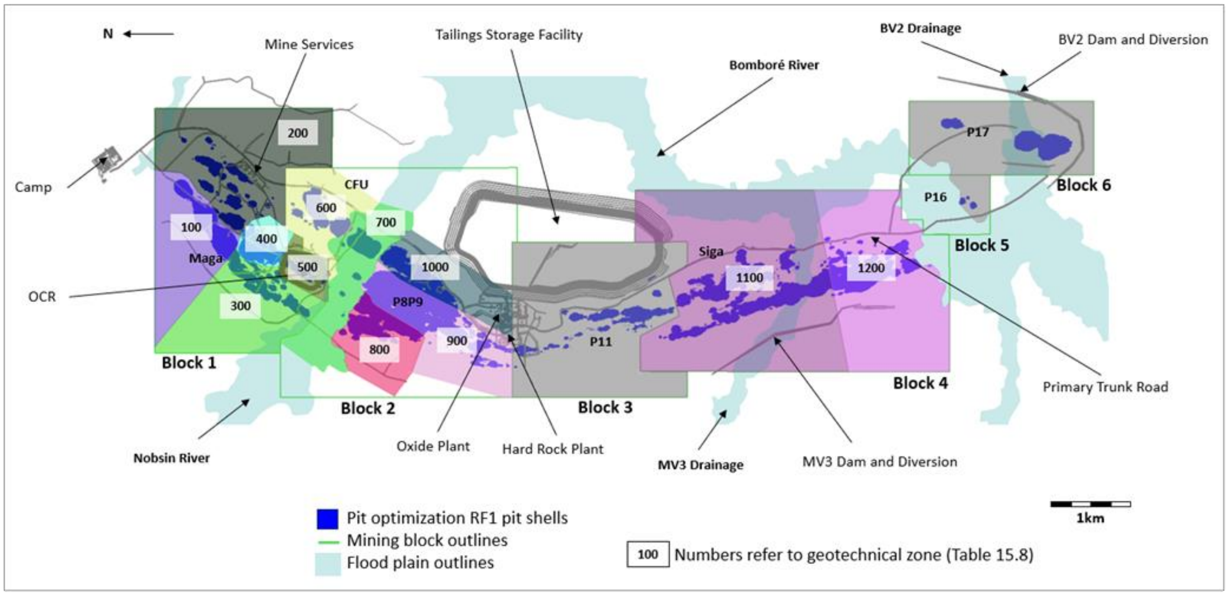
Location	Maximum Slope Height m	Operating Practices	Bench Configuration and Height m	Catch Berm Width m	Bench Face Angle °	Design Inter-Ramp Slope Angle °
Lower Transition – All Areas						
All	40	Trim Blasting	Double bench 2 x 6 m; 12 m between benches	6	63.4	45
All	50	Trim Blasting	Double bench 2 x 6 m; 12 m between benches	8	63.4	40
Fresh Rock – North Area						
West (Footwall sectors)	Any	Good Quality Trim Blasting	Double bench 2 x 6 m; 12 m between benches	5	68	50
East (Hanging wall), North and South (End Wall) Sectors	Any	Good Quality Trim Blasting	Double bench 2 x 6 m; 12 m between benches	5	68	50
East (Hanging wall), North and South (End Wall) Sectors	Any	Excellent Quality Pre-Split Blasting	Double bench 2 x 6 m; 12 m between benches	5	75	55
Fresh Rock – South Area						
West (Footwall sectors)	Any	Good Quality Trim Blasting	Double bench 2 x 6 m; 12 m between benches	5	60	45
East (Hanging wall), North and South (End Wall) Sectors	Any	Good Quality Trim Blasting	Double bench 2 x 6 m; 12 m between benches	5	68	50
East (Hanging wall), North and South (End Wall) Sectors	Any	Excellent Quality Pre-Split Blasting	Double bench 2 x 6 m; 12 m between benches	5	75	55

Source: AMC, 2019

Slope Angles Used in Pit Optimization

Prior to optimization, mining blocks were divided into zones with similar slope geometries using the 2019 feasibility study designs as a guide. A layout showing the geotechnical zones is presented in Figure 15.4.2.

Figure 15.4.2 Geotechnical Zones



Source: AMC, 2023

Blocks 1, 2, and 4 (Maga, CFU, P8P9, and Siga) were divided into groups of similar pit depth. The highest slopes for both eastern and western walls were used as the basis for the slope angle selection. Blocks 3, 5, and 6 (P11, P16, and P17) were not divided into regions due to no significant variation in depth between the separate pits.

All slope angles used in pit optimization include at least one 14 m wide ramp intersection based on the heights of each material in each geotechnical zone.

Optimization assumed the 10% saturation slope recommendations for all oxide and upper transition units. The Nobsin River pits will be mined and filled within a single dry season and the pits within the MV3 Drainage and BV2 Drainage will be mined following construction of drainage diversions leading to unsaturated conditions. The 33% saturation criteria were later applied in the design phase for the following areas:

- Areas within 50 m of the flood plain boundaries i.e. the northern end of P8P9 and southern ends of CFU and Siga.

- The new process water reservoir in the north-eastern corner of P8P9.

For fresh slopes, Orezone is planning for trim and pre-split blasting to align with the steepest slope recommendations.

The overall slope angles used in Whittle are presented in Table 15.4.5.

Table 15.4.5 Slope Angles Applied in Pit Optimization

	West			East		
Material	**Predicted slope height m**	**14 m ramp intersections**	**Whittle slope angle °**	**Predicted slope height m**	**14 m ramp intersections**	**Whittle slope angle °**
100 (Maga)						
Oxide	30	1	35	72	1	31
Upper Transition	6	1	35	6	1	31
Lower Transition	12	1	38	12	1	38
Fresh	72	1	45	36	1	43
200 (Maga)						
Oxide	42	1	33	42	1	33
Upper Transition	18	1	33	18	1	33
Lower Transition	18	1	38	18	1	38
Fresh	12	1	51	12	1	56
300 (Maga)						
Oxide	24	1	35	24	1	35
Upper Transition	6	1	35	6	1	35
Lower Transition	6	1	38	6	1	38
Fresh	12	1	51	12	1	56
400 (Maga)						
Oxide	12	1	50	42	1	33
Upper Transition	12	1	50	12	1	33
Lower Transition	18	1	38	18	1	38
Fresh	78	1	45	48	1	46
500 (Maga)						
Oxide	36	1	34	36	1	34
Upper Transition	18	1	34	18	1	34
Lower Transition	12	1	38	12	1	38
Fresh	12	1	51	12	1	56

	West			East		
Material	**Predicted slope height m**	**14 m ramp intersections**	**Whittle slope angle °**	**Predicted slope height m**	**14 m ramp intersections**	**Whittle slope angle °**
600 (CFU)						
Oxide	54	1	32	54	1	32
Upper Transition	30	1	32	30	1	32
Lower Transition	18	1	38	18	1	38
Fresh	12	1	51	12	1	56
700 (Nobsin River)						
Oxide	42	1	33	42	1	33
Upper Transition	12	1	33	12	1	33
Lower Transition	12	1	38	12	1	38
Fresh	12	1	51	12	1	56
800 (P8P9)						
Oxide	48	1	33	48	1	33
Upper Transition	12	1	33	12	1	33
Lower Transition	12	1	38	12	1	38
Fresh	72	1	45	72	1	49
900 (P8P9)						
Oxide	36	1	34	36	1	34
Upper Transition	18	1	34	18	1	34
Lower Transition	18	1	38	18	1	38
Fresh	108	1	46	108	1	51
1000 (P8P9)						
Oxide	60	1	32	10	1	32
Upper Transition	12	1	32	2	1	32
Lower Transition	12	1	38	2	1	38
Fresh	12	1	51	2	1	56
1100 (Siga)						
Oxide	30	1	35	30	1	35
Upper Transition	12	1	35	12	1	35
Lower Transition	12	1	38	12	1	38
Fresh	48	1	38	48	1	46

	West			East		
Material	**Predicted slope height m**	**14 m ramp intersections**	**Whittle slope angle °**	**Predicted slope height m**	**14 m ramp intersections**	**Whittle slope angle °**
1200 (Siga)						
Oxide	**30**	**1**	**35**	**30**	**1**	**35**
Upper Transition	12	1	35	12	1	35
Lower Transition	12	1	38	12	1	38
Fresh	114	2	39	114	2	47
P11						
Oxide	30	1	35	30	1	35
Upper Transition	18	1	38	18	1	38
Lower Transition	36	1	43	36	1	43
Fresh	48	1	38	48	1	38
P16						
Oxide	24	1	42	24	1	42
Upper Transition	6	1	42	6	1	42
Lower Transition	6	1	45	6	1	45
Fresh	6	1	55	6	1	55
P17						
Oxide	40	1	26	40	1	26
Upper Transition	40	1	26	40	1	26
Lower Transition	100	1	39	100	1	48
Fresh	100	1	39	100	1	48

Source: AMC, 2023

15.4.3 Pit Optimization Results

The results of the optimization are summarized in Table 15.4.6.

Table 15.4.6 Summary of Pit Optimization Results for Selected Pit Shells

				Total Ore	
Mining Block	**Revenue Factor**	**Waste Tonnes Mt**	**Strip Ratio W:O**	**Ore Tonnes Mt**	**Gold Grade g/t Au**
Block 1 (Maga)	1	33.2	1.8	18.8	0.69
Block 2 (CFU & P8P9)	1	67.8	1.6	43.3	0.73
Block 3 (P11)	1	5.3	2.0	2.6	0.76
Block 4 (Siga)	1	44.4	1.5	29.8	0.77

				Total Ore	
Mining Block	**Revenue Factor**	**Waste Tonnes Mt**	**Strip Ratio W:O**	**Ore Tonnes Mt**	**Gold Grade g/t Au**
Block 5 (P16)	1	0.3	2.1	0.1	1.07
Block 6 (P17)	1	27.1	6.2	4.4	1.46
Total		**178.0**	**1.8**	**99.1**	**0.77**

Source: AMC, 2023

The optimizations were carried out by varying the revenue factor (RF) which is the factor by which Whittle scales the revenue per block to generate a series of nested pit shells. The nested pit shells indicate the likely order in which mining would take place. The RF for each optimization varied up to RF 1.5 in increments of 2%.

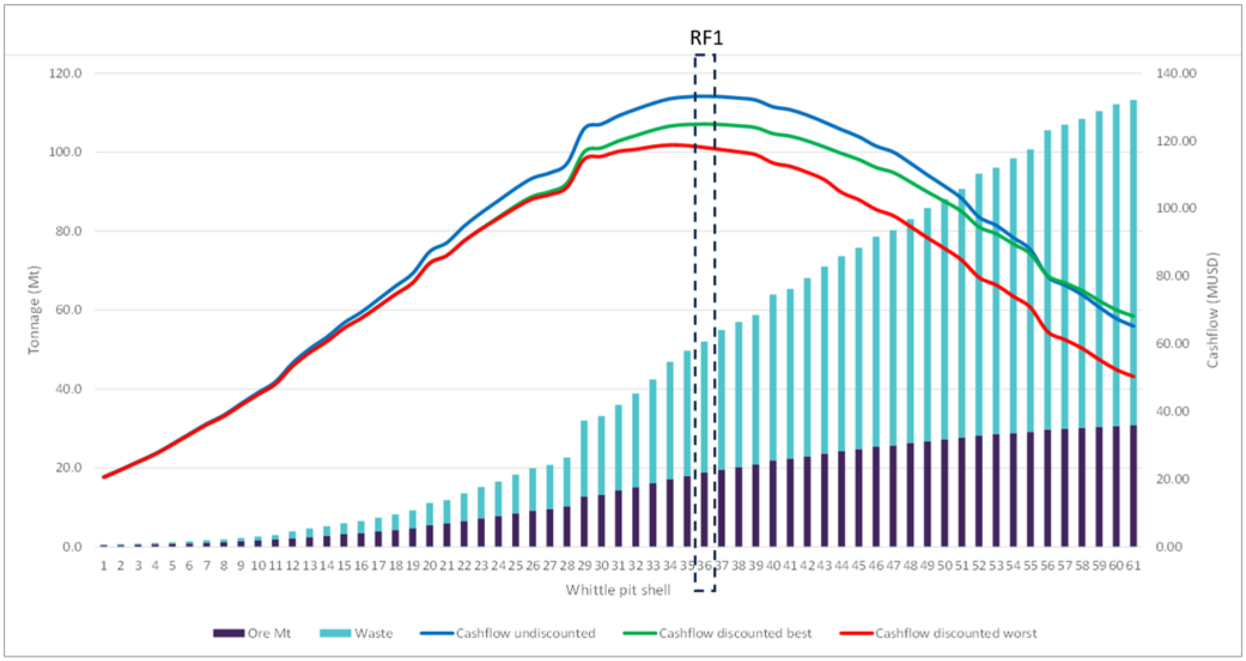
Whittle generated two discounted open pit cash flows as follows:

- Best case cash flow: The sequence that gives the maximum value by mining nested shells in the order they are generated by Whittle. This method gives the best value; however, it does not take into account practical mining sequence or the spatial relationship between pushbacks.

- Worst case cash flow: The simplest mining sequence whereby pits are mined in their entirety from top-to-bottom 'bench-by-bench'. This gives the most practical mining solution but lowest relative value.

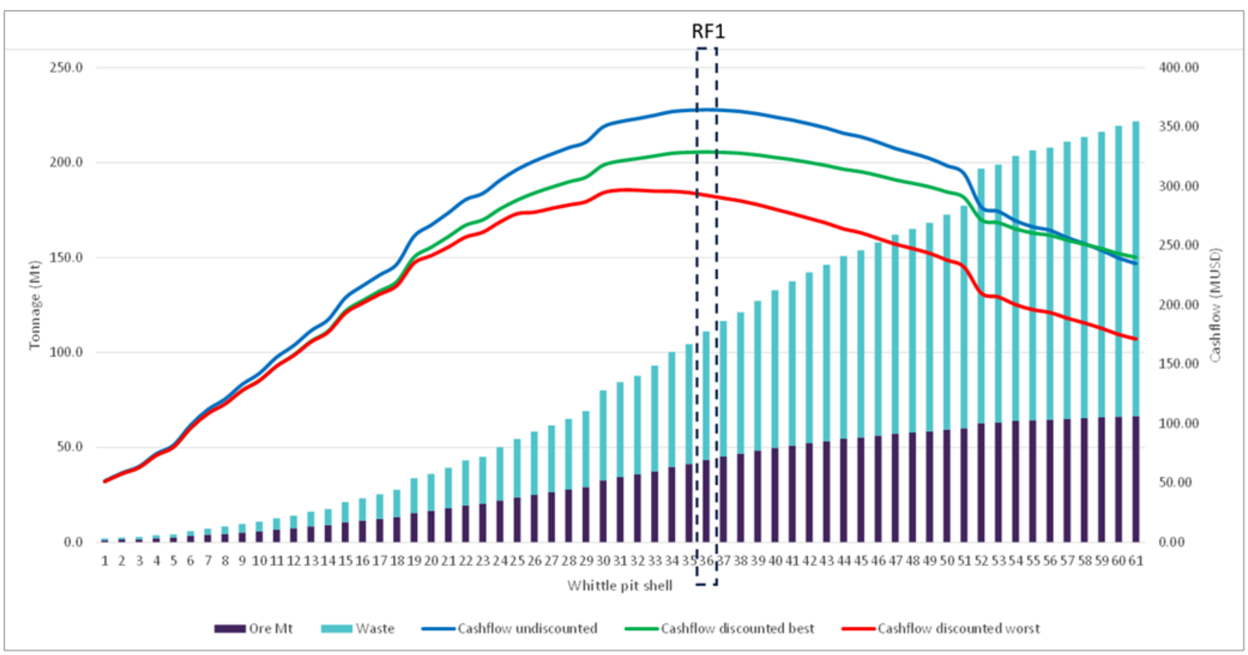
The results of the pit optimizations for each mining block are presented in Figure 15.4.3 to Figure 15.4.8.

Figure 15.4.3 Block 1 (Maga) Optimization Results



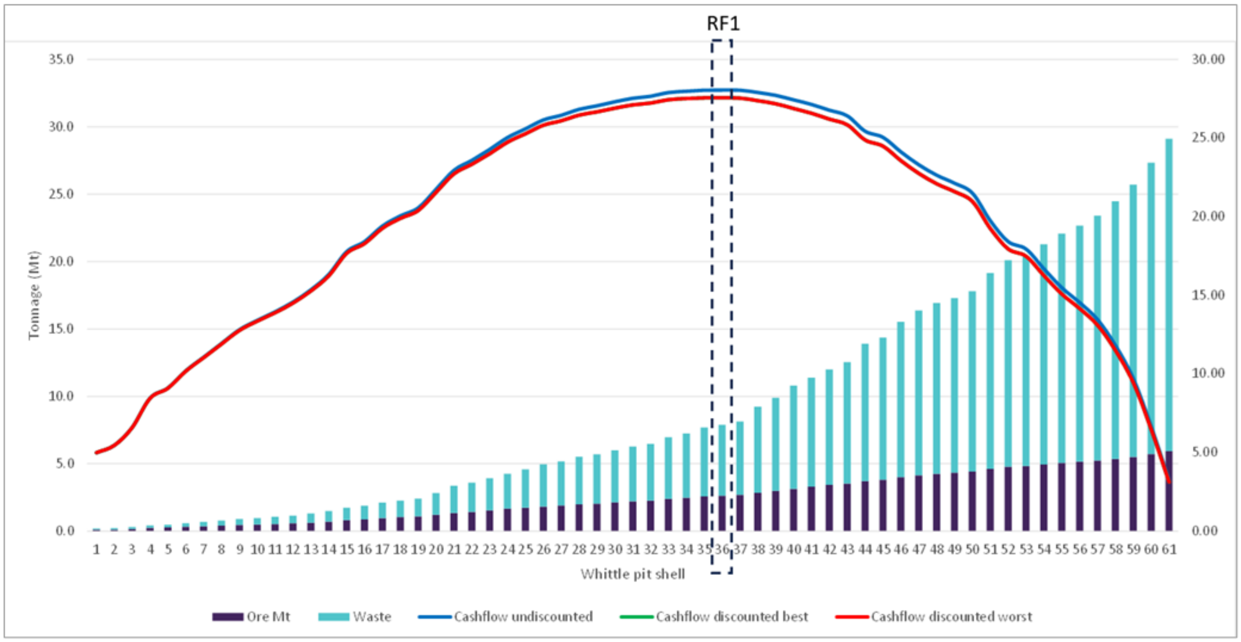
Source: AMC, 2023

Figure 15.4.4 Block 2 (CFU and P8P9) Optimization Results



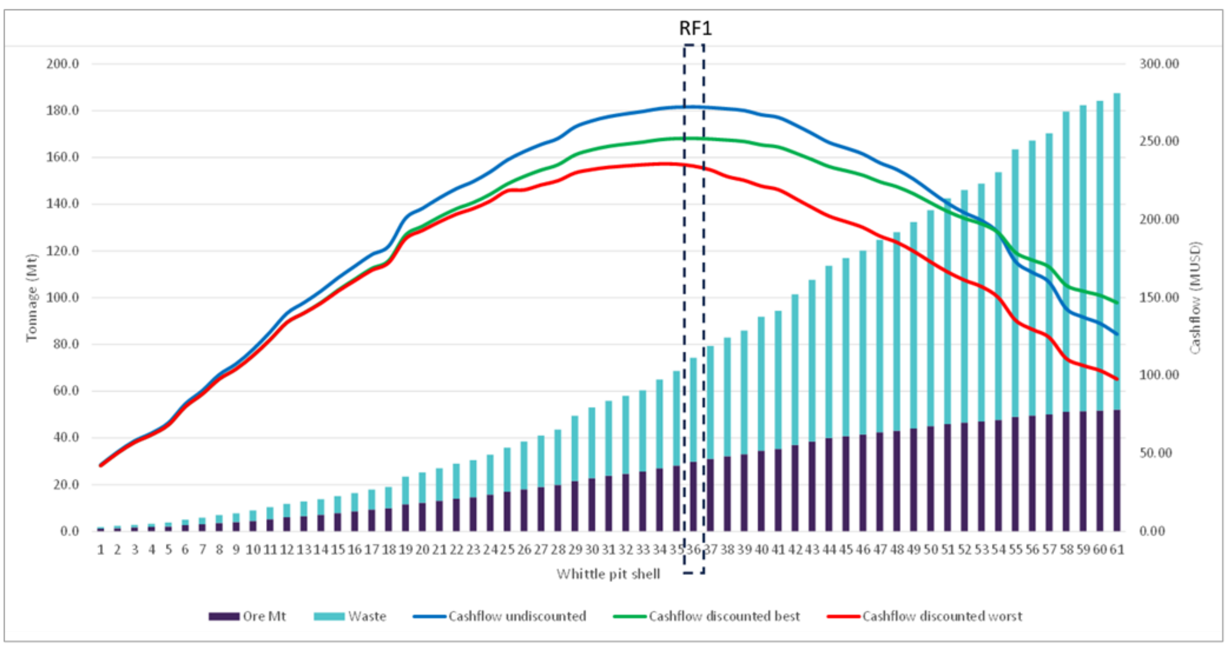
Source: AMC, 2023

Figure 15.4.5 Block 3 (P11) Optimization Results



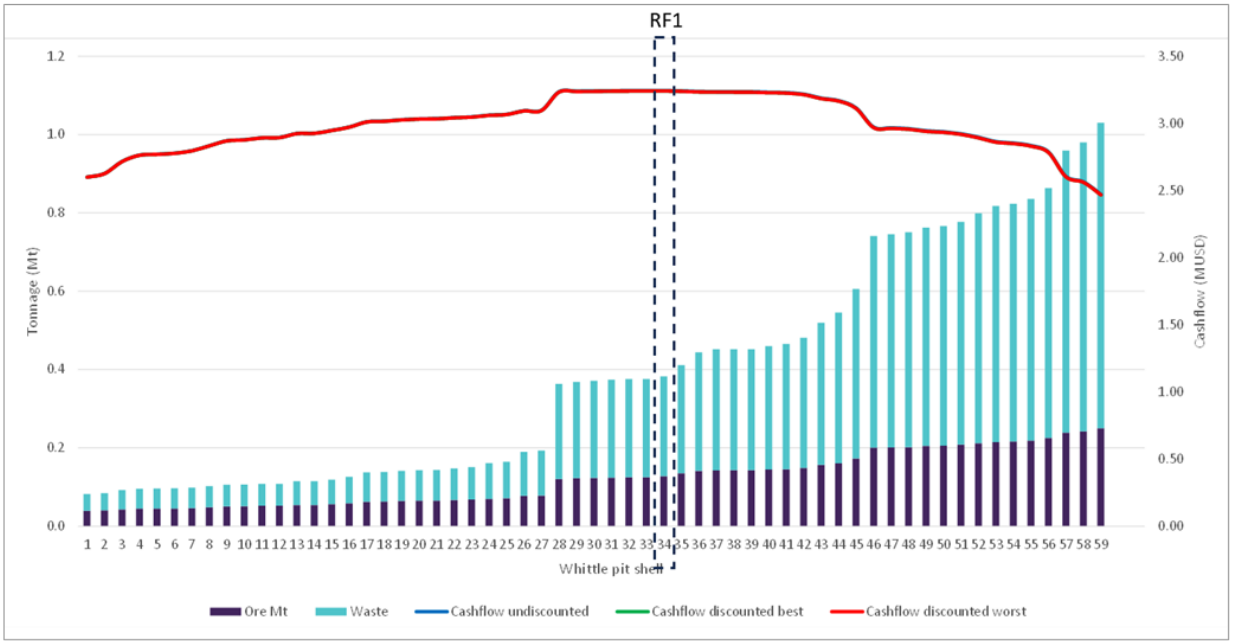
Source: AMC, 2023

Figure 15.4.6 Block 4 (Siga) Optimization Results



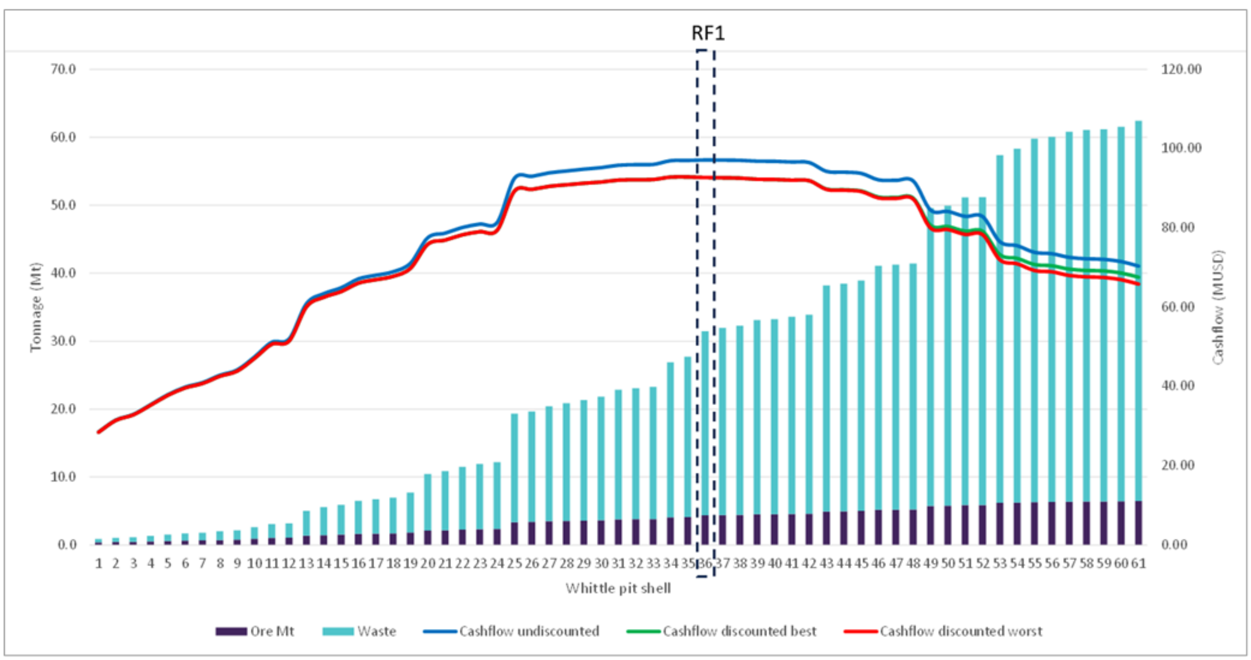
Source: AMC, 2023

Figure 15.4.7 Block 5 (P16) Optimization Results



Source: AMC, 2023

Figure 15.4.8 Block 6 (P17) Optimization Results



Source: AMC, 2023

AMC makes the following observations:

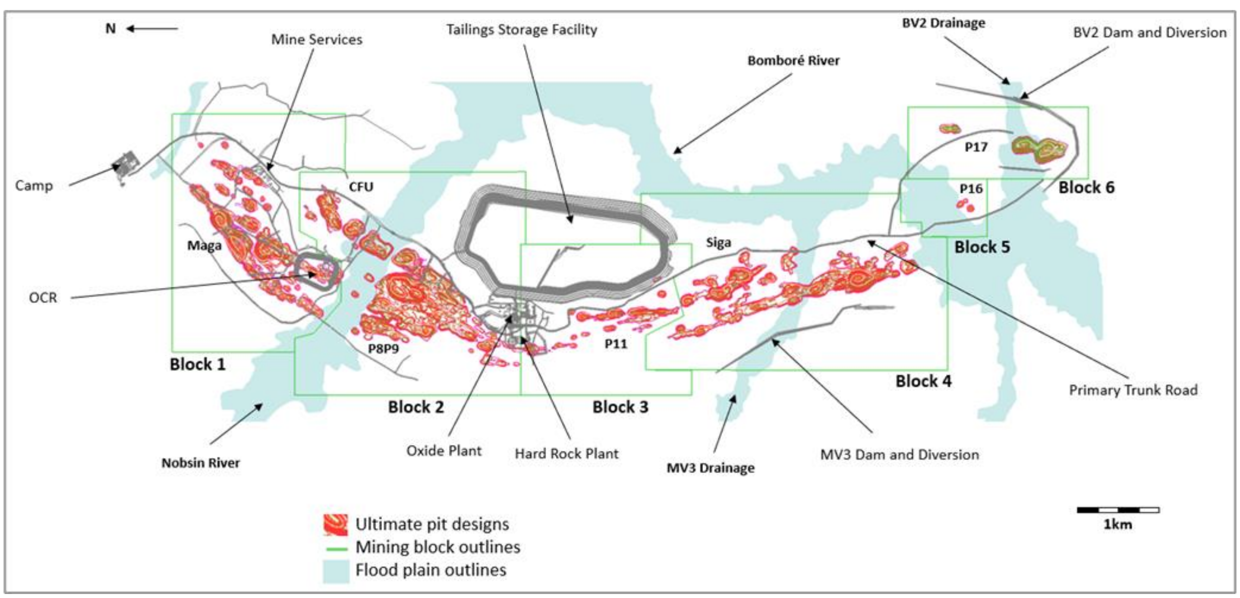
- There are no significant step changes in nested pit shells for Maga, CFU, P8P9, P11, and Siga.
- As is normal, pit size and ore quantities increase steadily with increasing revenue factor (i.e. metal price). Except for P16 and P17, the pit size and stripping ratio rise smoothly, although P11 and Siga show moderately steeper increase in stripping ratio compared to B1 and B2. P16 and P17 display some jumps and significantly higher steps in stripping ratio with bigger pits.
- Due to the relatively small size of P11, P16, and P17, best case and worst-case cash flows are effectively identical.

Orezones strategy is to maximize the gold contained in the Mineral Reserves and thus the RF1 pit shells were selected to form the basis of design. The QP is of the opinion that this pit selection strategy is appropriate and reasonable based on the comparison of undiscounted and discounted cash flows. None the less, it would be prudent to maintain a clear margin around mineralized areas so as not to sterilize potential ore in the event that metal price and economics improve in the future.

15.5 Open Pit Design

AMC generated ultimate pit designs based on the selected RF1 Whittle shells. The design incorporates 76 individual designed pits varying in depth from 18 to 180 m along a 14 km strike. 10 current starter pit phases are included in the mine plan along with a new starter phase pit in Siga and P17S. Certain larger pits were split into smaller parcels to add granularity to the mining schedule. These smaller parcels have individual pit names and are treated as pit pushback phases. A plan view of the design is shown in Figure 15.5.1.

Figure 15.5.1 Ultimate Pit Designs



Source: AMC, 2023

To summarize, the pit design is based on 6 m benches, mined in two 3 m flitches in the oxides. In the hard rock, 6 m benches are double stacked to 12 m in final pit walls, also mined in 3 m flitches to maximize mining selectivity. The design incorporates 14 m dual lane and 7 m single lane ramps at a 10% maximum gradient. The design is discussed in further detail in Section 16.

15.5.1 Comparison of Pit Design to Pit Optimization

Pit designs were evaluated against the mining block models to provide quantities of ore and waste and associated gold grades. These quantities were compared against the pit optimization results (Table 15.5.1).

Table 15.5.1 Ultimate Pit Designs Compared to Pit Optimization

Mining block	Evaluation	Ore Tonnes Mt	Gold Grade g/t Au	Waste Tonnes Mt	Strip Ratio W:O
Block 1 (Maga)	Optimization	18.8	0.69	33.2	1.8
	Pit design	17.2	0.67	33.7	2.0
	Difference (%)	-8%	-3%	2%	11%
Block 2 (CFU & P8P9)	Optimization	43.3	0.73	67.8	1.6
	Pit design	41.9	0.71	72.6	1.7
	Difference (%)	-3%	-2%	7%	11%
Block 3 (P11)	Optimization	2.6	0.76	5.3	2.0
	Pit design	2.6	0.73	6.4	2.4
	Difference (%)	0%	-4%	21%	21%
Block 4 (Siga)	Optimization	29.8	0.77	44.4	1.5
	Pit design	29.3	0.76	48.8	1.7
	Difference (%)	-2%	-1%	10%	12%
Block 5 (P16)	Optimization	0.1	1.08	0.3	2.1
	Pit design	0.2	1.03	0.4	2.3
	Difference (%)	23%	-4%	38%	12%
Block 6 (P17)	Optimization	4.4	1.46	27.1	6.2
	Pit design	3.9	1.45	26.0	6.6
	Difference (%)	-10%	-1%	-4%	6%
Total	**Optimization**	**99.1**	**0.77**	**178.0**	**1.8**
	Pit design	**95.2**	**0.75**	**187.8**	**2.0**
	Difference (%)	**-4%**	**-2%**	**6%**	**10%**

Source: AMC, 2023

Mining recoveries have been applied to the design evaluations to match those presented in Table 15.4.1 to allow like for like comparison with the optimization reporting.

As shown in Table 15.5.1, design ore tonnages are 4% less than the pit optimizations with gold grade 2% less and waste tonnages 6% more. This is within the expected tolerances for converting a pit shell into a practical mine design and is the result of including practical access and mining widths to designs which cannot be accurately accounted for in optimization.

15.6 Ore inventory

The following modifying factors post pit-optimization were applied to generate ore inventory for the mine schedule.

15.6.1 Oxide grade reduction factors

Oxide grade reduction factors were derived by Orezone based on recent mine to mill reconciliation data. These factors are 95% for Block 1, Block 2, and Block 4 (except Siga East), 85% for Block 3 and Siga East of Block 4, 50% for Block 5, and 66% for Block 6. These factors are only applied to the oxide ore. Orezone considers that a significant contribution to these factors is due to high grade ore depletion by artisanal miners.

15.6.2 Ore inventory definition

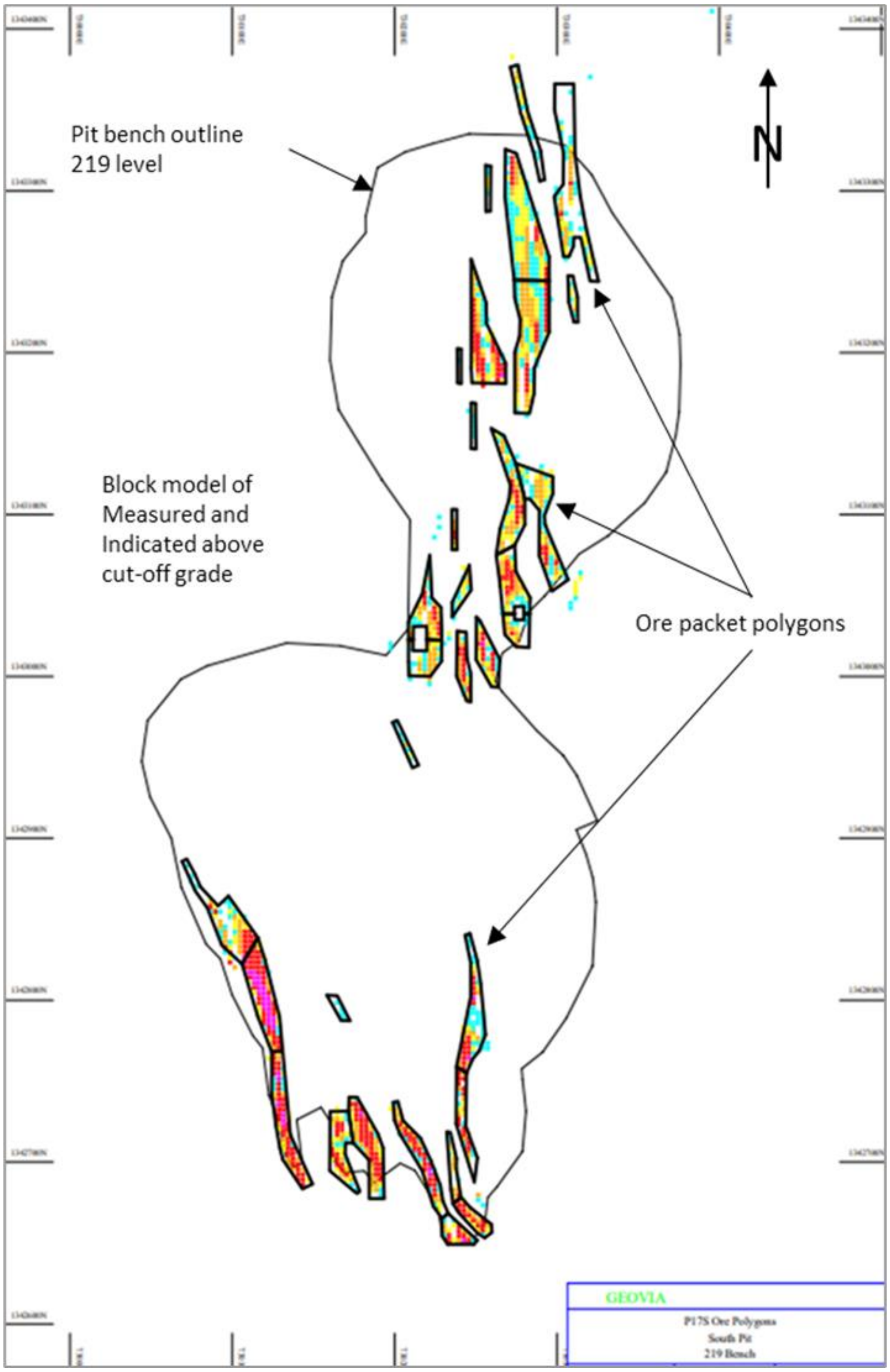
Ore inventories for Block 1 (Maga), Block 2 (CFU and P8P9), Block 3 (P11), and Block 4 (Siga) are based on the marginal cut-off grades (pit rim) summarized in Table 15.6.1. These cut-off grades were estimated using the pit optimization inputs presented in Table 15.4.1.

Table 15.6.1 Pit Rim Cut-off Grades for Ore Definition (Except P16 snd P17)

Parameter	Units	Block 1 Maga	Block 2 CFU and P8P9	Block 3 P11	Block 4 Siga
Oxide	g/t Au	0.261	0.252	0.252	0.262
Upper transition	g/t Au	0.269	0.260	0.260	0.270
Lower transition	g/t Au	0.481	0.471	0.464	0.474
Fresh	g/t Au	0.506	0.482	0.489	0.499

Ore inventories for Block 5 (P16) and Block 6 (P17) are based on polygons developed by Orezone delimiting mineable shapes of oxide material averaging above 0.30 g/t Au and fresh rock above 0.50 g/t Au. The polygons were used to code the blocks in the model directly as ore and waste. In general, the minimum width of a polygon is the re-block width (approximately 3 m) with continuity along strike of at least 12.5 m required (i.e., the grade control drill spacing). For a waste zone internal to a polygon to be separated out as waste it must be at least two re-block widths (minimum 6 m) with continuity along strike, otherwise the waste is mined as ore. Polygons are digitized in plan view for every 6 m bench inside of the ultimate pit design. Figure 15.6.1 presents an example plan view of polygons for the 219-bench elevation of the P17S pit.

Figure 15.6.1 Ore Polygon Example, P17S Bench 219



Source: Orezone, 2023

The ore inventory for mine planning and reserves estimation was based on the reserve mine model without any additional ore loss factor applied for all deposits. Although small additional ore loss factors were considered in pit optimization for Blocks 1 to 4, they were not incorporated in inventory definition. AMC performed verifications using Whittle and confirmed that the shell size was not significantly affected by small ore loss factors.

15.7 Conversion of Mineral Resources to Mineral Reserves

Inferred Resources were considered as waste. All economically mineable Indicated Resources were converted to Probable Reserves.

To define Proven Reserves, AMC drafted polygons delimiting areas of reasonable continuity and contiguous blocks of Measured Resources. Where Measured Resources maintained reasonable continuity the economically minable portion of said Measured Resources was converted to Proven Reserves. Where continuity of Measured Resources was interrupted by Indicated Resources, the economically minable portion of said Measured Resources was converted to Probable Reserves. In result, the economically minable portions of Measured Resources converted to Proven and Probable Reserves is as follows:

- Block 1 (Maga) – 100% of Measured Resources converted to Proven Reserves.
- Block 2 (CFU and P8P9) - Does not contain Measured Resources within the pit.
- Block 3 (P11) – 95% of Measured Resources converted to Proven Reserves, 5% converted to Probable.
- Block 4 (Siga) - 53% of Measured Resources converted to Proven Reserves, 47% converted to Probable.
- Block 5 (P16) - 100% of Measured Resources converted to Proven Reserves.
- Block 6 (P17) - 100% of Measured Resources converted to Proven Reserves.

15.8 Bomboré Mineral Reserve estimate by mining block

Table 15.8.1 presents the open pit Mineral Reserve estimate by mining block by weathering unit, excluding the existing oxide stockpiles. The oxide stockpiles include 7.9 Mt of Probable oxide ore at 0.40 g/t Au containing 102 koz from Block 1 and Block 2.

Table 15.8.1 Bomboré Open Pit Mineral Reserve Estimate by Mining Block excluding oxide stockpiles as of 28 March 2023

Classification	Units	Block 1 Maga	Block 2 CFU & P8P9	Block 3 P11	Block 4 Siga	Block 5 P16	Block 6 P17	Total
Summary of Mineral Reserves								
Proven								
Ore	kt	1,062	-	1,549	5,812	131	923	9,476
Gold grade	g/t Au	0.44	-	0.73	0.79	0.64	1.95	0.86
Contained gold	koz Au	15	-	37	148	3	58	261
Probable								
Ore	kt	16,124	42,378	926	23,680	12	3,061	86,180
Gold grade	g/t Au	0.69	0.71	0.65	0.75	0.23	1.22	0.74
Contained gold	koz Au	358	973	19	571	0	120	2,041
Proven and Probable								
Ore	kt	17,187	42,378	2,475	29,491	142	3,984	95,656
Gold grade	g/t Au	0.67	0.71	0.70	0.76	0.61	1.38	0.75
Contained gold	koz Au	373	973	56	719	3	177	2,301
Mineral Reserves by Material Type								
Proven								
Proven Oxide								
Ore	kt	383	-	982	2,639	109	0	4,113
Gold grade	g/t Au	0.42	-	0.60	0.63	0.58	0.29	0.60
Contained gold	koz Au	5	-	19	53	2	0	79
Proven Upper Transition								
Ore	kt	671	-	306	1,078	15	0	2,071
Gold grade	g/t Au	0.45	-	0.82	0.76	0.90	0.58	0.67
Contained gold	koz Au	10	-	8	26	0	0	44
Proven Lower Transition								
Ore	kt	9	-	178	640	7	3	837
Gold grade	g/t Au	0.60	-	1.10	1.02	1.05	0.94	1.03
Contained gold	koz Au	0	-	6	21	0	0	28

Classification	Units	Block 1 Maga	Block 2 CFU & P8P9	Block 3 P11	Block 4 Siga	Block 5 P16	Block 6 P17	Total
Proven Fresh								
Ore	kt	-	-	82	1,454	0	920	2,456
Gold grade	g/t Au	-	-	1.22	1.03	0.65	1.95	1.38
Contained gold	koz Au	-	-	3	48	0	58	109
Probable								
Probable Oxide								
Ore	kt	8,725	17,480	507	6,999	11	54	33,776
Gold grade	g/t Au	0.50	0.55	0.49	0.52	0.23	0.70	0.53
Contained gold	koz Au	139	307	8	116	0	1	572
Probable Upper Transition								
Ore	kt	2,954	8,413	131	5,202	0	69	16,768
Gold grade	g/t Au	0.62	0.62	0.54	0.56	0.40	1.24	0.60
Contained gold	koz Au	59	167	2	94	0	3	324
Probable Lower Transition								
Ore	kt	1,474	3,963	35	2,294	-	88	7,853
Gold grade	g/t Au	1.01	0.93	0.78	0.84	-	1.10	0.92
Contained gold	koz Au	48	118	1	62	-	3	232
Probable Fresh								
Ore	kt	2,971	12,522	254	9,185	-	2,850	27,782
Gold grade	g/t Au	1.17	0.95	1.01	1.01	-	1.23	1.02
Contained gold	koz Au	112	381	8	299	-	113	912
Subtotal Proven & Probable								
Proven & Probable Oxide								
Ore	kt	9,108	17,480	1,489	9,638	120	55	37,889
Gold grade	g/t Au	0.49	0.55	0.56	0.55	0.55	0.70	0.54
Contained gold	koz Au	145	307	27	170	2	1	652
Proven & Probable Upper Transition								
Ore	kt	3,625	8,413	437	6,280	15	69	18,839
Gold grade	g/t Au	0.59	0.62	0.74	0.59	0.90	1.24	0.61
Contained gold	koz Au	68	167	10	120	0	3	369
Proven & Probable Lower Transition								
Ore	kt	1,482	3,963	213	2,934	7	90	8,690
Gold grade	g/t Au	1.01	0.93	1.04	0.88	1.05	1.09	0.93
Contained gold	koz Au	48	118	7	83	0	3	260

Classification	Units	Block 1 Maga	Block 2 CFU & P8P9	Block 3 P11	Block 4 Siga	Block 5 P16	Block 6 P17	Total
Proven & Probable Fresh								
Ore	kt	2,971	12,522	336	10,639	0	3,770	30,238
Gold grade	g/t Au	1.17	0.95	1.06	1.01	0.65	1.40	1.05
Contained gold	koz Au	112	381	11	347	0	170	1,021

Notes:

1. CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) were used for reporting of Mineral Reserves.
2. Mineral Reserves are estimated using a long-term gold price of $1,500 per troy oz for all mining areas.
3. Mineral Reserves are stated in terms of delivered tonnes and grade before process recovery.
4. "Oxide" includes Regolith, Oxide, and Upper Transition material. Hard Rock includes Lower Transition and Fresh material.
5. Mineral Reserves are based on modified re-blocked mine models with variable internal dilution and mining recoveries.
6. Mineral Reserves for Block 1 (Maga), Block 2 (CFU and P8P9), Block 3 (P11), and Block 4 (Siga) are based on marginal cut-off grades that range from 0.252 to 0.270 g/t Au for Oxides, and 0.464 to 0.516 g/t Au for Hard Rock.
7. Mineral Reserves for mining blocks Block 5 (P16) and Block 6 (P17) are based on polygons developed by Orezone delimiting oxide material averaging above 0.30 g/t Au and fresh rock above 0.50 g/t Au.
8. The Mineral Reserve estimates include oxide grade reduction factors applied by Orezone based on recent mine to mill reconciliation data.
9. Tonnage and grade measurements are in metric units. Contained Au is reported as troy ounces.
10. Processing recovery varies by weathering unit and location.
11. Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability.
12. Mineral Reserves are reported effective 28 March 2023.
13. Rounding of some figures might lead to minor discrepancies in totals.

15.9 Comparison with Previous Mineral Reserve Estimates

The previous Mineral Reserve estimate for Bomboré was effective 26 June 2019. It was estimated by AMC based on resource models provided by RPA and Orezone. The Reserve was issued in a NI 43 101 Technical Report prepared for the final feasibility study for the Bomboré project.[1]

The current Mineral Reserve estimate described in this NI 43-101 Technical Report is effective 28 March 2023.

Table 15.9.1 provides a comparison of the 2019 and 2023 Mineral Reserve estimates.

[1] Ref.: NI43-101 Technical Report Feasibility Study of the Bomboré Gold Project Burkina Faso, 12 August 2019.

Table 15.9.1 Comparison with Previous Mineral Reserve Estimate

Classification	Proven			Probable			Proven & Probable		
	Tonnes kt	Gold Grade g/t Au	Contained Gold koz Au	Tonnes kt	Gold Grade g/t Au	Contained Gold koz Au	Tonnes kt	Gold Grade g/t Au	Contained Gold koz Au
Effective 26 June 2019	23,453	0.81	610	46,647	0.82	1,225	70,100	0.81	1,835
Effective 28 March 2023	9,476	0.86	261	94,042	0.71	2,143	103,518	0.72	2,403
Difference	-60%	6%	-57%	102%	-13%	75%	48%	-11%	31%

Note: Rounding of some figures might lead to minor discrepancies in totals.

The ore tonnage of current Mineral Reserves effective 28 March 2023 is 48% higher than the Mineral Reserves effective 26 June 2019; with gold grade 11% less and contained gold 31% more. Changes to the Mineral Reserve estimate are due predominantly to:

- Updated gold price assumptions. An average gold price of US$1,250 per troy oz was used to estimate the June 2019 Mineral Reserves and US$1,500 per troy oz was used for the current March 2023 Mineral Reserves.

- Lower pit rim marginal cut-off grades.

- Updated resource block models incorporating new drilling.

- Updated mine planning block models with modifying factors impacting dilution and ore loss.

16.0 MINING METHODS

The Bomboré mine has been in commercial production since December 2022 and will be further developed as an open pit operation mining oxide and hard rock material over the life of mine from over 70 separate pits of variable size and depth across a mineralized zone approximately 14 km long and 3 km wide. The 'oxide' includes the regolith, oxide, and upper transition weathering units. The regolith and oxide material is primarily free-digging material. The 'hard rock' includes lower transition and fresh rock. Upper transition and hard rock will require drill and blast prior to being loaded onto trucks.

The production schedule is based on the Mineral Reserve Estimate described in Section 15. Mining is planned to span 11.3 years from the effective date with run-of-mine (ROM) ore delivered to either the oxide plant or the hard rock plant. Some ore, particularly lower grade material, will also be stockpiled and reclaimed as needed to complement the mill feed.

The key project life-of-mine (LOM) highlights are:

- 283.2 Mt total material mined:
  - 103.5 Mt of ore:
    - 95.7 Mt of ore at 0.75 g/t Au mined and processed, including 56.7 Mt of oxide ore at 0.56 g/t Au and 38.9 Mt of hard rock ore at 1.02 g/t Au
    - 7.9 Mt of existing oxide stockpiles at 0.40 g/t Au reclaimed
  - 2.1 Moz of Au produced
  - 187.6 Mt waste
  - 2.0 strip ratio
  - 11.3-year mine life.

Mining of ore and waste is conducted by contractors with an owner's team responsible for site management, grade control, and mine planning activities. Mining of oxides is currently undertaken with 50 to 80 t diesel hydraulic excavators equipped with 3 to 5 m^3 buckets. Similar shovels are planned for mining the hard rock. The haulage requirements for oxide and hard rock material have been estimated based on rigid frame highway trucks with 26 t payload as currently deployed in the mining operations. Orezone is considering the application of trucks with higher payload of 30 to 60 t for all material types as part of the hard rock expansion.

ROM ore will be hauled to the process plants and low-grade and medium-grade material hauled to the ore stockpiles. Waste will be hauled to waste dumps with approximately 25% used for site and TSF construction.

The mine generally consists of flat terrain crossed by wide, shallow river flood plains, which flood to varying degrees during the wet season. Low hills are occasionally encountered along the eastern flank. The hills are composed of weathered oxide material with laterite caps typically 0.5 m thick.

The climate consists of a wet and dry season with rainfall generally occurring in the five months between June and October. On average, 800 mm of rainfall occurs each year in daily short bursts of heavy rainfall during the wet season. The impacts of the wet season on the mining operation have been taken into consideration in the mine schedule.

16.1 Mine Planning

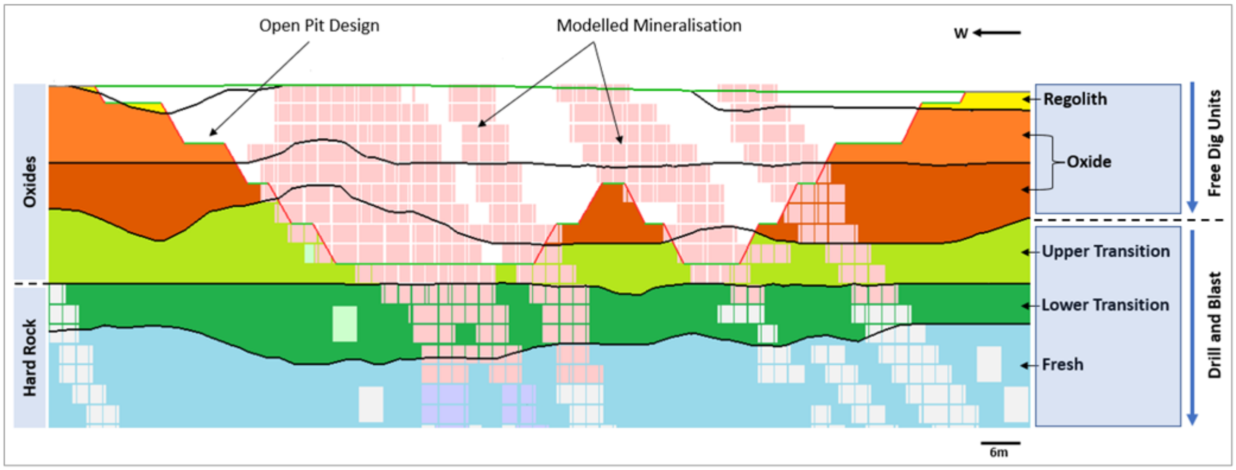
16.1.1 Material Types

The Bomboré block models are divided into weathering horizons based on geological logging and multi-element XRF analyses. The horizons take the form of smoothly undulating layers and include the following materials:

- Surface soil.
- Oxides:
  - Regolith
  - Oxide
  - Upper Transition.
- Hard Rock:
  - Lower Transition
  - Fresh Rock.

The Mineral Reserve Estimate is composed of ore from all the weathering horizons. A typical cross section through a pit showing the different weathering horizons is shown in Figure 16.1.1.

Figure 16.1.1 Typical Section of Pit Design Through Weathering Horizons



Source: AMC, 2023

Mineralization continues through all weathering horizons and each material may be classified as ore or waste as outlined in Section 15.

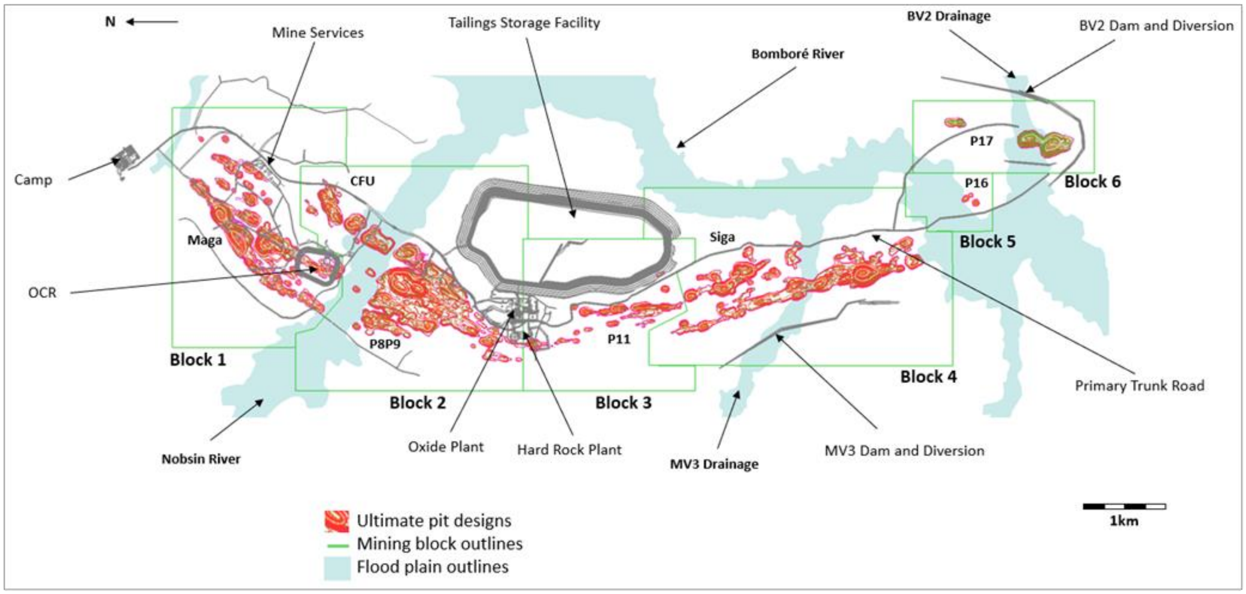
Surface soil, approximately 20 cm deep is stripped during the dry season using tracked dozers and stored around the perimeters of the waste rock dumps (WRDs) and the tailings storage facility (TSF) for use in reclamation. The remaining material is classified as follows:

- High-grade ore – Hauled to the ROM pad or the process plants.
- Low-grade and medium-grade ore – Hauled to long-term stockpiles.
- Waste – Hauled to the TSF for construction and WRDs.

16.1.2 Open pit design

AMC generated ultimate pit designs based on the selected RF1 Whittle shells. The design incorporates 76 separate pits varying in depth from 18 to 180 m along a 12 km north to south length. A plan view of the designs is shown in Figure 16.1.2.

Figure 16.1.2 Ultimate Pit Designs

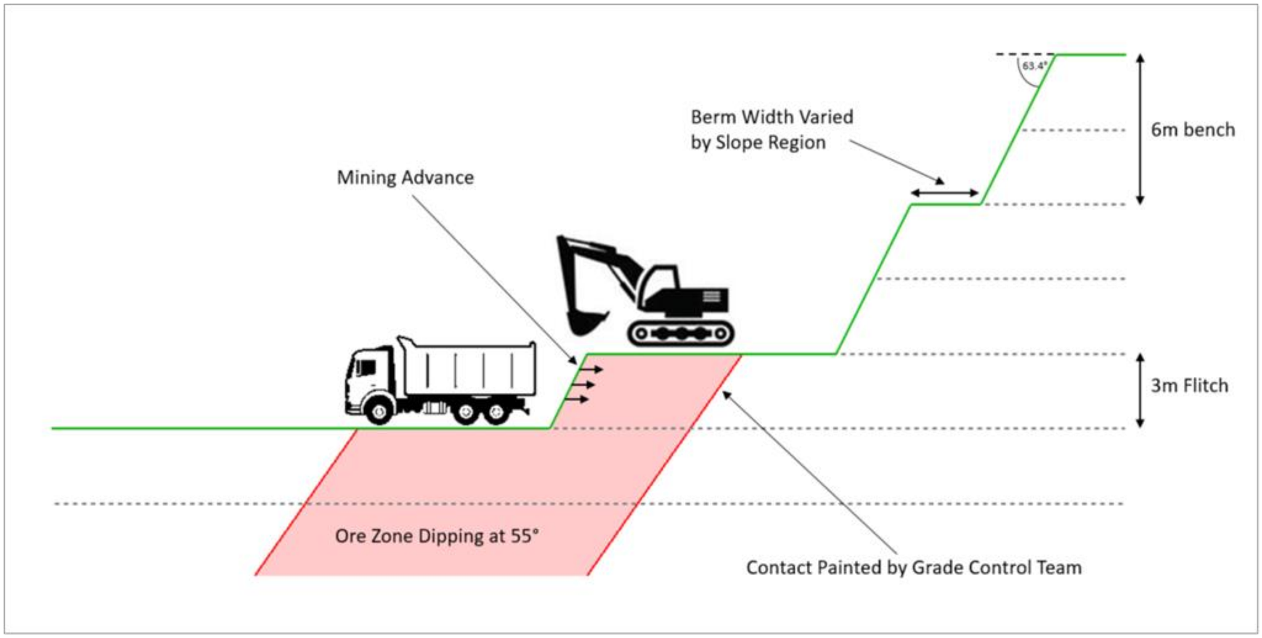


Source: AMC, 2023

The pit design is based on 6 m benches for oxides and double stack 2 x 6 m bench height for hard rock. Pit designs of ten existing active mining areas designed by Orezone were used as starter pits for Maga, CFU, and P8P9. Additional starter pits were designed by AMC for Siga and P17S.

In the oxide horizons, ore will be excavated in 3 m flitches to increase mining selectivity. A typical layout of an oxide mining area annotated with design parameters is shown in Figure 16.1.3.

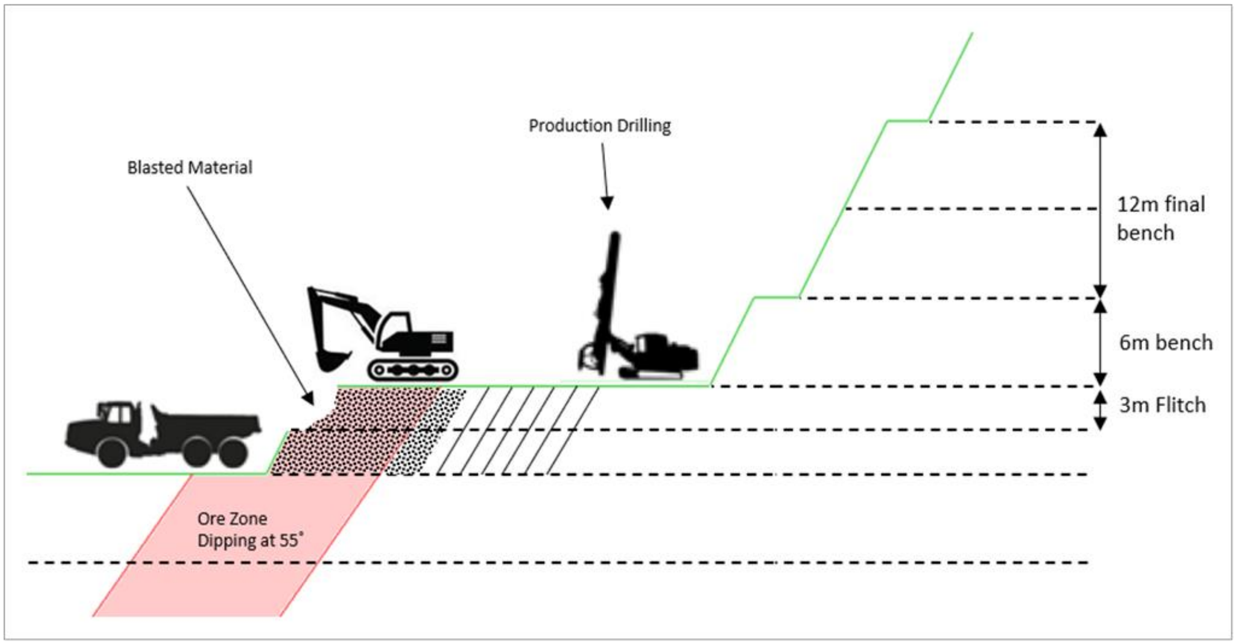
Figure 16.1.3 Cross Section of Typical Bench Layout (Oxides)



Source: AMC, 2023

Free dig operations will continue down for regolith and oxide as far as practically achievable by the mining fleet. An allowance of 10% for oxide material blasting is included, as per the experience in mining operations to date. As mining progresses into the upper transition and hard rock horizon, full drill and blast operations will be required. A typical layout of upper transition and hard rock mining is shown in Figure 16.1.4.

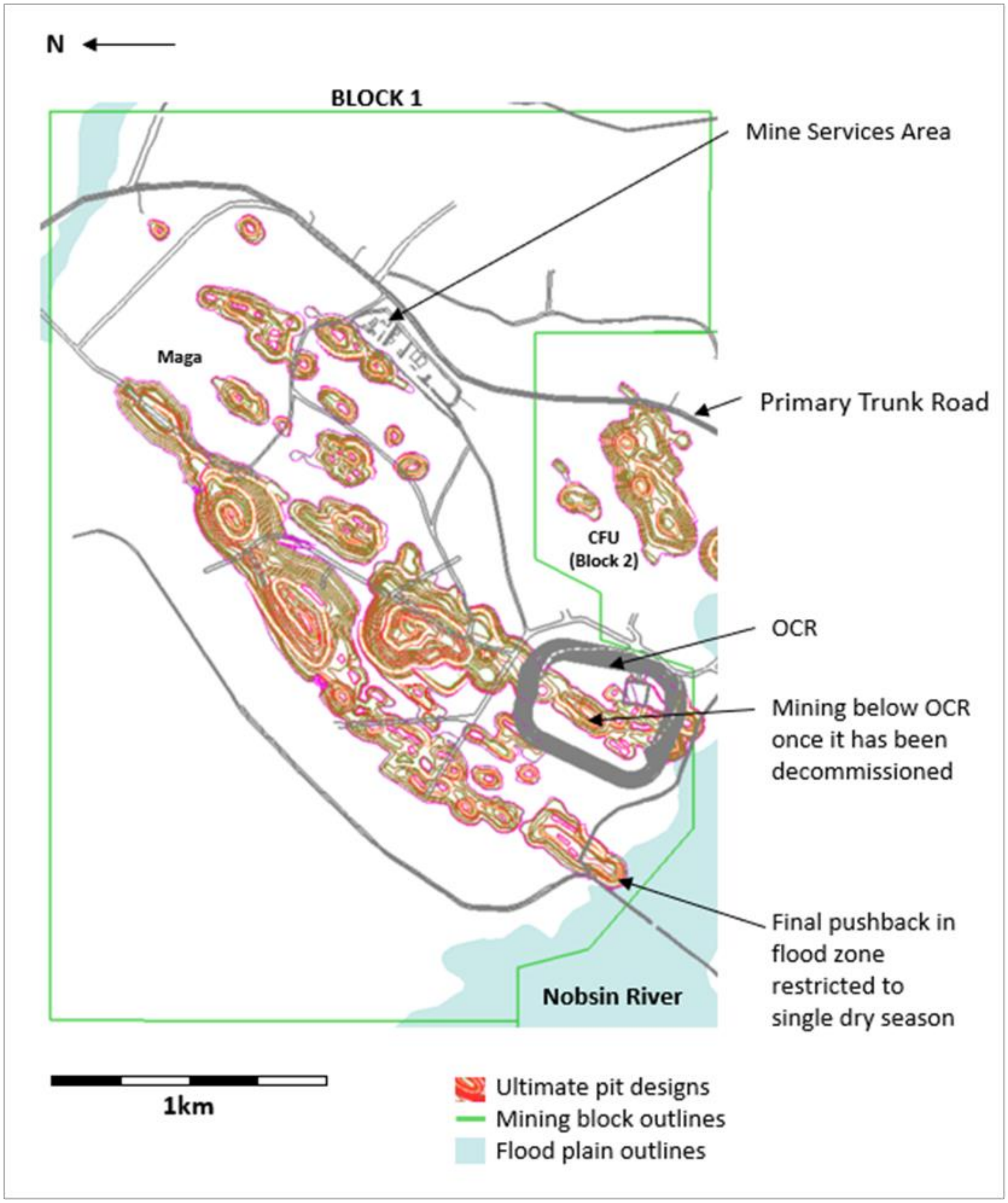
Figure 16.1.4 Cross Section of Typical Bench Layout (Upper Transition and Hard Rock)



Source: AMC, 2023

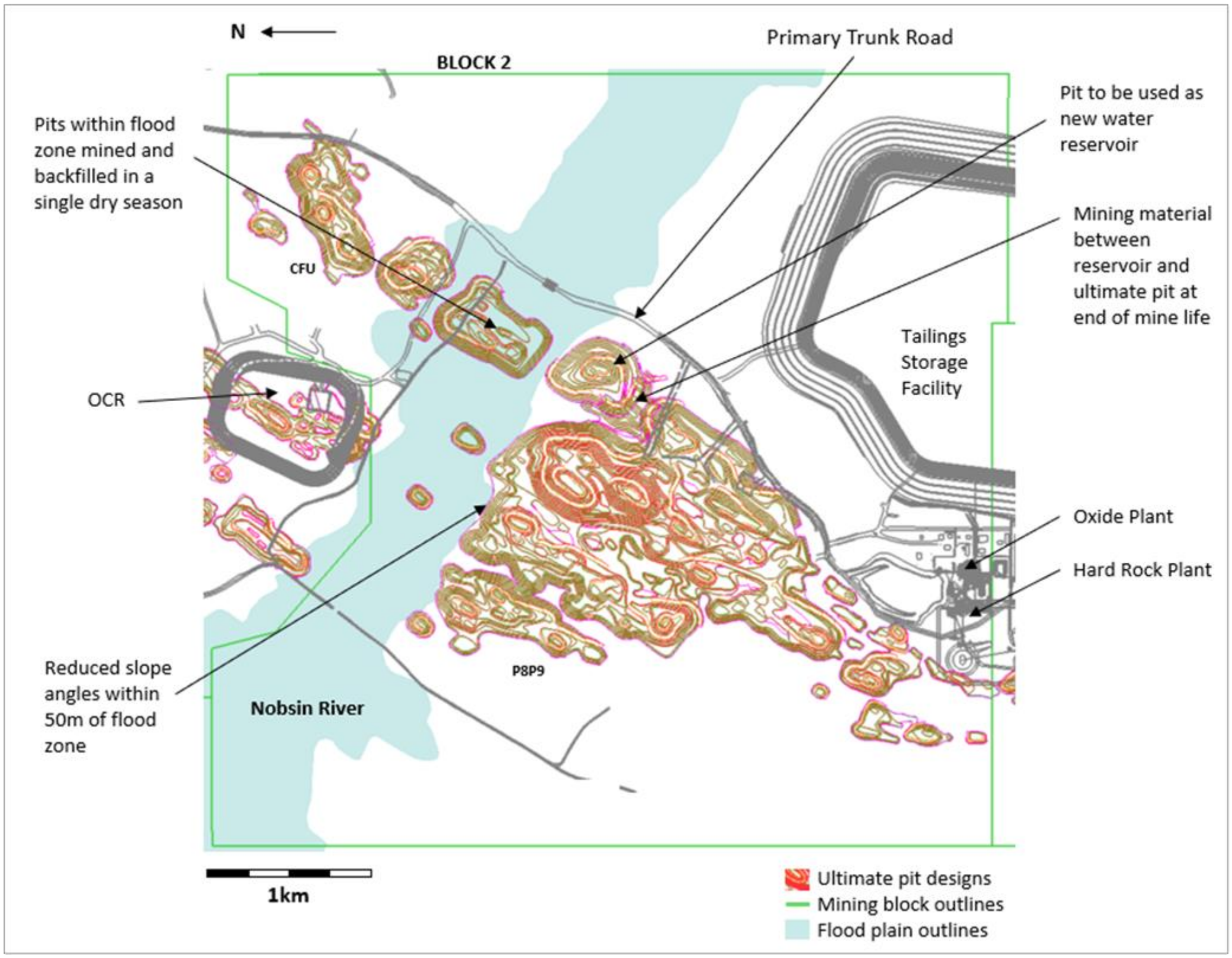
Figure 16.1.5 to Figure 16.1.10 show the pit designs by mining block.

Figure 16.1.5 Block 1 Pit Design



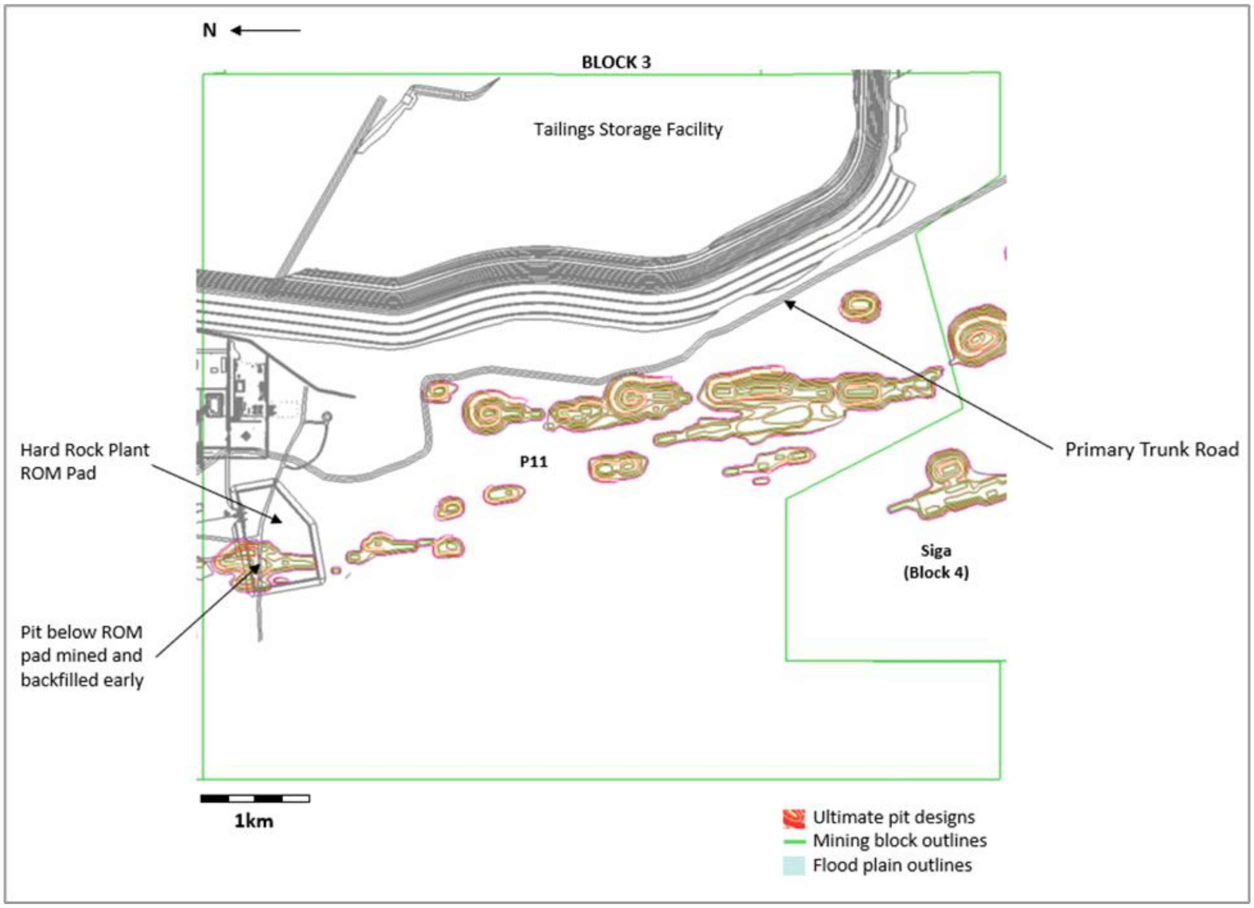
Source: AMC, 2023

Figure 16.1.6 Block 2 Pit Design



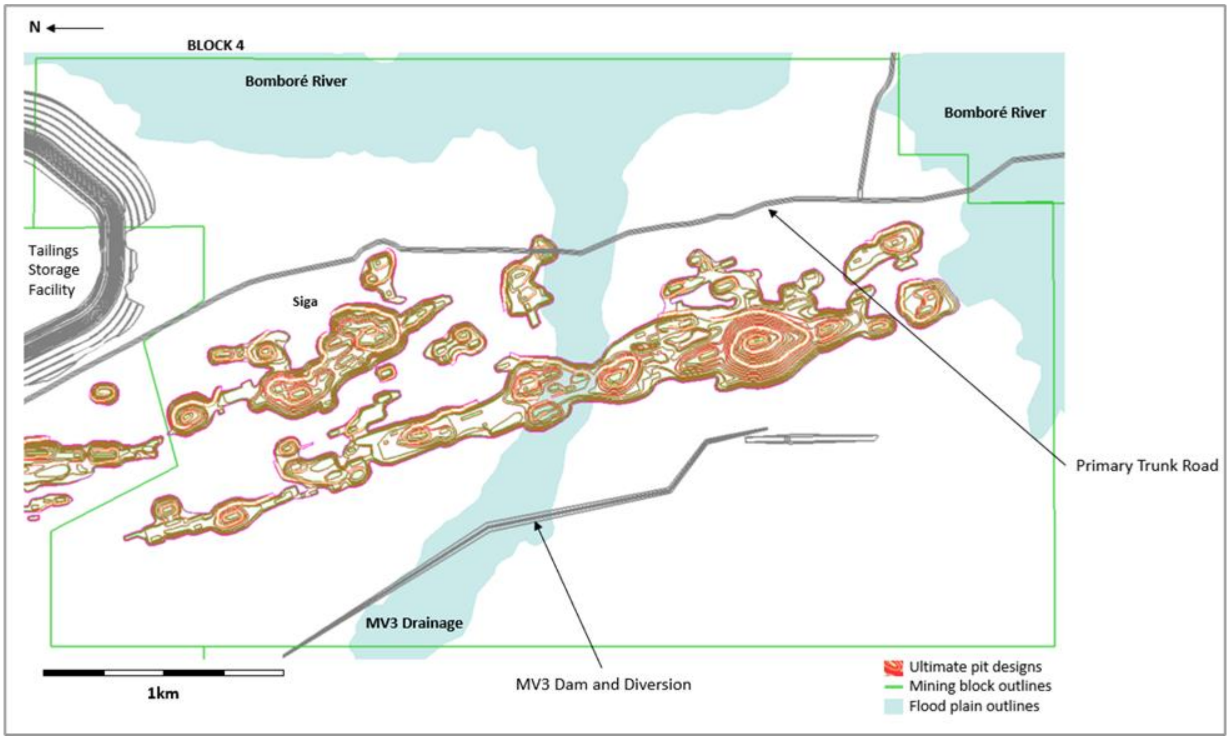
Source: AMC, 2023

Figure 16.1.7 Block 3 Pit Design



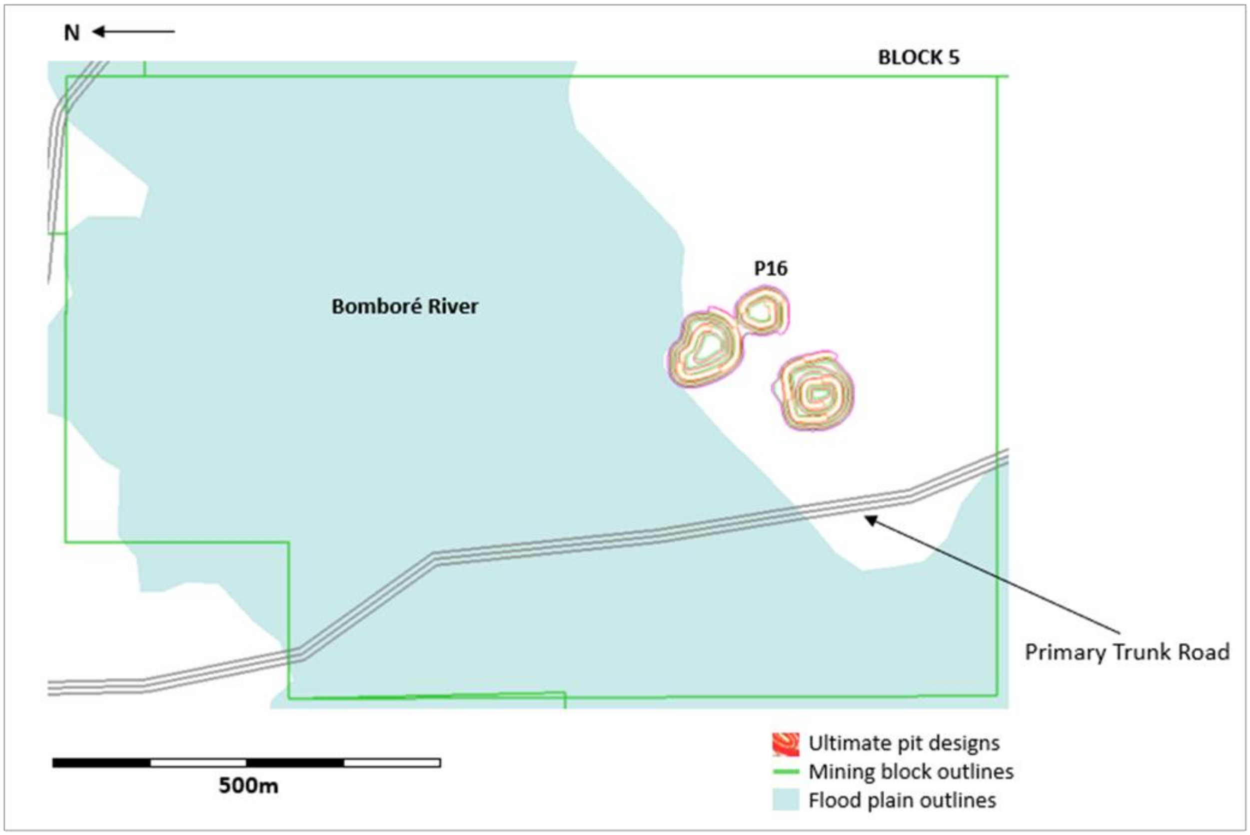
Source: AMC, 2023

Figure 16.1.8 Block 4 Pit Design



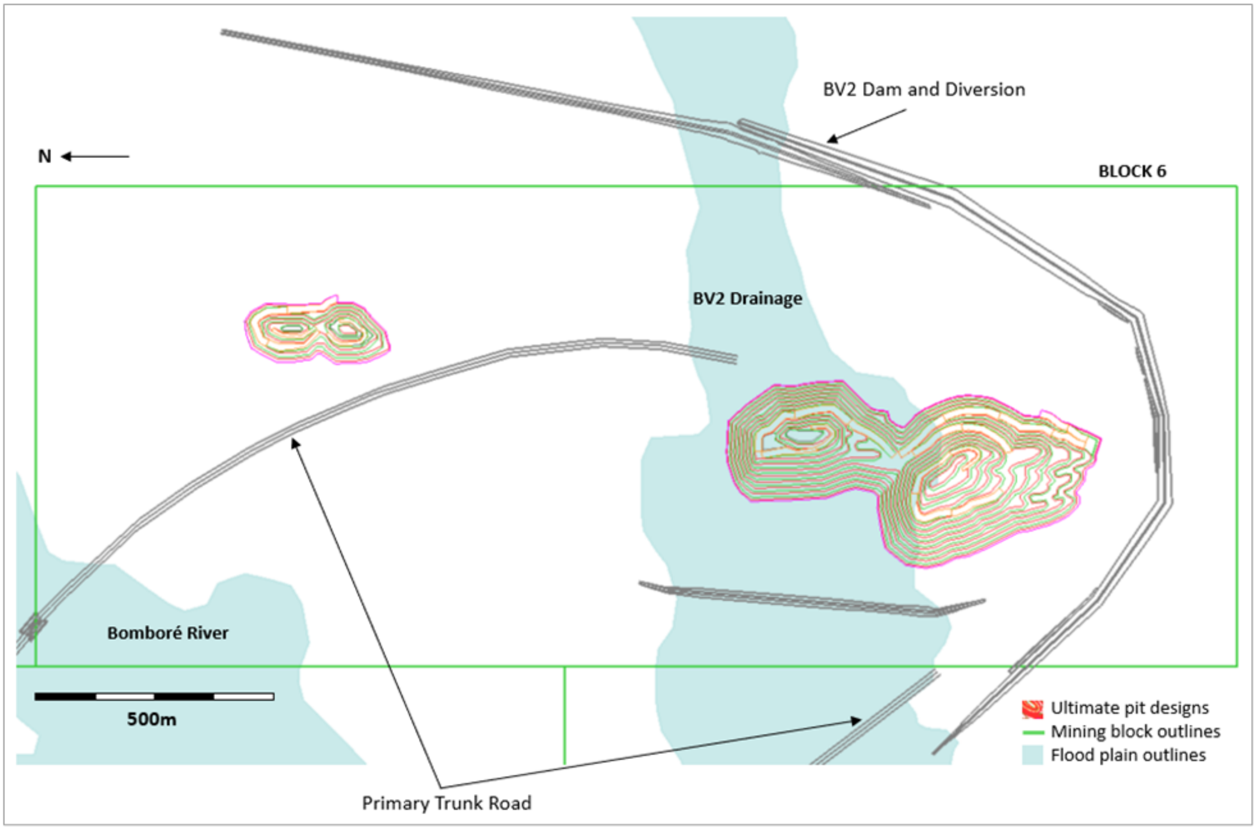
Source: AMC, 2023

Figure 16.1.9 Block 5 Pit Design



Source: AMC, 2023

Figure 16.1.10 Block 6 Pit Design



Source: AMC, 2023

16.1.3 Slope Design Parameters

The slope design criteria including face angles and berm widths are summarized in Section 15 in the following tables:

- Oxides and upper transition excluding P17: Table 15.4.2.
- P17 oxides and upper transition: Table 15.4.3.
- Lower transition and hard rock: Table 15.4.4.

16.1.4 Mining footprint

The ultimate pit designs generated by AMC have a surface footprint of approximately 457 ha; a breakdown of surface area by mining block is presented in Table 16.1.1.

Table 16.1.1 Open Pit Mining Footprint

Mining Block	Surface Footprint ha
Block 1 (Maga)	107
Block 2 (CFU & P8P9)	175
Block 3 (P11)	28
Block 4 (Siga)	123
Block 5 (P16)	2
Block 6 (P17)	22
Total	**457**

Source: AMC, 2023

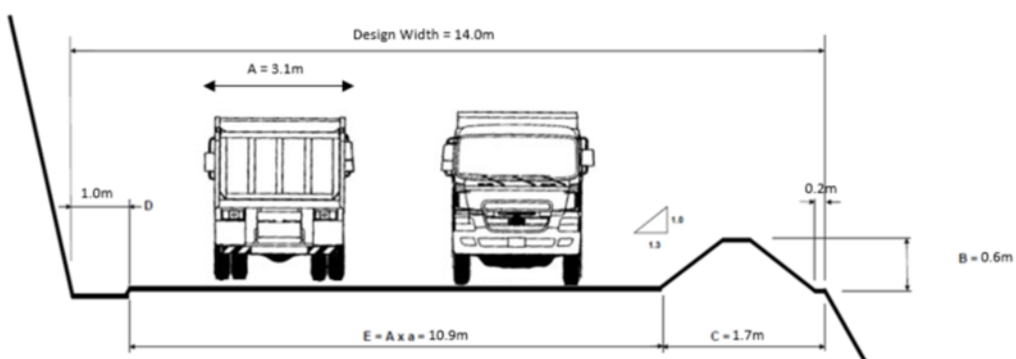
16.1.5 In Pit Ramp Design

AMC designed in pit ramps at a maximum 10% gradient. In pit ramps were designed with widths of 14 m for dual lane haulage and 7 m for single lane haulage. A cross section of a typical haul road design is shown in Figure 16.1.11.

Figure 16.1.11 Dual and Single Lane In-Pit Ramp Design

Dual Lane Number of Truck Widths "a" = 3.5

Truck	A Operating Width (m)	Tyre Diameter (m)	B Bund Height (m)	C Bund width (m)	D Drain width (m)	E Pavement Width (m)	F Total (m)
35t Dump Truck	3.1	1.1	0.6	1.7	1.0	10.9	13.6

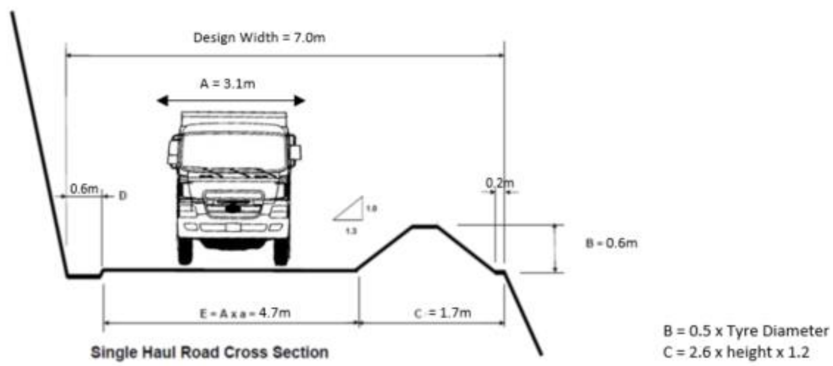


Dual Haul Road Cross Section

B = 0.5 x Tyre Diameter
C = 2.6 x height x 1.2
E = a x Operating Width

Single Lane Number of Truck Widths "a" = 1.5

Truck	A Operating Width (m)	Tyre Diameter (m)	B Bund Height (m)	C Bund width (m)	D Drain width (m)	E Pavement Width (m)	F Total (m)
35t Dump Truck	3.1	1.1	0.6	1.7	0.6	4.7	7.0



Source: AMC, 2023

The design criteria was based on 35 t payload rigid body trucks with an operating width of between 2.5 m and 3.5 m. Dual lane 14 m haul roads will allow trucks to operate safely and provide flexibility should articulated or larger trucks be considered in the future.

Due to the free dig nature of the oxide units, the exact location of ramps and pit exits of oxide-only pits may be optimized during detailed mine planning. Ramp routes may also be modified during operations when required to suit changing mining conditions. Many of the smaller satellite pits may be mined using a temporary ramp along the orebody strike. Where possible, AMC designed the haul roads along the eastern (hanging wall side) pit slopes to reduce haulage distances.

16.1.6 Surface haul roads

The mine will be served by a primary trunk road approximately 15 km in length running along the eastern side of the pits. The primary trunk road has an existing permanent bridge crossing the Nobsin River to allow for continuous haulage from Block 1 to the process plant during the wet season. A second permanent bridge crossing similar to the Nobsin bridge will be constructed in the south, crossing the Bombore River, to provide year-round access to Block 5 (P16) and Block 6 (P17).

Secondary surface roads will connect the primary trunk road to the pit ramp exits. Due to the large number of pits and dump points, there will be a large network of secondary roadways required and these will be modified regularly to suit requirements. Construction material for the surface roads will be sourced from mine waste and on-site borrow pits.

During the wet season, when heavy rainfall occurs, haulage will be stopped until the rain eases. This will prevent damage and rutting to road surfaces and ensure that road maintenance is kept to a minimum.

AMC recommends that during detailed design, surface haul roads and pit exit points be optimized to reduce construction requirements. The free dig nature of the oxide mining operations and relatively flat topography means that changes to road alignments can be easily built into future operations. Where smaller excavations are planned, narrower, and/or more temporary haulage routes may be established.

The mining contractor will maintain haul roads using their ancillary fleet to focus on active routes. The contractor ancillary fleet will consist of small excavators, motor graders, dozers, haul trucks, and compactors.

16.1.7 Drill and blast

The current experience on site is that approximately 10% of the oxide material needs blasting in order to facilitate the contractor excavators to load more efficiently. Dozer ripping has also proven effective at loosening the material for excavation but is limited to smaller areas. No fresh rock has yet been excavated to any significant degree. The current blasts are using a powder factor of approximately 0.20 kg/t in the oxides. A powder factor of 0.22 kg/t for transition and 0.25 kg/t for fresh rock has been assumed. AMC believes these are appropriate powder factors considering that minimizing blast movement and mixing are key drivers over fragmentation.

A contractor quote for drilling provided the basis for the blasthole drilling estimates. Similarly, explosive suppliers provided quotations for load and shoot services. The lowest drill and blast overall cost was chosen for the current estimate by selecting the largest hole size of 140 mm from three options provided in the drilling quotation. As the water table is expected to lie near the fresh rock contact and due to rainfall, emulsion explosives have been estimated. However, there is an opportunity for cost savings if drier conditions allow the use of less expensive ANFO products for blasting. Based on 6 m high benches, 140 mm holes, the suggested powder factors and emulsion explosives, the blast patterns shown in Table 16.1.2 are proposed.

Table 16.1.2 Blast Pattern Design

	Unit	Oxide	All Transition	Fresh
Rock density	t/m^3	1.80	2.20	2.82
Bench height	m	6.0	6.0	6.0
Powder Factor	kg/t	0.20	0.22	0.25
Explosive density	g/cc	1150	1150	1150
Hole size (diameter)	mm	140	140	140
Burden	m	5.2	4.6	4.0
Equilateral triangle spacing	m	6.0	5.3	4.6
Collar	m	3.2	2.9	2.5
Sub-drill	m	1.0	0.9	0.8

Source: AMC, 2023

Blasting along strike and monitoring of blast movement is recommended. Modelling of blast movement will allow the ore-waste contact post-blast position to be predicted.

Control wall blasting against final walls in the fresh rock is allowed in the estimate including trim blasting and pre-split of the final pit walls. Trim blasting is expected to be required to buffer final walls against production blasts. Presplitting will be 12 m high (double bench). Hole spacing will typically be between 1 to 2 m with no sub-drill. Loading will generally be with packaged explosive with no stemming. Trials will need to be undertaken to optimize the pre-split and buffer designs.

16.1.8 Grade control

The Owners Team are responsible for grade control activities, mine planning, and supervision and management of the mining, drilling, and blasting contractors' activity on site.

Prior to mining operations, grade control drilling is completed in multiple passes down to 12.5 x 12.5 m spacing to better define the mineralized zones. Grade control holes are drilled to cover 24 m of vertical with a reverse circulation drill rig. The entire length of grade control holes is sampled at 1 m intervals for assay. Samples are sent to the on-site lab for assay (see Section 11 for description of the assay lab). There may be future opportunities to reduce the amount of grade control drilling required in hard rock by sampling of production blastholes.

Based on the results of the grade control drilling, dig polygons are digitized in plan identifying ore from waste on 3 m flitches. In general, a minimum mining width of 4 m is observed during the dig packet creation. Polygons may contain occasional assays below the cut-off grade as long as the polygon average grade stays above the grade bin cut-off. Conversely, isolated assays above cut-off may not be continuous and contiguous enough to design a dig polygon. Packet grades are estimated using the OK gold grade estimation from the bench samples within the packet polygon.

The dig polygons are transferred to the surveyors who flag the top level of the bench accordingly. Maps of the packets are also prepared for the operations personnel. Ore controllers are present full time at each excavator to help guide the excavator and confirm dispatch of the loaded trucks.

As blasting becomes prevalent, AMC recommends that blast movement vectors be applied to the field positions of the muck pile flagging. If excavators are eventually employed that have on-board computer dispatch systems with precision location abilities, then the ore packets can be followed directly by the shovel operator without the need for surveyors' flagging.

The production geologists maintain regular face inspections and are in close contact with operations to ensure close collaboration and control of the mining advance.

Reconciliation between the resource block model, dig packets and plant results are performed monthly.

16.1.9 Pit dewatering

Ground water estimates were based on the Golder Hydrogeological study completed in April 2013 as part of the Feasibility Level Pit Slope Design Report. Based on the Golder study, there should be minimal inflows of ground water until the pits are excavated below the saprolite / fresh rock boundary, with the majority of inflow occurring at this boundary (i.e., the lower transition zone). As such, groundwater inflows were only considered in pits that excavate below this boundary.

Pit dewatering quantities will vary from year to year based on the number of and configuration of active pits. On average, approximately 1 Mm^3 of water will be pumped out of the pits on an annual basis. These quantities are based on historical rainfall data and estimated groundwater inflow volumes.

Mobile centrifugal diesel pumps rated at 80 L/sec of water at 135 m of head, would be suitable for the depth of pits envisioned. A variety of pumps with differing duty performance will be required though as there will be low head, high flow applications and smaller capacity pumps required. This approach has proven successful in operation over the first two rainy seasons.

Water pumped from the pits will be collected and managed in collection ponds, sediment ponds, the OCR and Reservoir 2. Reservoir 2 will be the mined-out Pit 20 in Block 2 (see Figure 16.3.1). This pit will be mined out by mid-2025 to serve as a process water reservoir supplementing the existing OCR.

Water will be pumped from the pit sumps to the nearest collection ponds on surface with pipe runs varying from 500 to 4,000 m depending on the location and depth of pit. HDPE fusion pipe 200 to 254 mm diameter pipe is proposed. Due to the robust nature of the pipe, it was assumed that a minimum 75% of pipe will be reusable and can be moved from completed pits to newly opened pits. A total of 37,500 m of new pipe is estimated for the LOM. Ongoing design and optimization of the pump and pipeline network will be required.

In pits where mining spans across the wet season, bench floors must be mined at an appropriate slope angle to ensure that water flows effectively to temporary sumps. The ultimate pit design sits predominantly within saprolite material and build-up of water on mining floors can have the potential to cause muddy conditions which can be detrimental to safety and productivity if not managed effectively. However, the operation has operated successfully over two rainy seasons to date, indicating that water management systems are proficient.

16.1.10 Dust control

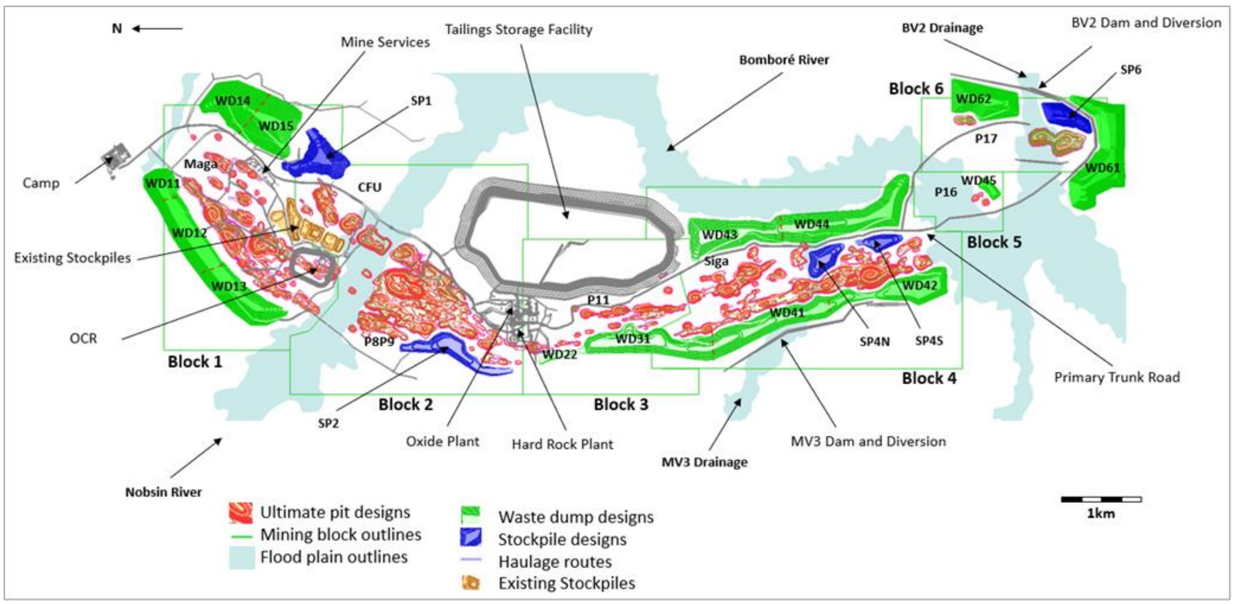
Dust will be generated primarily at digging faces, dump spots and along sections of haul roads in active use. The mining contractor will be required to maintain adequate water trucks in constant service to control dust on site. During the wet season dust suppression efforts are still required but are dramatically reduced.

Along the primary trunk road and multi-year haul roads a locally available dust suppression agent is applied and worked into the base to reduce the frequency of required watering. In addition, a sprinkler network is being established in long-term usage areas starting with the oxide plant ROM pad.

16.2 Waste dump and stockpile design

AMC generated designs for five stockpiles to accommodate lower grade ore during mining. Eight waste dumps were designed to accommodate the waste material not used in the construction of site or the TSF. A layout showing the locations of the waste dumps and stockpiles is presented in Figure 16.2.1.

Figure 16.2.1 Plan View of WRDs and Stockpiles



Source: AMC, 2023

16.2.1 Waste Dump and Stockpile Design Parameters

Table 16.2.1 summarizes waste rock dump and stockpile design criteria based on current geotechnical and closure recommendations.

Table 16.2.1 Waste Dump and Stockpile Design Criteria

Parameter	Units	Value
Lift height	m	6.0
Face angle	Ratio	3:1 (waste dumps) 1.5:1 (stockpiles)
Berm width	m	6.0

Parameter	Units	Value
Berm cross slope	%	1.5
Ramp width	m	16.0
Ramp gradient	%	5.0

Source: Orezone, 2023

All waste dumps and stockpiles have been designed with a 40 m setback from pits and infrastructure including the perimeter fence. The setback is for geotechnical stability and to allow development of access roads and water management structures as required. Long-term low-grade stockpiles followed the 3:1 dump slope criteria.

AMC has assumed a swell factor of 25% for oxides and 30% for hard rock when converting in-situ volumes to bulk and placed volumes for evaluating the capacity of stockpiles and WRDs. Due to the low density of the oxide material and potential compaction from the mining fleet, a lower swell factor for oxides may be achieved during operation and should be quantified by measurements.

Topsoil stripped within the footprint of WRDs, and stockpiles will be stored around their perimeters for rehabilitation. Rehabilitation will proceed once final design is achieved with an emphasis on completing slopes prior to moving onto new areas to allow rehabilitation to begin as soon as possible. Once the final design slope is achieved, topsoil can be placed onto the graded landforms followed by seeding.

16.2.2 Ore stockpiles

The stockpiles will be used to store low-grade and medium-grade ore. The maximum design capacity of the ore stockpiles in Million Loose Cubic Meters (MLCM) is summarized in Table 16.2.2.

Table 16.2.2 Ore Stockpile Design Capacities

Stockpile	Maximum Crest Elevation m	Overall Height m	Volume MLCM
SP1	323	50	5.1
SP2	299	27	2.7
SP4 north	287	26	1.8
SP4 south	281	18	0.8
SP6	293	31	2.0

Source: AMC, 2023

The SP1 stockpile will receive ore generated in Mining Block 1 north of the Nobsin River.

The SP2 stockpile will store ore from Mining Blocks 2 and 3.

The SP4 Stockpiles are intended for the storage of ore mined in Mining Blocks 4 and 5. The SP6 stockpile will receive ore from Mining Block 6.

The stockpiles will receive both oxide and hard rock ore which will be segregated to ensure minimal mixing. Should mixing of materials occur (i.e., resulting in a blend of oxide and hard rock ore), the mixed ore will be processed in the hard rock plant with no impact to the process other than a potentially higher throughput rate as less grinding effort is required with oxide feed.

16.2.3 Waste management

The design capacity of the WRDs is summarized in Table 16.2.3.

Table 16.2.3 WRD Capacities

WRD	Maximum Crest Elevation m	Overall Height m	Volume MLCM
WD14 & 15	353	80	24.3
WD11 & 12 & 13	317	49	25.9
WD22	293	7	0.1
WD43 & 44	305	45	22.7
WD31 & 41 & 42	305	44	21.2
WD62	305	43	8.0
WD61	305	44	14.6
WD45	275	13	0.4

Source: AMC, 2023

46.0 Mt of waste will be required for construction of the TSF embankment. A summary of the tonnages by period is presented in Table 16.2.4. Waste used for TSF construction can be from oxide or hard rock origins.

Table 16.2.4 TSF Construction Waste Requirements

Cell	Units	Total	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033
Cell 1	Mt	3.2	2.6	0.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Cell 2 - NE quadrant	Mt	12.3	0.0	0.6	1.0	1.4	1.6	1.3	1.3	1.5	1.6	1.7	0.4
Cell 2 - SE quadrant	Mt	12.3	0.0	0.6	1.0	1.4	1.6	1.3	1.3	1.5	1.6	1.7	0.4
Cell 2 - SW quadrant	Mt	10.7	0.0	0.3	0.5	0.9	1.3	1.2	1.3	1.5	1.6	1.7	0.4

Cell	Units	Total	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033
Cell 2 - NW quadrant	Mt	7.5	0.0	0.0	0.0	0.0	0.0	1.0	1.3	1.5	1.6	1.7	0.4
Total	**Mt**	**46.0**	**2.6**	**2.0**	**2.5**	**3.6**	**4.5**	**4.8**	**5.4**	**5.9**	**6.4**	**6.9**	**1.4**

Source: Orezone, 2023

1.6 Mt of waste will be required for on-site construction activities at the hard rock plant through 2024 and 2025.

16.2.4 ROM stockpile

The oxide ROM pad is located at the northern end of the processing facilities and is approximately 250 m long and 140 m wide and can hold approximately 200,000 m^3. A full ROM pad has a capacity for approximately two weeks of production to allow feed to the mill during any downtime of the mining operations caused by unfavourable weather, and conversely, dumping capacity during processing downtime. Additional surge capacity can be supplied by the nearby HG, MG, and LG stockpiles when required.

16.3 Mine plan

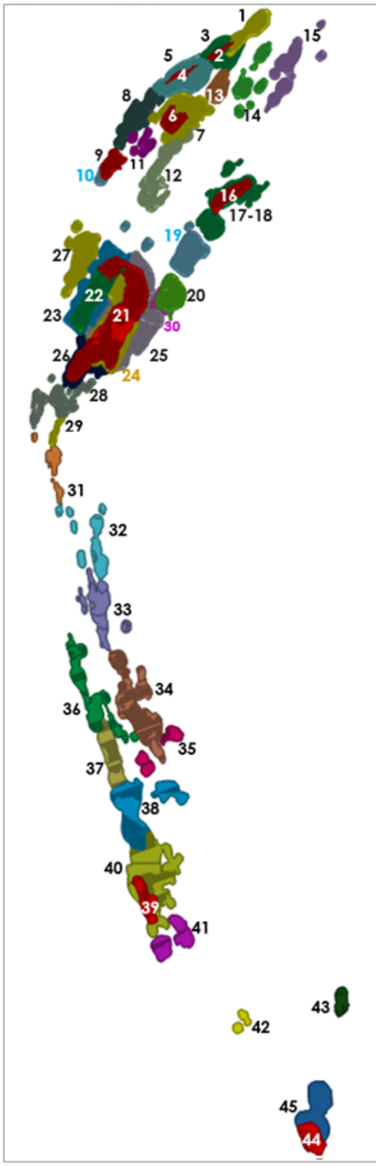
AMC used the Minemax Scheduler (Minemax) software to generate the LOM mine plan. Minemax aims to maximize the discounted operating cash flows while adhering to the constraints associated with mining and processing. The schedule was created based on an annual discount rate of 5%.

16.3.1 Schedule inputs

Mining areas

AMC grouped the pits into 45 mining areas based on geography, pit value, and tonnage. Grouping was generally done to consolidate smaller pits into a meaningful size for scheduling. Conversely, large pits were cut into smaller sectors to allow more granularity and resolution in the schedule. Larger pits were cut along logical boundaries where ridges separate deeper pits. Figure 16.3.1 shows the ore and waste quantities of each area. The existing starter pits with active mining operations have been kept as separate entities and scheduled first.

Figure 16.3.1 Grouping of Mining Areas for Production Schedule



BLOCK	PIT	Total Tonnes	Waste Tonnes	Strip ratio	Ore Tonnes	Ore Oxide+UT	Oxide+UT Au grade	Ore LT+Fresh	LT+Fresh Au grade
		Mt	Mt		Mt	Mt	g/t	Mt	g/t
Mining Block 1	1	2.6	2.1	4.7	0.4	0.4	0.88	0.0	1.63
	2	0.3	0.1	0.4	0.2	0.2	0.86	0.0	1.10
	3	5.9	4.8	4.2	1.1	0.2	0.63	0.9	1.07
	4	0.4	0.1	0.4	0.3	0.2	1.04	0.1	1.11
	5	9.7	6.7	2.2	3.1	1.8	0.55	1.3	1.04
	6	1.1	0.8	2.4	0.3	0.3	0.64	0.1	0.93
	7	13.8	10.0	2.6	3.9	2.2	0.41	1.7	1.25
	8	3.4	1.8	1.1	1.6	1.5	0.52	0.1	0.97
	9	1.5	0.6	0.7	0.9	0.9	0.46	0.0	0.89
	10	0.4	0.2	1.3	0.2	0.2	0.45	-	-
	11	0.8	0.5	1.8	0.3	0.3	0.53	0.0	0.73
	12	3.8	1.4	0.6	2.3	2.1	0.45	0.2	0.77
	13	1.9	1.1	1.2	0.9	0.9	0.46	0.0	1.00
	14	2.1	1.4	1.9	0.7	0.7	0.57	-	-
	15	3.1	2.1	2.3	0.9	0.9	0.55	0.0	1.02
Mining Block 2	16	0.9	0.6	1.8	0.3	0.3	0.56	-	-
	17	0.2	0.1	1.8	0.1	0.1	0.67	-	-
	18	7.4	5.2	2.4	2.2	2.1	0.55	0.1	1.01
	19	6.8	4.7	2.2	2.1	1.8	0.69	0.3	1.06
	20	3.6	2.4	2.0	1.2	1.2	0.48	-	-
	21	5.7	3.3	1.4	2.4	2.2	0.63	0.2	0.77
	22	4.3	2.1	1.0	2.1	2.1	0.53	0.0	0.80
	23	12.3	6.6	1.2	5.7	4.3	0.48	1.4	0.93
	24	11.6	6.6	1.3	5.0	3.8	0.66	1.2	0.90
	25	47.3	31.3	2.0	15.9	3.7	0.60	12.2	0.95
	26	4.4	2.4	1.3	1.9	1.2	0.54	0.7	0.73
	27	5.6	3.8	2.0	1.9	1.8	0.51	0.1	1.06
	28	3.2	2.2	2.2	1.0	0.8	0.55	0.2	1.24
	29	0.3	0.2	1.1	0.2	0.1	0.46	0.1	0.80
	30	1.0	0.6	1.6	0.4	0.4	0.47	-	-
Mining Block 3	31	1.3	0.9	1.9	0.4	0.2	0.51	0.3	0.98
	32	3.2	2.5	3.7	0.7	0.6	0.70	0.1	1.23
	33	4.4	3.0	2.2	1.4	1.2	0.57	0.2	1.08
Mining Block 4	34	16.1	10.5	1.9	5.6	3.2	0.66	2.4	1.13
	35	1.6	1.0	1.7	0.6	0.6	0.46	0.0	0.85
	36	5.6	3.3	1.4	2.3	1.9	0.49	0.3	1.02
	37	4.9	2.6	1.1	2.3	1.7	0.54	0.6	0.86
	38	14.4	9.0	1.7	5.4	2.5	0.54	2.9	0.84
	39	2.0	0.7	0.5	1.3	1.1	0.89	0.2	1.40
	40	29.7	19.4	1.9	10.3	3.8	0.48	6.4	1.02
	41	3.9	2.1	1.2	1.8	1.0	0.54	0.8	0.82
Mining Block 5	42	0.4	0.3	1.9	0.1	0.1	0.58	0.0	1.04
Mining Block 6	43	2.0	1.4	2.5	0.6	0.1	1.04	0.5	1.28
	44	6.1	5.1	5.5	0.9	0.0	0.63	0.9	1.71
	45	22.4	20.0	8.0	2.5	0.0	1.03	2.5	1.30
TOTAL		**283.2**	**187.6**	**2.0**	**95.7**	**56.7**	**0.56**	**38.9**	**1.02**

Source: AMC, 2023

Existing Ore Stockpiles

The inventory of existing ore stockpiles as of 28 March 2023 is presented in Table 16.3.1. The stockpiles contain oxide ore from Block 1 and Block 2. The stockpile tonnes and grade are based on the grade control modelling and grade control drilling completed on 12.5 x 12.5 m centres.

Table 16.3.1 Inventory of Existing Ore Stockpiles as of 28 March 2023

Stockpile	Tonnes Mt	Au Grade g/t
Oxide high-grade	0.5	0.87
Oxide medium-grade	6.2	0.39
Oxide low-grade	1.1	0.27
Total	**7.9**	**0.40**

Source: Orezone, 2023

Schedule Targets and Constraints

The critical production targets of the schedule are:

- Oxide plant to produce 4.3 Mt with a gold feed grade between 0.77 and 0.78 g/t in 2023 (nine months), in line with the actual and forecasted 2023 production.
- Oxide plant steady process feed of 5.9 Mtpa from 2024 onwards.
- Hard rock plant to process 1.1 Mt during its ramp-up period from September to December 2025, followed by a steady throughput of 4.4 Mtpa from 2026 onwards.
- Target total recovered gold is a minimum of 200k ounces per year from 2026 to 2030.
- Waste production profile to comply with the TSF and on-site construction requirements.

The following constraints were applied to the LOM schedule:

- The mine plan to start on 29 March 2023 accounting for actual and forecast production in Q2 and Q3 2023 from Orezone. AMC developed the mining sequence from Q4 2023.
- Total material mined (ore and waste) at 16.5 Mt in 2023 (nine months), maximum 21.0 Mt in 2024, and unconstrained from 2025.
- Maximum vertical advance rate of 20 benches per year for pits with more than 80% of oxide material, and 12 benches per year for all other pits.
- Earliest mining start date due to access constraints:

– Block 3 (P11): 1 May 2024

  - Block 4 (Siga) excluding flood zone: 1 May 2024 for Siga north and 1 October 2024 for Siga south
  - Block 4 flood zone: 1 November 2025
  - Block 5 (P16): 1 January 2025
  - Block 6 (P17) excluding flood zone: 1 October 2025
  - Block 6 flood zone: 1 November 2025.
- Pits in Block 1 and Block 2 flood zones are to be mined and backfilled within a single dry season.
- Pit 20 in Block 2 to be fully mined by 31 March 2025 (to be used as process water reservoir).
- Pits in Block 4 and Block 6 flood zones do not need to be backfilled since the MV3 Drainage and the BV2 Drainage will have an upstream dam and diversion constructed.
- Pits in mining areas 29 and 31, close to the hard rock plant stockpile and ROM pad, are to be mined and backfilled by the end of 2024.
- Material for TSF embankment construction is based on staging requirements provided by Orezone (Table 16.2.4). It was assumed that any waste type is suitable for the TSF construction.
- Construction activities at the hard rock plant require 1.6 Mt of waste in 2024 and 2025.
- Waste dumps WD12 and WD13 can be accessed from August 2025.
- Maximum global stockpile size of 20.0 Mt (excludes consideration of pit backfilling for additional stockpile capacity).
- Maximum 50% of hard rock plant feed per period from the P17 pits.
- Pit 12 and Pit 30, located under Block 1 and Block 2 reservoirs respectively, are to be mined as late as possible.

16.3.2 Mining schedule

The mining schedule for the years 2023 to 2024 is generated every quarter, followed by yearly increments. The mining operation will span over a period of 11.3 years. The total annual material mined will peak at 30 Mtpa between 2025 to 2029, and then decrease to approximately 26 Mtpa for the next three years before further reducing until the end of the mine's life in 2034. The year 2023 consists of nine months starting from 29 March.

During the life of the mine, a total of 95.7 Mt of ore will be mined, including 56.7 Mt of oxide ore and 38.9 Mt of hard rock ore. Additionally, 187.6 Mt of waste will be mined, and approximately 46.0 Mt of waste will be used in constructing the TSF embankment.

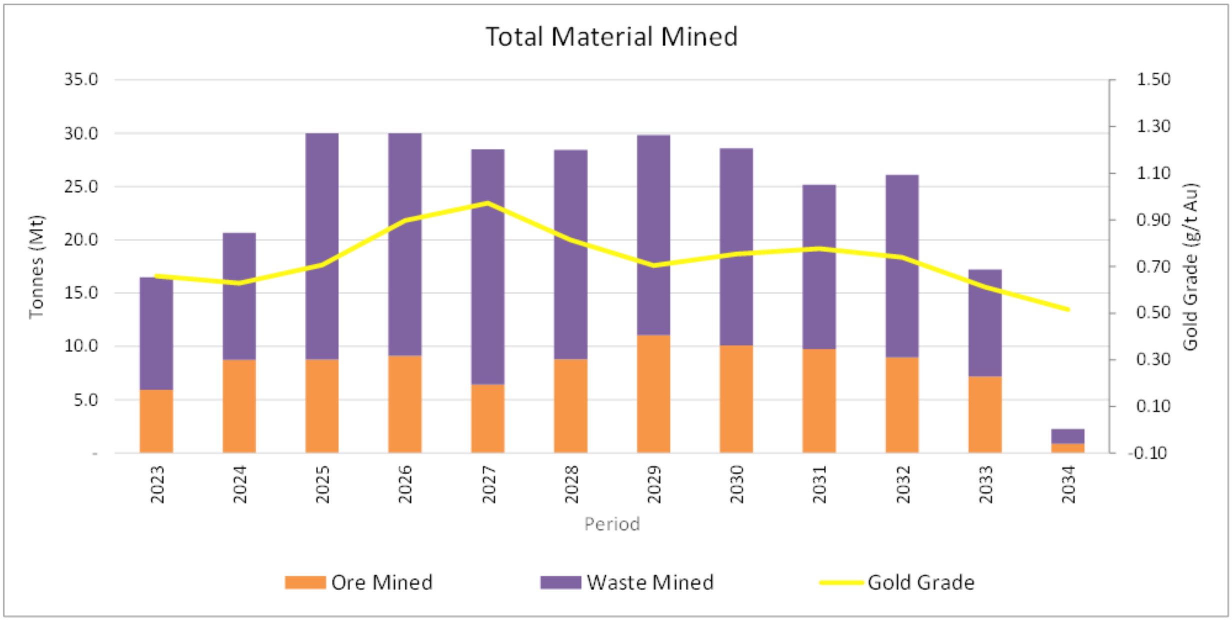
Table 16.3.2 and Figure 16.3.2 present the annual material mined.

Table 16.3.2 Annual Material Mined

	Unit	Total	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Total ore mined	Mt	95.7	5.9	8.8	8.8	9.1	6.4	8.8	11.0	10.1	9.8	9.0	7.2	0.9
Waste mined	Mt	187.6	10.6	11.9	21.2	20.9	22.1	19.6	18.8	18.5	15.4	17.1	10.0	1.4
Total mined	Mt	283.2	16.5	20.7	30.0	30.0	28.5	28.4	29.8	28.6	25.2	26.1	17.2	2.3
Strip ratio	W:O	2.0	1.8	1.4	2.4	2.3	3.4	2.2	1.7	1.8	1.6	1.9	1.4	1.6
Oxide ore mined	Mt	56.7	5.5	8.2	7.3	4.4	1.3	4.0	5.5	4.4	5.5	4.3	5.7	0.7
Oxide ore gold grade	Au g/t	0.56	0.64	0.61	0.64	0.58	0.70	0.51	0.49	0.48	0.53	0.54	0.49	0.44
Oxide waste mined	Mt	100.5	9.8	11.0	17.3	9.8	2.5	7.8	8.9	4.8	8.6	11.2	7.6	1.1
Oxide total mined	Mt	157.3	15.3	19.2	24.6	14.2	3.8	11.8	14.4	9.2	14.1	15.5	13.2	1.7
Hard rock ore mined	Mt	38.9	0.4	0.6	1.5	4.7	5.1	4.8	5.6	5.7	4.3	4.6	1.5	0.2
Hard rock ore gold grade	Au g/t	1.02	0.95	0.93	1.04	1.20	1.04	1.07	0.92	0.96	1.10	0.93	1.06	0.78
Hard rock waste	Mt	87.1	0.8	0.9	3.9	11.1	19.6	11.8	9.9	13.7	6.8	5.9	2.5	0.3
Hard rock total mined	Mt	126.0	1.2	1.5	5.4	15.8	24.7	16.6	15.4	19.4	11.0	10.6	4.0	0.5

Source: AMC, 2023

Figure 16.3.2 Annual Material Mined



Source: AMC, 2023

16.3.3 Process Feed Schedule

The oxide plant is expected to process 4.3 Mt in 2023 (nine months), followed by an annual throughput of 5.9 Mt from 2024 onwards. The hard rock plant will ramp up its production to achieve 1.1 Mt from September to December 2025, before reaching its peak capacity of 4.4 Mtpa from 2026 onwards.

As presented in Table 16.3.3, the oxide and hard rock plants both achieve targeted process feed. The oxide process feed comprises the ore that was available at the existing stockpiles as of 28 March 2023. This includes 7.9 Mt of oxide ore with an average gold grade of 0.4 g/t.

The production schedule aims to process the highest-grade ore available in order to maximize the project value. The production rate is maintained by feeding the ore directly from the pits and rehandling the ore from the stockpiles. Throughout the life of the mine, a total of 2,109 koz of gold will be produced. The consolidated feeding rate of 10.3 Mtpa will be achieved from year 2026.

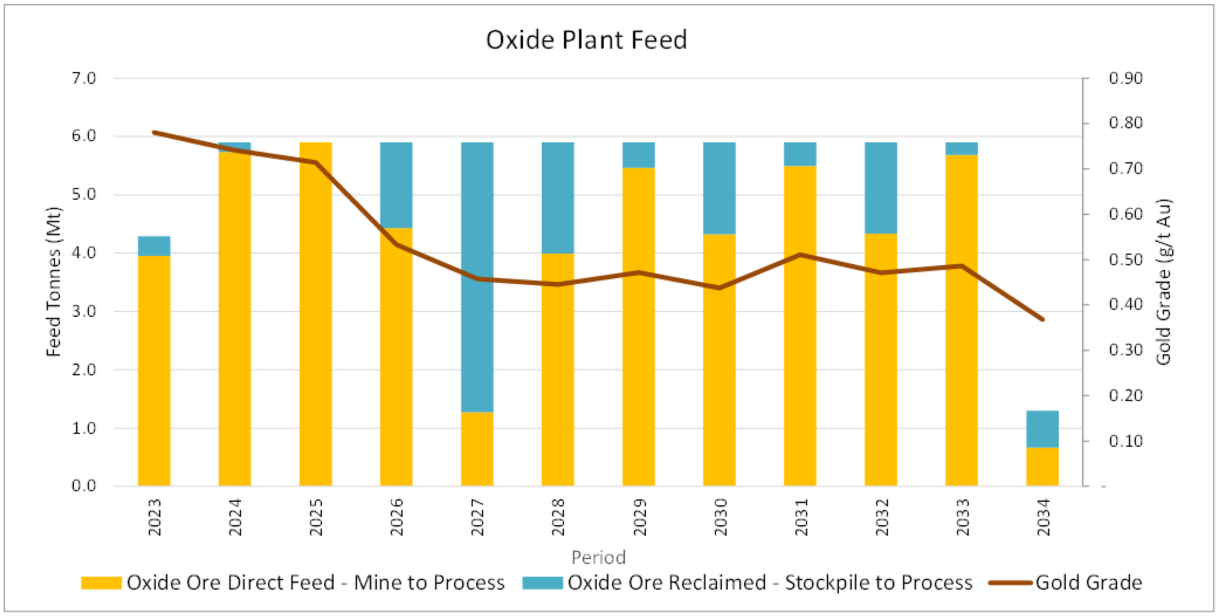
Table 16.3.3 Annual Production Schedule

	Unit	Total	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Consolidated feed														
Total ore feed	Mt	103.5	4.3	5.9	7.0	10.3	10.3	10.3	10.3	10.3	10.3	10.3	10.3	3.9
Gold grade	Au g/t	0.72	0.78	0.74	0.79	0.87	0.75	0.73	0.71	0.71	0.75	0.68	0.58	0.48
Total gold produced	koz	2,109	98	128	161	257	219	216	201	202	215	194	167	52
Oxide plant														
Oxide plant total feed	Mt	64.6	4.3	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	1.3
Gold grade	Au g/t	0.54	0.78	0.74	0.71	0.53	0.46	0.45	0.47	0.44	0.51	0.47	0.49	0.37
Oxide ore direct feed to plant	Mt	51.2	4.0	5.7	5.9	4.4	1.3	4.0	5.5	4.3	5.5	4.3	5.7	0.7
Gold grade	Au g/t	0.59	0.77	0.74	0.71	0.58	0.70	0.51	0.49	0.49	0.53	0.54	0.49	0.44
Oxide ore reclaimed	Mt	13.3	0.3	0.2	-	1.5	4.6	1.9	0.4	1.6	0.4	1.6	0.2	0.6
Gold grade	Au g/t	0.36	0.87	0.87	-	0.39	0.39	0.31	0.30	0.30	0.29	0.29	0.29	0.29
Hard rock plant														
Hard rock plant total feed	Mt	38.9	-	-	1.1	4.4	4.4	4.4	4.4	4.4	4.4	4.4	4.4	2.6
Gold grade	Au g/t	1.02	-	-	1.21	1.33	1.14	1.12	1.02	1.08	1.08	0.95	0.70	0.54
Hard rock ore direct feed to plant	Mt	32.8	-	-	1.1	3.7	4.4	4.4	4.4	4.4	4.3	4.4	1.5	0.2
Gold grade	Au g/t	1.11	-	-	1.21	1.38	1.14	1.12	1.02	1.08	1.10	0.95	1.06	0.78
Hard rock ore reclaimed	Mt	6.1	-	-	-	0.7	-	0.0	-	-	0.1	-	2.9	2.4
Gold grade	Au g/t	0.58	-	-	-	1.07	-	1.27	-	-	0.51	-	0.52	0.52

Source: AMC, 2023

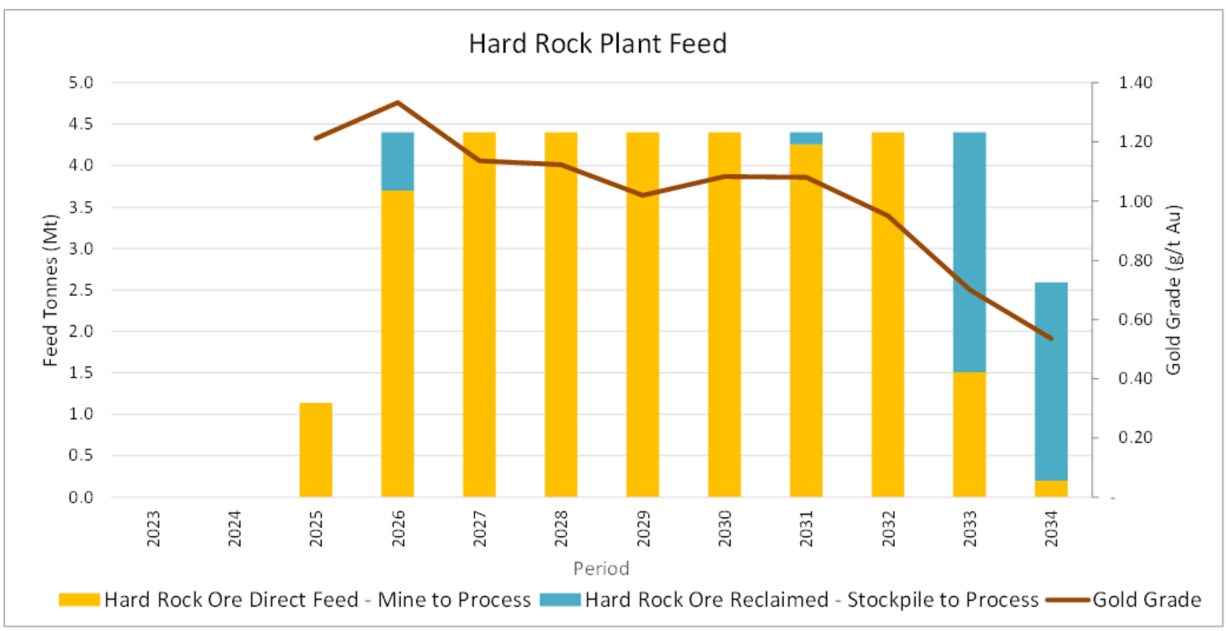
Figure 16.3.3 and Figure 16.3.4 show the planned feed of the oxide and hard rock plants respectively.

Figure 16.3.3 Oxide Plant Feed Schedule



Source: AMC, 2023

Figure 16.3.4 Hard Rock Plant Feed Schedule



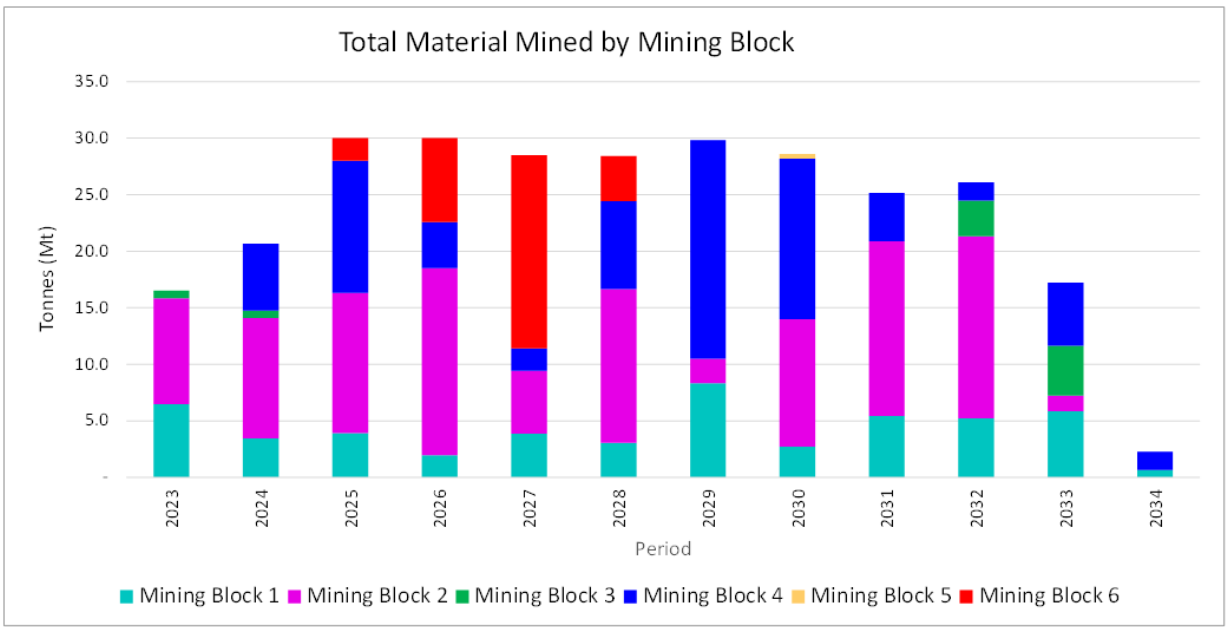
Source: AMC, 2023

16.3.4 Mine Sequence

The mining sequence was developed based on value and within the access constraints. Current operations will continue in Block 1 (Maga) and Block 2 (CFU and P8P9) from 2023. Pit 31, located in Block 3 (P11) near the hard rock plant, will be mined and backfilled by the end of 2024. Block 4 (Siga) will commence in 2024, followed by Block 6 (P17) in 2025.

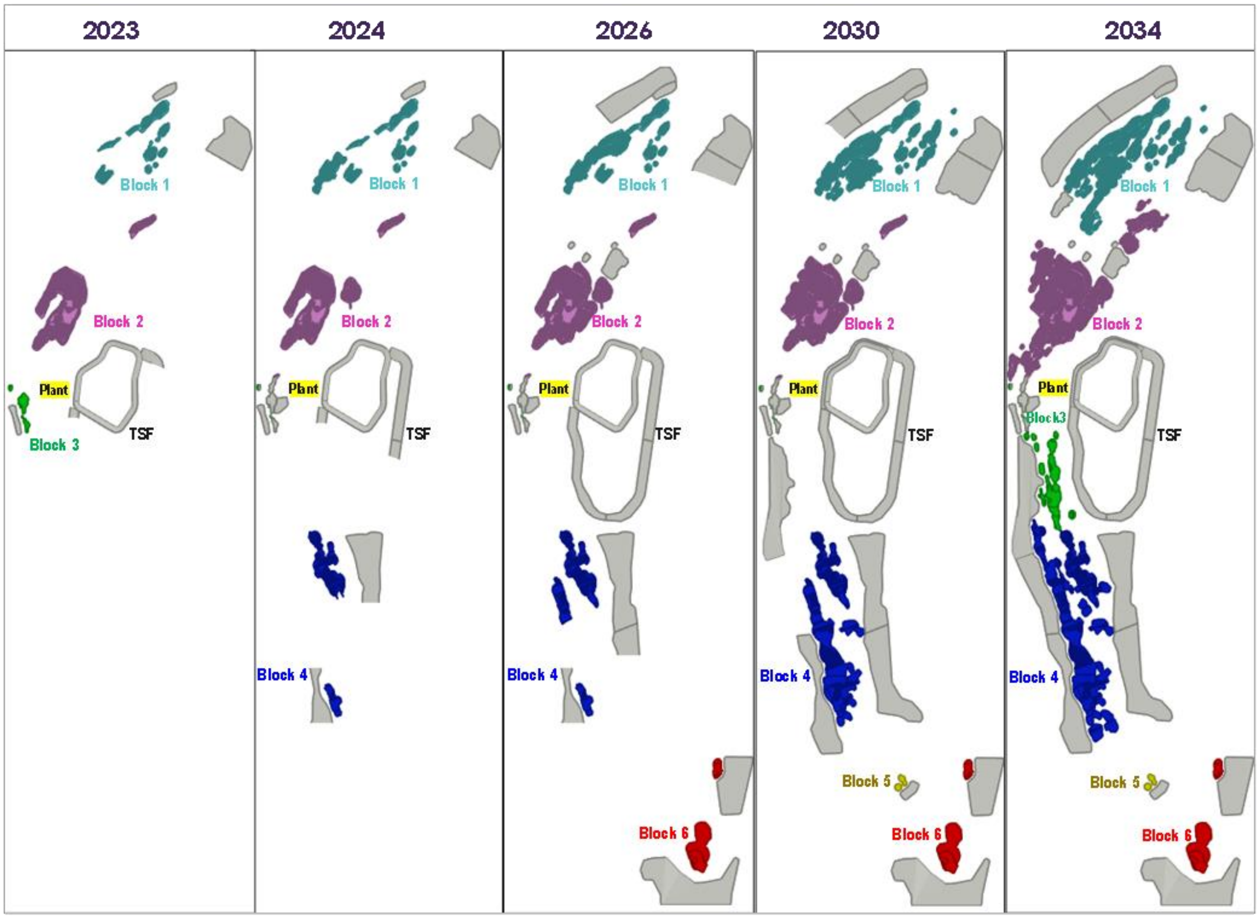
Figure 16.3.5 illustrates the total material mined by block. Figure 16.3.6 illustrates mine progression images for the end-of-period maps of years 2023, 2024, 2026, 2030, and 2034. However, these pictures should not be construed as detailed end of period mine advance maps.

Figure 16.3.5 Annual Total Material Mined by Mining Block



Source: AMC, 2023

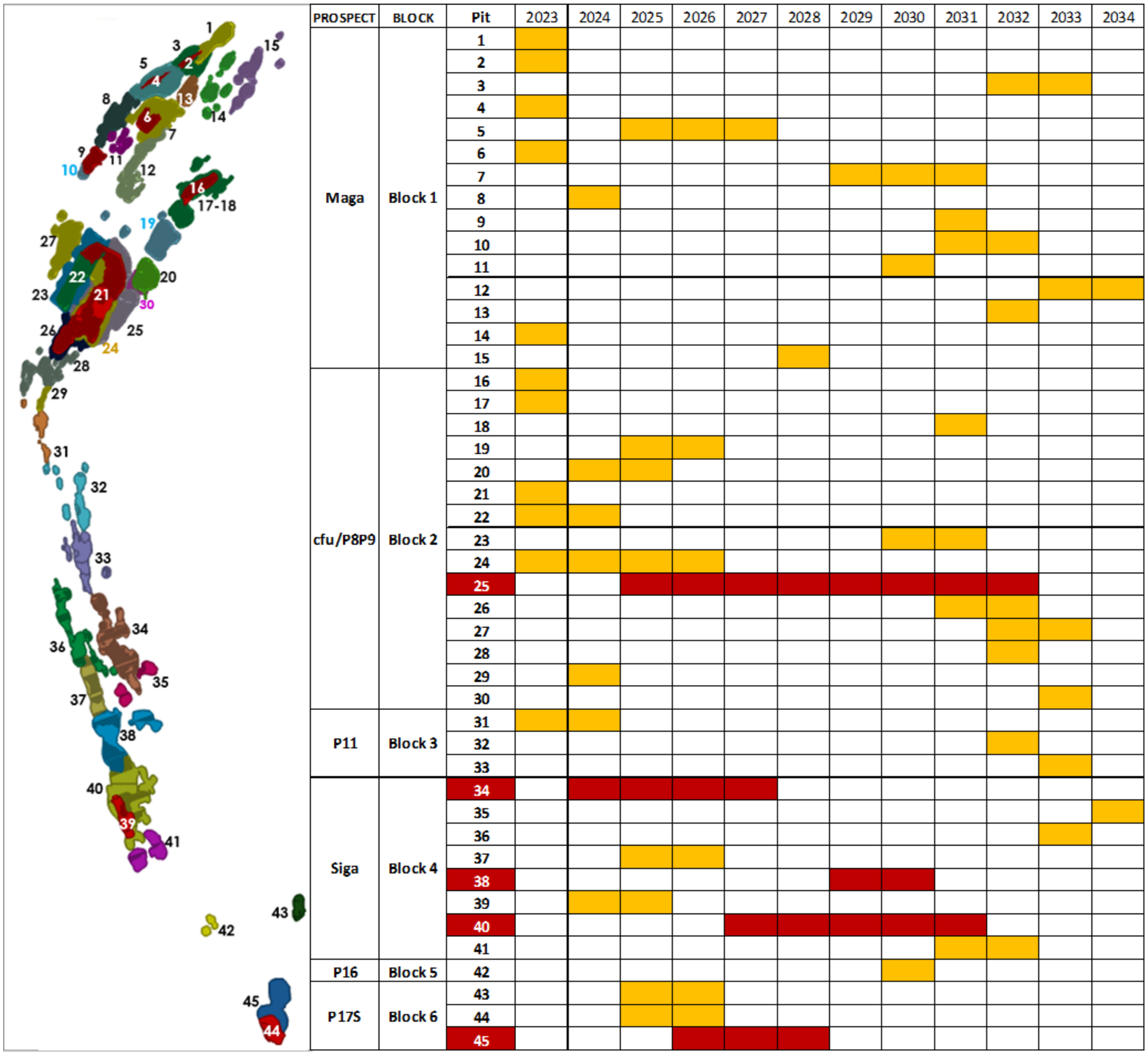
Figure 16.3.6 Period Map



Source: AMC, 2023

The mining sequence of the 45 mining areas is summarized in Figure 16.3.7. The mining areas highlighted in red colour contain more than 2.0 Mt of hard rock ore.

Figure 16.3.7 Mining Sequence by Mining Area



Source: AMC, 2023

16.3.5 Stockpile Management

The grade bins used for ore stockpiling are presented in Table 16.3.4.

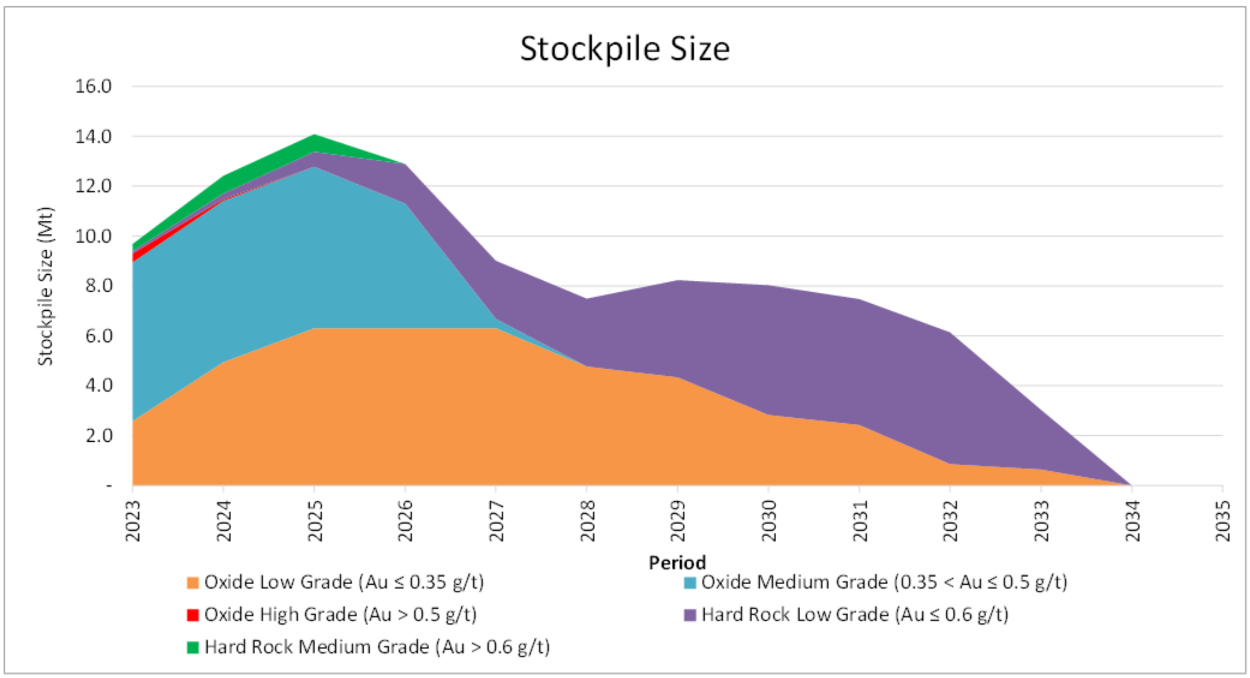
Table 16.3.4 Stockpiles Grade Bins

	Oxide	Hard rock
Low grade	Au g/t ≤ 0.35	Au g/t ≤ 0.60
Medium grade	0.35 < Au g/t ≤ 0.50	Au g/t > 0.60
High grade	Au g/t > 0.50	N/A

Source: AMC, 2023

Six stockpiles are situated adjacent to mining areas throughout the site. Lower and medium grade ore will be sent to the nearest available stockpile. The combined tonnage of these stockpiles is expected to reach approximately 14.1 Mt by 2025. (Figure 16.3.8).

Figure 16.3.8 Total Stockpile Size



Source: AMC, 2023

The oxide high-grade stockpile consists of ore that was already stored in the existing stockpiles as of 28 March 2023. All the high-grade oxide ore mined is directly fed from the pits to the oxide plant.

The construction of the hard rock medium-grade stockpile occurs between 2023 and 2024, before the commissioning of the hard rock plant. No additional hard rock medium-grade ore is stockpiled after that.

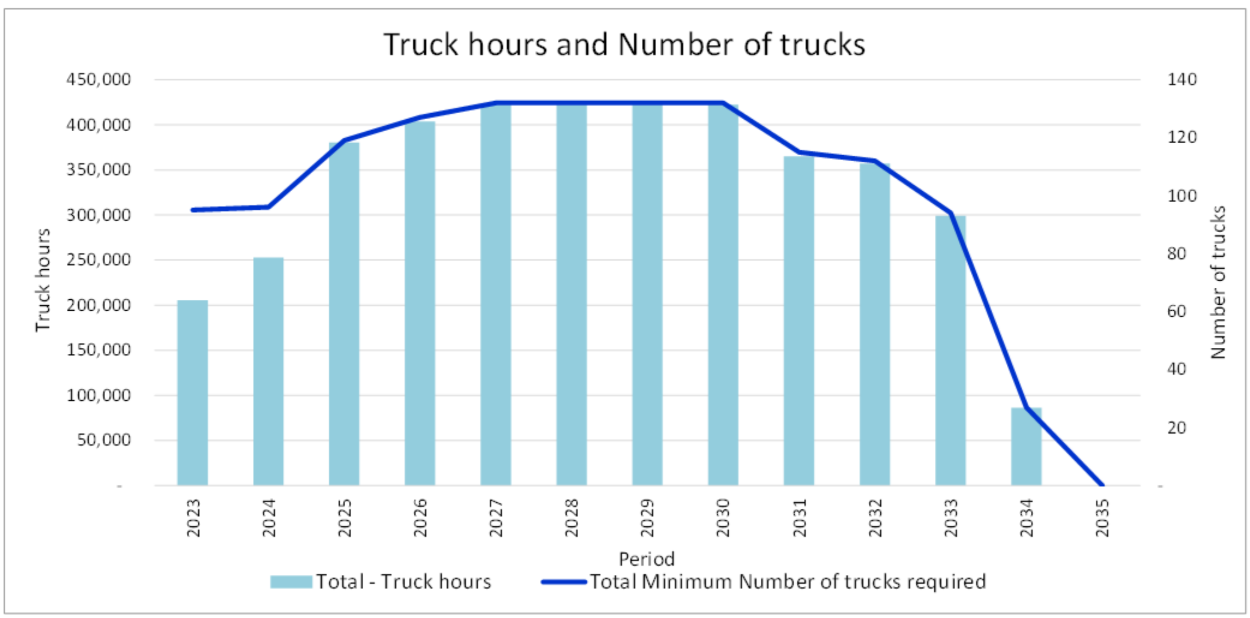
16.4 Mine equipment requirements

AMC estimated the mine equipment requirements by considering the production schedule and the equipment capacity. The average excavator productivity for oxide material is estimated to be 1.9 Mt per year and for hard rock it is 2.0 Mt per year.

Haulage distances and cycle times from pits to all possible destinations were modelled using the Haul Infinity software package. AMC estimated the truck hours based on rigid frame dump trucks with a 26 t payload capacity. Currently the contractor has assorted equipment models deployed on site. Orezone is considering the application of trucks with higher payload of 30 to 60 t for all material types as part of the hard rock expansion.

The chart in Figure 16.4.1 displays the number of trucks required and resulting truck hours. The minimum number of trucks needed was estimated based on 3,200 operating hours per year per truck, using current mining operational experience as a reference.

Figure 16.4.1 Truck Hours and Truck Requirements



Source: AMC, 2023

The mining fleet (see Table 16.4.1) was estimated by AMC based on the existing fleet and contractor quotation. The final equipment specification will be established during contractor negotiations.

Table 16.4.1 Mining Equipment

Equipment	Number required 2023 to 2024	Maximum number required from 2025
Oxide Excavator (e.g. Komatsu PC850)	12	9
Hard rock Excavator (e.g. Komatsu PC1250)	1	9
Haul Truck (26 t Rigid Truck)	96	132
Bulldozer (e.g. CAT D7 or D8R)	13	16
Front End Loader	3	5
Drills	1	8
Water Truck (25,000 L)	7	8
Grader (e.g. CAT 140M)	5	7
Compactor	2	2
Lowbed Transport Truck	2	2
Crew Bus (50 person)	5	7
Fuel Truck (15,000 L)	2	4
Crane (40 t)	1	2
Service Vehicle	3	5
Potable Light Towers	20	25
Light Vehicles (e.g. Toyota Hilux)	25	35
Mine Rescue Truck	2	2

Source: AMC, Orezone, 2023

16.5 Fuel Consumption

Fuel requirements for the mining equipment have been estimated based on the mine production schedule and fuel consumption assumptions listed in Table 16.5.1.

Table 16.5.1 Fuel Consumption Parameters

Equipment	Units	Value
Oxide excavator (e.g. Komatsu PC850)	L/hr	57
Hard rock excavator (e.g. Komatsu PC1250)	L/hr	77
Haul truck (26 t Rigid Highway Truck)	L/hr	15

Source: AMC, 2023

The annual fuel requirements for mobile equipment are summarized in Table 16.5.2. Fuel consumption for support equipment is estimated at 15% of total fuel requirements for excavators and trucks. The mining fleet will consume a total of 127 ML of fuel over the life of the mine.

Table 16.5.2 Fuel Requirements for Mining Fleet

Equipment	Unit	Total	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Trucks	ML	60.6	3.1	3.8	5.7	6.1	6.3	6.3	6.3	6.3	5.5	5.4	4.5	1.3
Excavators	ML	49.7	2.6	3.2	4.8	5.4	5.7	5.2	5.4	5.4	4.4	4.5	2.8	0.4
Support equipment	ML	16.5	0.8	1.0	1.6	1.7	1.8	1.7	1.8	1.8	1.5	1.5	1.1	0.2
Total	**ML**	**126.9**	**6.5**	**8.0**	**12.1**	**13.2**	**13.8**	**13.3**	**13.4**	**13.5**	**11.4**	**11.4**	**8.4**	**1.9**

Source: AMC, 2023

16.6 Explosives Consumption

Drill and blast consumables are based on the production schedule and estimated drill patterns (see Table 16.1.2) and are summarized in Table 16.6.1. Blasthole drilling will be performed by a contractor while blasting will be under a 'load and shoot' contract with an international explosives supply company.

In the peak year of 2027, about four to five blasts per week of 100 to 125 kt each will be required. These blasts will be spread between multiple pits excavating fresh rock such as pits 25, 34, 40, and 45.

Table 16.6.1 Drill and Blast Requirements

Blast Requirements	Unit	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Total mined	Mt/yr	16.5	20.7	30.0	30.0	28.5	28.4	29.8	28.6	25.2	26.1	17.2	2.3
Total blasted	Mt/yr	2.7	3.4	7.9	17.2	25.1	17.8	16.8	20.3	12.4	12.2	5.3	0.7
Drill metres	M	63,800	79,920	186,960	416,300	609,170	430,330	407,030	492,520	299,370	292,810	127,130	16,000
Powder factor	kg/t	0.23	0.23	0.24	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.24	0.24
Emulsion explosives	t	640	800	1,890	4,260	6,260	4,410	4,170	5,050	3,060	2,990	1,290	160

Source: AMC, 2023

Explosives will be stored on-site at a magazine situated between WD15 and SP1 in the NE sector of the property. The explosives supplier will manage the magazine and ensure the requisite explosive products are maintained on site.

16.7 Surface Haul Road Development

The secondary haul roads from the primary trunk road to the pits and from pits to WRDs and stockpiles will be developed as needed based on the production schedule. Table 16.7.1 summarizes the construction of new secondary haul roads.

Table 16.7.1 Secondary Surface Haul Road Schedule

	Unit	Total	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Total	km	95	0*	13	15	4	4	4	6	8	13	15	11	2

Note: * Existing roads in use, no new road construction required for 2023.

Source: Orezone, 2023

16.8 Mining Personnel

Orezone will maintain an Owner's Team on site, responsible for site management, contractor management, grade control, and mine planning. Orezone informed that in Q3 2023, the Owner's Team had approximately 170 on site in the mining department. This is considered reasonable and suitable for the current operations. The owner's team will be expanded in parallel as the operations move from oxide material mining only to oxide and hard rock mining operations and tonnages increase from 2025.

The Owner's Mine Management and Technical Services team shall be responsible for ore control, mine planning, blast designs, survey, road layout, water management designs and all other technical aspects of the mine. Record keeping, reporting and reconciliation tasks will be performed by the owner's team. The Owner's Mine Production Personnel will be responsible for oversight of the contractor in pit mining activities as well as providing ancillary support and management of pit dewatering.

Mining contractors will provide on-site administrative and management personnel along with required equipment operators, mechanics, welders, electricians, and labourers sourced from the local communities to achieve the scheduled mining production. Hourly contractor employees will work a 12 h shift of seven days on and seven days off. Four work crews will rotate to provide 24/7 continual operations. Mining contractor operations currently employ approximately 750 personnel on site for operations, maintenance, and management. At peak production approximately 1,100 contractor personnel are estimated.

16.9 Mining Risks & Opportunities

In AMC's opinion, the risks to the mine plan and associated estimates are:

- Metal price decrease.
- Block model inaccuracy including mining factors, artisanal depletion, and classification.
- Mixing and inherent dilution and losses from blasting.
- Productivity lower than planned with associated detrimental economic impacts.
- Final pit wall slopes unstable at planned design.

- Greater than expected water inflows from surface or groundwater sources.

The key opportunities are:

- Metal price increase.
- Ongoing optimization of mine plans and productivities.
- Increase in ore tonnes from ongoing exploration and conversion of Inferred Resources to Measured and Indicated Resources.
- Re-schedule certain pits.

The risks and opportunities identified by AMC are highlighted in the following sections.

Risks

Metal Price Decrease:

Being a low-grade mine, the pit size and project economics are sensitive to gold metal prices.

- Mitigation: Strive for reduced costs. Maintain stockpiles at various grade bins to help in adjusting plant feed grade and cut-off in concordance with metal price fluctuations. Investigate pit pushback options to tailor pit size and stripping ratio according to prevailing economics.

Block Model Inaccuracies:

Since the mine plan is based on the block model, if the model is optimistic and overstating ore tonnes or grade then the mine plan will similarly be optimistic. Misclassification of Measured and Indicated Resources and the associated Proven and Probable Reserves, is not a direct risk to project feasibility as both are considered as 'ore'. However, for certain financial situations, there may be an impact if Measured/Proven ore is overstated.

- Mitigation: Conduct regular reconciliation analysis between the block model, ore control, dispatch, and mill results. A regular comprehensive reconciliation assessment will show how the block model is performing with respect to the final mill results and allow for applying compensating factors to the mine plan. As well, block model construction can likely be continually improved as knowledge is gained. Align classification methods across all block models and standardize block model procedures. AMC applied mitigation measures in this study by drafting polygons around contiguous Measured blocks. Only Measured blocks inside these polygons were converted to Proven Reserves. The Orezone team do complete reconciliation on a regular on-going basis.

Blast Induced Movement and Mixing:

Blasting will cause movement and mixing of ore and waste across contacts thus increasing dilution and ore losses.

- Mitigation: Employ mining procedures that reduce blast movement such as reduced powder factor, blasting along strike and choked blasting. Adjust dig packets for blast movement vectors. Excavate from the hanging wall side towards the foot wall. Maintain vigilant geological inspections and control.

Lower Productivities:

Although the operating mining cost is a set contract rate, lower productivities than planned will put pressure on achieving mining rates and mill feed. The contractor will need to increase the fleet size and might suffer financially.

- Mitigation: Investigate the use of larger more reliable equipment. Maintain good maintenance practices and supply lines. Reduce all waiting times and promote supervision and practices that enhance machine utilization doing productive work including reducing re-handle. Investigate reducing haul distances and machine moves.

Pit Wall Instabilities:

The final pit wall slopes, particularly in the Fresh rock, are predicated on achieving excellent wall conditions after measures such as pre-splitting and wall scaling. As well, the geotechnical parameters (rock strength, pore pressures, structures etc) need to conform to projections. Flatter slopes will be required if these conditions are not met.

- Mitigation: Ongoing geotechnical monitoring and studies should be conducted to update the geotechnical knowledge. Optimize control blasting of the walls. Reduce water infiltration behind the pit walls. Investigate the use of support such as bolting etc. Install wall movement monitors and employ qualified engineers to manage wall stability.

Higher Water Inflows:

During extended wet periods or flood events, there is potential that the water management system is insufficient to cope with the water flows. Sub-surface groundwater flows may also be higher than anticipated. The risk is that pits will be inaccessible for longer periods than desirable.

- Mitigation: Maintain ore in stockpiles sufficient to feed the mill in case of rain events. Ensure roadways are elevated with adequate drainage ditches, culverts etc. Have road sheeting available to apply on critical roadways. Have adequate pump and pipeline resources available. Divert surface flows away from the pits. Maintain ore faces in pushbacks on upper levels out of the flooded pit bottoms.

16.9.1 Opportunities

Metal Price Increase

Being a low-grade mine, the pit size and project economics are sensitive to metal prices. Should gold price increase the mine may readily expand as well as the cut-off grade decrease. Ensure that infrastructure and dumps, etc. do not inhibit pit expansions. Update the long-term mine plans periodically in conjunction with metal prices in order to understand and be prepared for adjustments to plan.

Ongoing Optimization of Mine Plans and Productivities

Continual optimization efforts will provide reduced costs, higher efficiencies and thus permit a lower cut-off grade and potential mine extensions. All aspects of the planning and operations can benefit from ongoing optimization.

Increased Resources and Reserves

Further drilling may result in increased Mineral Reserves if Inferred Resources can be converted to Measured and Indicated Resources. As well new zones may be discovered.

Re-Schedule Certain Pits

Pit 31, located under the hard rock ROM pad is a lower value pit. Scheduling this pit later in the mine life would add value to the project. Given the large number of pits, constraints and alternatives, there may be other minor schedule adjustments found that could further optimize the schedule.

17.0 RECOVERY METHODS

17.1 Summary

The total Bomboré processing capacity will be 10.3 Mtpa. The existing Oxide plant will process the Oxide and Upper Transition ores at a nominal rate of 5.9 Mtpa, which is 0.7 Mtpa above the nameplate capacity (5.2 Mtpa nameplate) as reported by the current Bomboré plant operations. A new 4.4 Mtpa Hard Rock plant will process the Lower Transition and Fresh ores. Gold will be recovered from the Bomboré ore based on conventional unit operations with metallurgical flowsheets developed for optimum recovery while minimizing initial capital expenditure and life of mine operating cost.

The trucks transporting ore to the Oxide plant currently rear-dump the ore onto a static grizzly and into the receiving bin. The grizzly is kept clear, as necessary, by a front-end loader. The soft saprolite ore is introduced into a MMD sizer via an inclined apron feeder and then fed by conveyor into a single stage 3,200 kW ball mill, in closed circuit with hydrocyclones, to produce a grind size of 80% passing 125 microns (P_{80} 125 µm). Lime is added onto the conveyor belt to maintain the pH of downstream slurry. The ball mill discharge slurry is screened with a trommel, and oversize pebbles are dropped into a bunker for manual removal. Cyclone overflow is screened to remove trash and is pumped to the leach circuit at a slurry density of 40% w/w solids. One pre-oxidation tank and seven CIL tanks achieve 21 hours of residence time for gold recovery. Liquid cyanide is pumped to the leach circuit to leach the gold. Activated carbon is used to adsorb the gold out of the slurry and loaded carbon is acid washed and pressure stripped in a Zadra elution circuit. A carbon regeneration kiln removes organic foulants from the carbon and reactivates the carbon surface. Gold is recovered in electrowinning cells and is smelted in an electric furnace. The final product is doré bullion bars.

The trucks transporting ore to the Hard Rock plant will rear-dump the ore onto a static grizzly and into the receiving bin. There will be two dump pockets to facilitate simultaneous dumping. The grizzly will be kept clear, as necessary, by a front-end loader. The hard ore will be delivered to a jaw crusher via an inclined apron feeder where it will be crushed to minus 314mm and conveyed to a 24h capacity crushed ore stockpile. Ore will be reclaimed via two apron feeders and then fed by conveyor into a single stage 18,000 kW SAG mill, in closed circuit with hydrocyclones, to produce a grind size of P_{80} 75 µm. Lime will be added onto the conveyor belt to maintain the pH of downstream slurry. The SAG mill discharge slurry will be screened over a vibrating horizontal screen and the oversize pebbles will be conveyed back to the SAG mill feed. Cyclone overflow, at a slurry density of 30% w/w solids, will flow by gravity to trash removal screens and then to a 29m diameter thickener. The slurry will be thickened to a density of 45% and pumped to the leach circuit. One pre-oxidation tank and seven CIL tanks achieve the required 24 hours of residence time for optimum gold recovery. Liquid cyanide is pumped to the leach circuit to leach the gold. Activated carbon is used to adsorb the gold out of the slurry and loaded carbon is acid washed and pressure stripped in a Zadra elution circuit. The existing carbon regeneration kiln will reactivate the carbon and the existing gold room will recover gold in two new electrowinning cells and the existing furnace will produce doré bullion bars.

17.2 Hard Rock Plant Process Design

The Hard Rock plant design is based on a metallurgical flowsheet developed for maximizing recovery while minimizing initial capital expenditure and life of mine (LOM) operating costs. The flowsheet is based on unit operations including crushing, milling, pre-oxidation, Carbon-in-Leach (CIL), Zadra elution, gold electrowinning and carbon regeneration that are well proven in the industry.

Process plant feed will comprise of two main ore types, Fresh and Lower Transition ores, which will be mined by drilling and blasting as required. Study design documents have been prepared by incorporating engineering and metallurgy design criteria derived from metallurgical samples taken from the five main mining locations, namely Maga, P8P9, P16, Siga and P17. The metallurgical testwork programs are summarized in Section 13 of this Technical Report.

The process plant design has been based on a nominal capacity of 4.4 Mtpa. Plant feed will consist of a blend of 15% Lower Transitional material and 85% Fresh material.

For simplicity, the portion of the plant currently operating and processing only the Oxide and Upper Transition ore will be referred to as the "Oxide plant" or "Oxide circuit", and the portion of the plant that is required for the inclusion of Lower Transition and Fresh ore will be referred to as the "Hard Rock plant" or "Hard Rock circuit".

The key criteria for selection of equipment type are suitability for duty, reliability, compatibility with existing Oxide plant equipment and ease of maintenance, with price then being a major criterion for selection between vendors of broadly similar equipment. The plant layout provides ease of access to equipment for operating and maintenance requirements while maintaining a layout that will facilitate construction progress in multiple areas concurrently as well as minimum interruption of Oxide plant operation.

The key design criteria for the plant are:

- Hard Rock plant throughput of 4.4 Mtpa with an ore blend of 85% w/w Fresh and 15% w/w Lower transition.
- Hard Rock crushing plant utilization of 75%.
- Hard Rock plant grinding and CIL circuit utilization of 91.3%, supported by the incorporation of surge capacity and standby equipment where required.
- Sufficient automated plant control to minimize the need for continuous operator interface and allow manual override and control if required.

17.2.1 Selected Process Flowsheet

The Hard Rock plant will process 4.4 Mtpa Lower Transition and Fresh ore. A LOM milling and gold production schedule is provided in Section 22 of this Technical Report.

The treatment plant design incorporates the following process unit operations:

Hard Rock Plant

- ROM ore fed through a static grizzly.
- Apron feeder, followed by a vibrating grizzly.
- Primary crushing with a single toggle jaw crusher to produce a P_{80} 175 mm.
- A crushed ore stockpile of 15,000 t to provide 24 hours live surge capacity. Mechanical equipment in the crushing area will treat 4.4 Mtpa nominally but is designed for 5.0 Mtpa.
- The grinding circuit consists of a closed-circuit single stage SAG mill to produce a final P_{80} 75 µm. Provision has been made to install a pebble crusher in the future should additional throughput capacity be desired. Mechanical equipment in the pebble circuit will treat 4.4 Mtpa nominally but it is designed for 5.0 Mtpa. In addition, layout space has been kept for a future ball mill and gravity gold circuit.
- A hydrocyclone pack with an overflow slurry density of approximately 30% w/w solids to maintain efficient particle size separation.
- A pre-leach thickener to increase leach slurry density, which in turns also minimizes leach tank volume and reduces overall reagent consumptions.
- A leach circuit with a pre-oxidation tank followed by seven CIL tanks to provide 24 hours of residence time for optimum recovery. Layout allowance is kept for one future CIL tank.
- Loaded carbon acid wash and pressure Zadra elution circuit with gold electrowinning and recovery to doré.
- Oxide plant carbon regeneration kiln will be shared to remove organic foulants from the carbon and reactivate the adsorption sites on the activated carbon.

A simplified overall process flow diagram depicting the unit operations incorporated in the Hard Rock plant is presented in Figure 17.2.1. Plan and isometric views of the process plant are provided in Figures 17.2.2 and 17.2.3.

Figure 17.2.1 Simplified Overall Process Flow Diagram for Hard Rock Plant

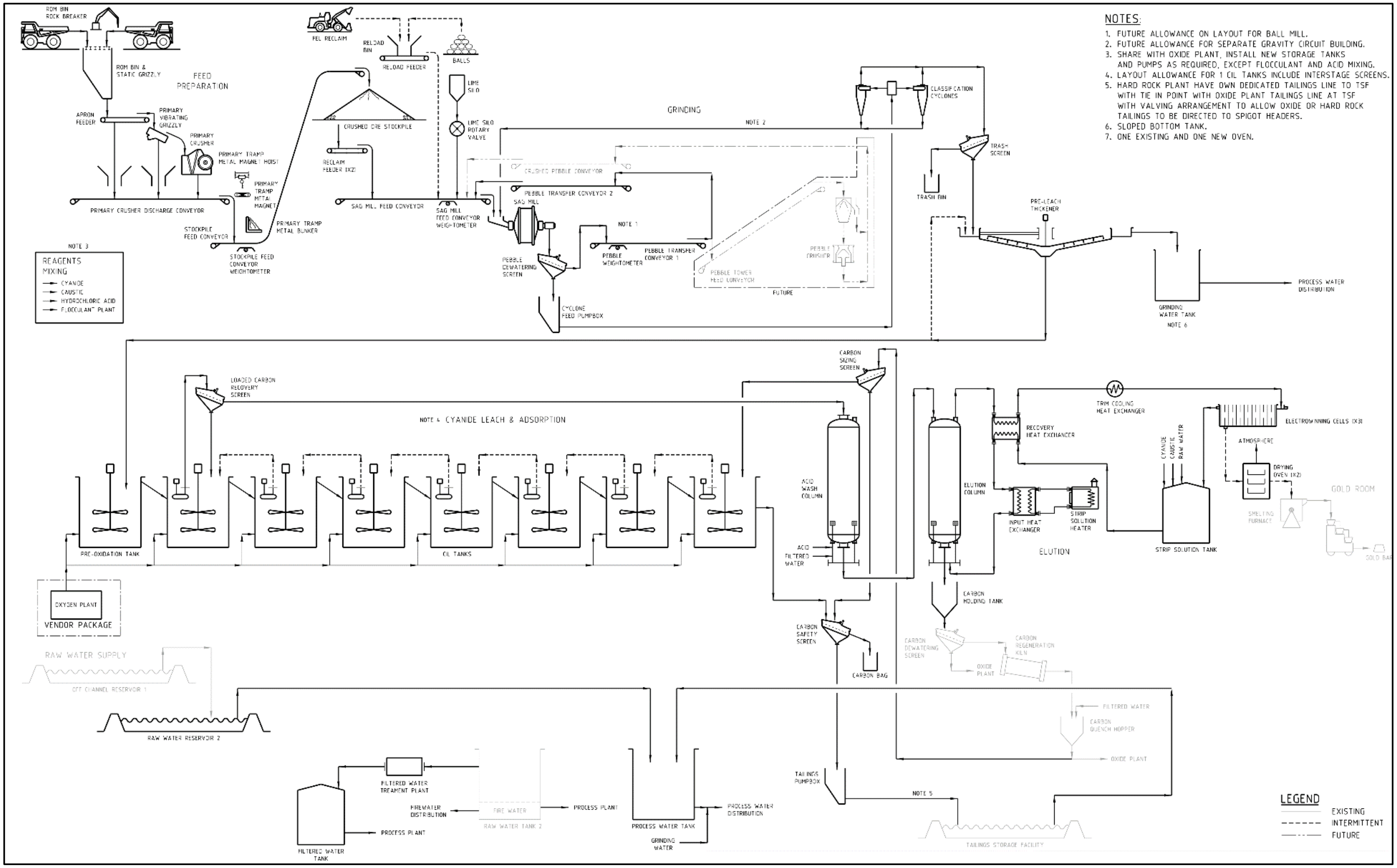


Figure 17.2.2 Hard Rock Plant Plan View

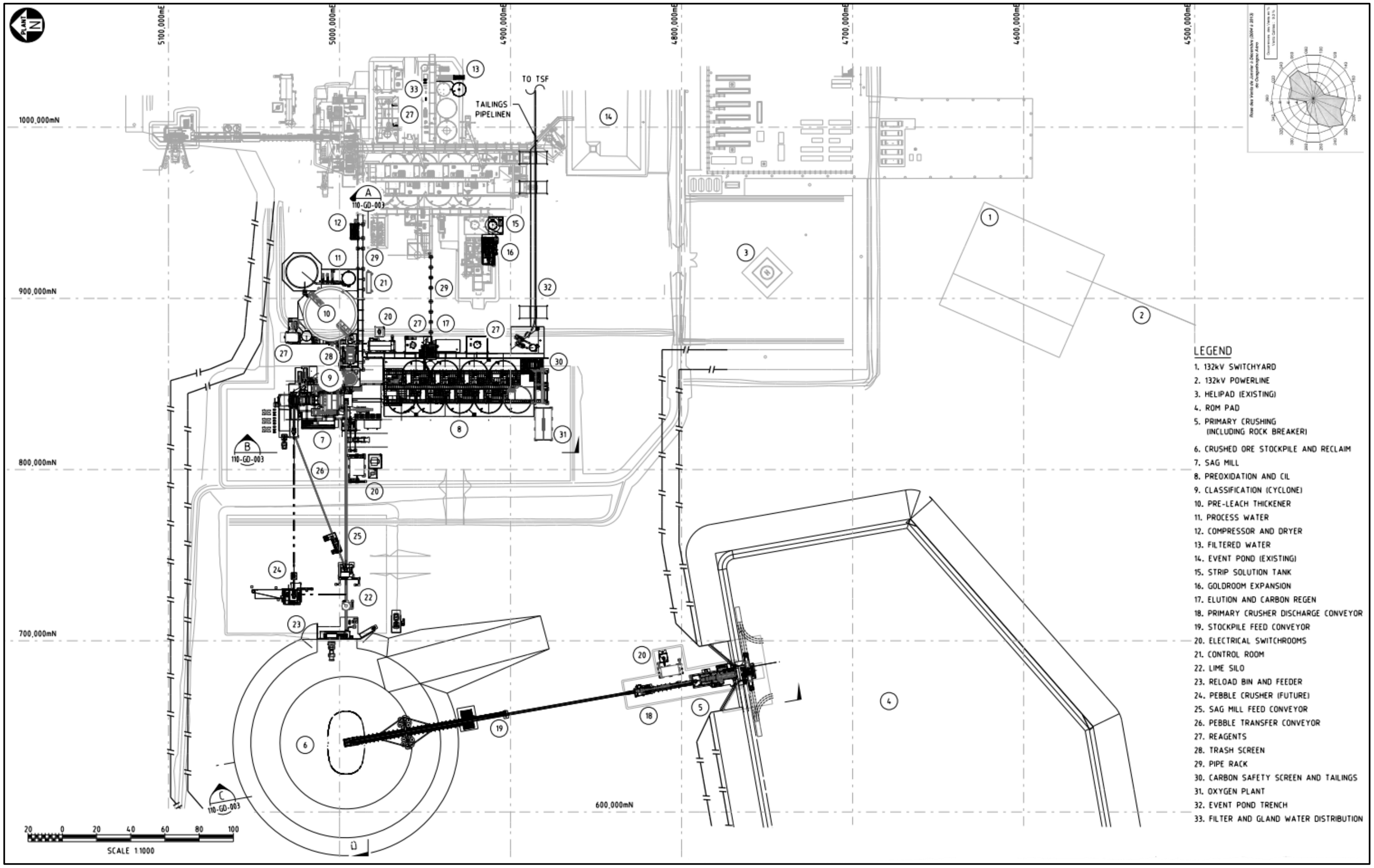
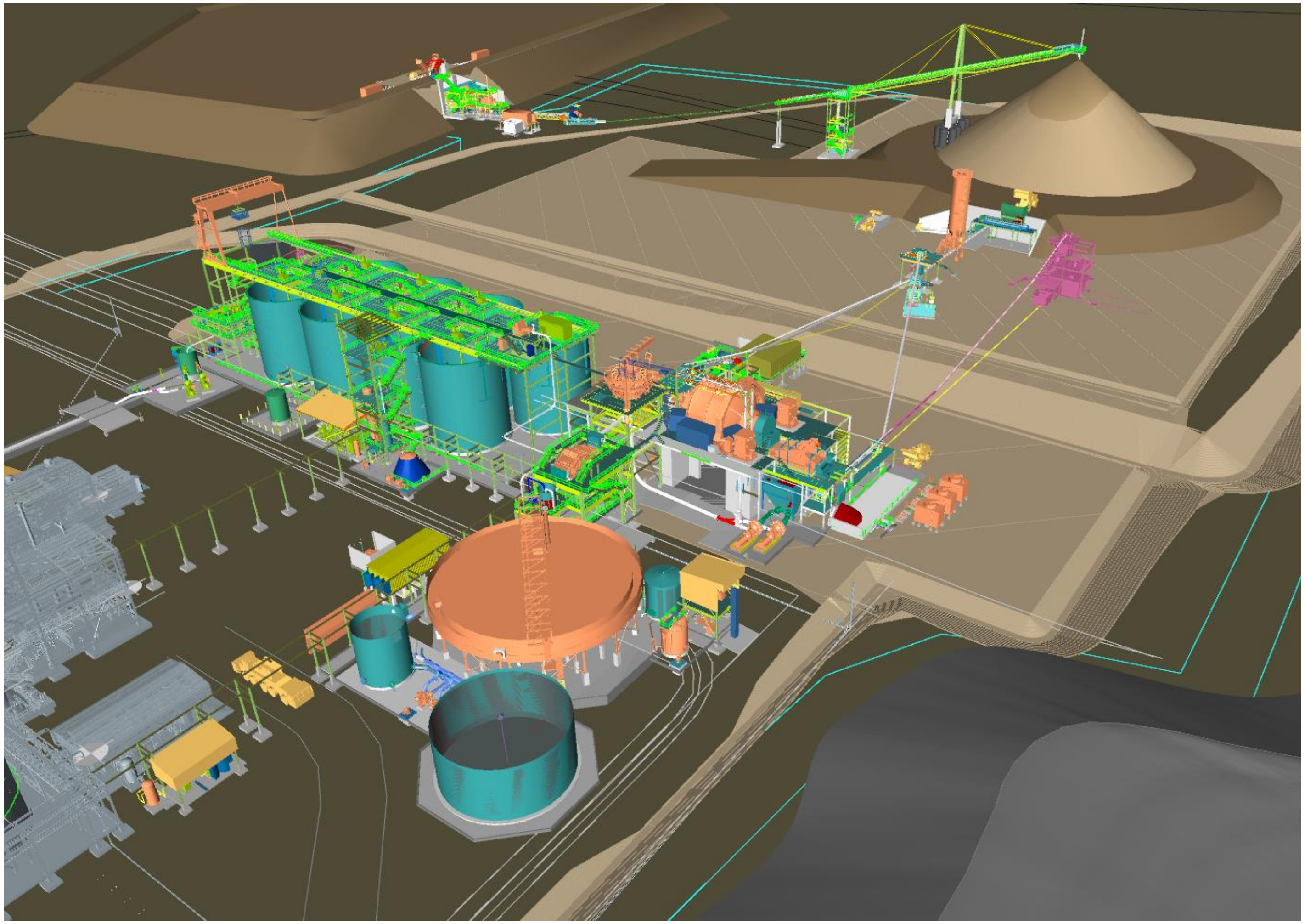


Figure 17.2.3 Hard Rock Plant Isometric View



17.2.2 Key Process Design Criteria

Key process design criteria for the Hard Rock plant are summarized in Table 17.2.1.

Table 17.2.1 Summary of Key Process Design Criteria for Hard Rock Plant Only

Description		Units	Design	Source
Plant Throughput		tpa	4,400,000	Orezone
		tpd	12,055	Calculated
Design Ore Blend	- Fresh	%	84.9	Mine Plan (2022)
	- Lower Transition	%	15.1	Mine Plan (2022)
Head Grade	- Gold (Design)	g Au/t	1.90	Lycopodium / Orezone
	- Gold (LOM)	g Au/t	1.02	Orezone
Overall Gold Recovery[1]		%	82.4	Testwork / Mine Plan (2022)
Ore Specific Density		t/m^3	2.79	Testwork
Ore Bulk Density		t/m^3	1.65	Lycopodium / Orezone
Angle of Repose		degrees	37.0	Lycopodium
Plant Availability		%	91.3	Lycopodium
Crushing Work Index (CWi, 85th Percentile)		kWh/t	16.8	Testwork
Bond Ball Mill Work Index (BWi, 85th Percentile)		kWh/t	15.7	Testwork
Bond Abrasion Index (Ai, average)			0.374	Testwork
Grind Size (P_{80})		µm	75	Testwork
Pre-leach Thickener Solids Loading[2]		$t/m^2.h$	1.0	Testwork
CIL Circuit Residence Time		hrs	24	Testwork
CIL Slurry Density		% w/w solids	45	Testwork
Number of Pre-oxidation Tanks			1	Lycopodium
	Oxygen Addition Rate	mg/L/min	0.129	Testwork
Number of CIL Tanks (Stages)			7	Lycopodium
	Oxygen Addition Rate	mg/L/min	0.092	Testwork
Sodium Cyanide Addition		kg/t ore	0.69	Testwork / Calculated
Lime Addition[3]		kg/t ore	0.98	Testwork / Calculated
Sodium Hydroxide Addition		kg/strip	1,021	Lycopodium
Elution Circuit Type			Pressure Zadra	Lycopodium / Orezone
Elution Circuit Size		t	12.0	Lycopodium
Frequency of Elution		strips/wk	7	Lycopodium
Existing Kiln Capacity (Design)		kg/h	800	Oxide Plant
Frequency of Smelting		smelt/wk	2	Lycopodium

1. At LOM head grade of 1.02 g Au/t including 0.021 g Au/t CIL solution and carbon fines losses.

2. Pre-leach thickener solids loading is based on testwork of SIGA and P17 materials.
3. Lime addition based on 90% w/w CaO content of supplied quicklime.

17.3 Hard Rock Plant Description

17.3.1 Ore Receiving and Crushing

ROM ore from the open pits will be transported to the crusher by 35t capacity rear dump trucks with a back-to-back tip arrangement into the ROM bin. Dust suppression with fog sprays will be automatically activated during tipping. Extraction of the dust from the discharge points below the crusher will be by dry dust extraction systems.

A static grizzly (800 mm x 800 mm), mounted above the ROM bin, will prevent the ingress of oversize material. A fixed rock breaker will be utilized to break oversize material retained on the static grizzly. Ore will be drawn from the ROM bin by a variable speed apron feeder to the vibrating grizzly which will direct oversize materials to the primary crusher operating in an open circuit. The vibrating grizzly's undersize will report directly to the primary crusher discharge conveyor and combine with the primary crushed ore and sent to the crushed ore stockpile via the stockpile feed conveyor.

The stockpile feed conveyor will be fitted with a weightometer to monitor and control the crushing area throughput by adjusting the output of the apron feeder's variable speed drive.

A static primary tramp metal magnet will be installed at the discharge end of the primary crusher discharge conveyor. Tramp metal will be manually removed from the magnet when necessary and stored in a primary tramp metal bunker.

The material handling and primary crushing circuit includes the following key equipment:

- Rom bin with static grizzly.
- Apron feeder with dribble conveyor.
- Fixed rock breaker.
- Primary vibrating grizzly.
- Primary jaw crusher.
- Primary crusher discharge conveyor and stockpile feed conveyor.
- Primary tramp metal magnet, hoist, and bunker.

17.3.2 Crushed Ore Stockpile and Mill Feed

The crushed ore stockpile will have a live capacity of 15,000 t to provide approximately 24 hours of plant feed. Primary crushed ore will be reclaimed with two variable speed reclaim feeders (both running with the option for one to do the full capacity while the other is on standby) onto the SAG mill feed conveyor. A dry dust extraction system will extract dust from the SAG mill feed conveyor. The SAG mill feed conveyor will be fitted with a weightometer, used for metal accounting, and controlling the speed of the reclaim feeder and hence the feed rate to the grinding circuit. A reclaim tunnel ventilation fan will be installed to ensure a constant supply of fresh air in the crushed ore reclaim tunnel. A sump with a installed sump pump will collect and recover slurry collected within the crushed ore reclaim tunnel and will be pumped to the cyclone feed pumpbox.

A reload bin will be installed over the mill feed conveyor, where crushed ore and grinding media (ball) will be fed as required. Material from this bin will be reclaimed with a variable speed reload feeder onto the SAG mill feed conveyor. Grinding media (125 mm balls) will be added as required to maintain the required ball charge in the mill.

Quicklime will be added directly to the SAG mill feed conveyor via a variable speed lime silo rotary lime valve. The lime silo will have a storage capacity of 68 t which is equivalent to 5.3 days of storage at nominal throughput. To mitigate the risk of lime shortages, a small inventory of 1 t quicklime bags will be kept in storage.

Water sprays will be used to suppress dust generated at the crushed ore stockpile.

The crushed ore reclaim circuit includes the following key equipment:

- Two reclaim feeders.
- SAG mill feed conveyor.
- Reload hopper and feeder.
- Lime dosing system.
- Reclaim area sump pump.

17.3.3 Grinding and Classification

Crushed ore, reclaimed from the crushed ore stockpile, will be conveyed to the SAG mill feed chute via the SAG mill feed conveyor. Process water will be added to the SAG mill feed chute to adjust the in-mill pulp density.

The grinding circuit will be a SS-SAG circuit comprising a single stage variable speed SAG mill. The SAG mill will operate in closed circuit with hydrocyclones while pebbles will be removed by a pebble dewatering screen and recycled back to the SAG mill feed conveyor via a series of pebble transfer conveyors. The final product (cyclone overflow) will have a targeted P_{80} of 75 µm. To achieve the required product size when treating ore at the 85th percentile of hardness, a dual pinion 18MW, 10.97m x 7.87m SAG mill (36′ dia) will be required.

SAG mill discharge will gravitate to a pebble dewatering screen from where oversize will be recycled back to the SAG Mill feed conveyor. Undersize from the pebble dewatering screen will gravitate to the cyclone feed pumpbox prior to being pumped to the classification cyclone cluster by variable speed duty / standby cyclone feed pumps. The classification cyclones' overflow will gravitate to the pre-leach thickener feed distribution box via a trash screen. Trash screen undersize will gravitate to the pre-leach thickener, whilst trash screen oversize will be discharged to a trash bin. The classification cyclone underflow will gravitate back to the SAG mill feed chute.

The classification cyclone cluster will consist of 18 installed 15″ cyclones. Fourteen of the 18 cyclones will be operating, while the remaining four will be installed as spares to allow in-line maintenance of the cyclones.

Spillage within the grinding circuit will be managed through a drive-in sump with a sump pump arrangement. A front-end loader will be used to clean out the sump as required while slurry will be pumped back to the cyclone feed pumpbox. In addition to the drive-in sump, another sump will be used in the mill feed end and accumulated slurry will be discharged into the cyclone feed pumpbox. During flooding events the excess water will gravitate via trenches to the Hard Rock plant CIL bund and eventually to the Oxide plant event pond.

Provision has been incorporated in the layout for possible future installation of a pebble crushing circuit, a stand alone gravity gold recovery circuit, and a ball milling circuit.

The grinding circuit includes the following key pieces of equipment:

- SAG mill.
- Classification cyclones.
- Cyclone feed pumpbox and pumps.
- Milling drive-in and mill feed end sump pumps.

17.3.4 Pre-Leach Thickening

Trash screen undersize will gravitate directly to the pre-leach thickener via the pre-leach thickener feed box to increase the solids concentration of the CIL feed. Flocculant will be diluted in the pre-leach thickener flocculant static mixer and will be added to the pre-leach thickener to aid with particle settling.

Overflow solution from the 29m diameter pre-leach thickener will gravitate to the grinding water tank. Underflow slurry from the pre-leach thickener, at 45% w/w solids, will be pumped by dedicated thickener underflow duty / standby pumps to the pre-oxidation feed distribution box.

An automatic slurry sampler, installed on the feed to the pre-oxidation feed box distributor will collect a representative sample of the pre-leach thickener's underflow stream for plant control and metallurgical accounting purposes.

The pre-leach thickener area will be serviced by a dedicated pre-leach thickener area sump pump. Spillage and wash down collected by the sump pump will be returned to the pre-leach thickener distribution box. Excess water will overflow from this bunded area to the CIL bund.

The pre-leach circuit includes the following main pieces of equipment:

- Trash screen.
- Pre-leach thickener.
- Pre-leach thickener flocculant static mixer.
- Pre-oxidation feed automatic sampler system.
- Pre-leach thickener area sump pump.

17.3.5 Pre-Oxidation, CIL Circuit

The leach circuit consists of one pre-oxidation tank and seven carbon-in-leach (CIL) tanks in series. Provision has been incorporated in the layout for an 8th CIL tank. Pre-leach thickener underflow will be pumped to the pre-oxidation feed distribution box. The slurry from the feed distribution box will be discharged into the pre-oxidation tank. If the pre-oxygenation tank is offline, the slurry will be diverted to the first CIL tank via an internal dart plug distribution system. Barren and carbon cool solution and area spillage will be fed to first CIL tank via CIL feed distribution box with an option to the second CIL tank via an internal dart plug distribution system when necessary. Slurry can bypass the pre-oxidation tank or any of the CIL tanks when necessary. The ability to bypass any tank witin the train will be by two pneumatic gates located within the tanks inter-stage launders. One gate will divert slurry to the following CIL tank while the second gate will allow slurry diversion to the subsequent CIL tank.

The CIL tank's combined live volume provides a residence time of approximately 27.7 hours at a slurry feed rate of 874 m^3/h. The physical dimensions of the leach tanks at 15.8 m diameter by 18.4 m high are the same as the Oxide plant CIL circuit tanks. The CIL tanks will normally be operated with a carbon concentration of 10 g/L and can go up to 20 g/L.

Sodium cyanide as a 20% w/v solution will be added to the circuit from the cyanide dosing pumps. Cyanide solution can be added to the first four CIL tanks as required, but generally in the first two tanks with a control loop with the cyanide analyzer. The operating pH of the CIL circuit will be maintained above 10.5 to maintain the protective alkalinity of the circuit and prevent the loss of cyanide in the form of gaseous hydrogen cyanide. Protective alkalinity will be maintained by the addition of quicklime to the SAG mill feed conveyor.

Oxygen will be sparged through oxygen spargers into the slurry in the pre-oxidation tank and all subsequent CIL tanks in the circuit to oxidize cyanide consuming species and to improve the leach kinetics by maintaining a high dissolved oxygen level throughout the circuit.

Fresh carbon or regenerated carbon from the carbon regeneration circuit will be returned to the last tank of the CIL circuit with the alternative option of addition to the second last CIL tank and will be advanced counter-currently to the slurry flow by recessed impeller carbon advance pumps in each CIL tank. The intertank screens in each CIL tank will retain the carbon within each tank whilst allowing the slurry to flow by gravity to the downstream tanks. This counter-current process will be repeated until the gold loaded carbon reaches the first CIL tank. Recessed impeller pumps will be used to transfer slurry from the first tank to the loaded carbon screen mounted above the acid wash column in the elution circuit. Undersize from the loaded carbon screen will be gravitated to the CIL feed distribution box. A recessed impeller pump in CIL tank 2 is used to transfer slurry from the second tank to the loaded carbon screen during the time when first tank is offline.

Slurry from the final CIL tank will gravitate to the vibrating carbon safety screen to recover any carbon leaking from worn screens or overflowing tanks. Instrumentation for pH measurement and control will be included in the pre-oxidation tank, and CIL tank 1 when the pre-ox tank is offline, to ensure a high enough pH is maintained prior to cyanide addition.

The leach area will be serviced by four area sump pumps. The sump pumps will return spillage to the closest CIL tank. The CIL bund area will overflow to the Oxide plant event pond in the case of emergencies.

The pre-oxidation and CIL circuit includes the following main pieces of equipment:

- Pre-oxidation tank with agitator.
- Seven CIL tanks with agitators, interstage screens and recessed impeller transfer pumps.
- Gantry crane.

- Cyanide analyzer and HCN gas monitor.
- Intertank maintenance frame and CIL high pressure washer.
- Four CIL area sump pumps.

17.3.6 Desorption and Carbon Regeneration

A 12t pressure Zadra elution circuit was selected for gold recovery from the activated carbon.

The desorption circuit will have separate acid wash and elution columns allowing a strip to be completed within 18.5 hours. At a carbon gold loading of ~1,800 g/t and silver loading of ~350 g/t, the required peak daily carbon movement will be 12 t, requiring one strip per day.

Acid Wash

A cold acid wash sequence will be required to remove accumulated calcium scale on the carbon surface. This process improves elution efficiency and has the beneficial effect of reducing the risk of calcium magnesium slagging within carbon during the regeneration process. The acid wash column fill sequence will be initiated once the carbon recovery pump in CIL tank #1 starts pumping to the loaded carbon screen. Carbon will be gravity fed from the loaded carbon recovery screen directly into the acid wash column, with the underflow slurry from this screen gravitating back into the CIL feed distribution box. Once the acid wash column is filled to the required level, the carbon fill sequence will be stopped.

The acid wash cycle will utilize a 3% w/w hydrochloric acid solution. Hydrochloric acid (32% w/w) will be diluted to 3% w/w by injecting a measured amount of acid into filtered water as it fills the acid wash column, from the bottom up. The carbon will be allowed to soak in the dilute acid for a period of half an hour.

Upon completion of the acid soak, the acid rinse cycle will be initiated; loaded carbon will be rinsed with water to displace acid solution and contaminants. Four bed volumes (4 BV) of water, at 2 BV/h, will be pumped through the column. Displaced solution from both the acid rinse and wash steps will pass through the acid wash discharge strainer before discharging to the tailings pumpbox.

The acid wash sequence will conclude with carbon being transferred to the elution column. The acid wash column will be contained in the acid mixing bund to ensure safe spillage handling.

The acid wash circuit includes the following main equipment:

- Loaded carbon screen.
- Acid wash column.

Elution

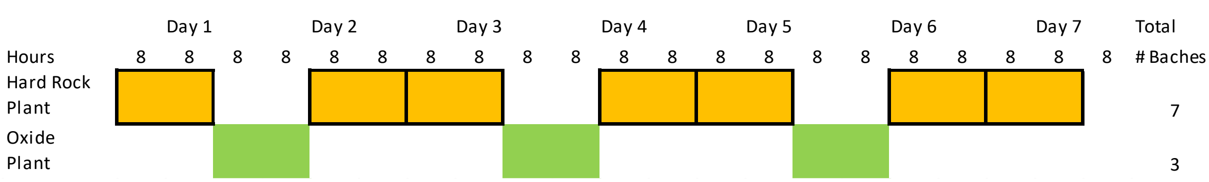
A 2% w/v caustic soda (NaOH) and 0.2% w/v sodium cyanide (NaCN) solution (barren eluate) from the strip solution tank will be pre-heated to 95°C using a diesel fired strip solution heater. During pre-heating solution will be circulated through the recovery heat exchanger, heat input heat exchanger, then returned to strip solution tank. Once pre-heated, the solution will be diverted through the heat recovery exchanger and fed to the elution column to commence stripping gold and silver from the loaded carbon. During this process the eluate will be further heated, under pressure, to 140°C.

Eluate will flow up through the carbon bed and out of the top of the column, passing through the recovery heat exchanger, trim cooling heat exchanger and then fed to the electrowinning cells for gold and silver recovery. Barren eluate exiting the electrowinning cells will return to the strip solution tank for re-use in that or subsequent elution cycles.

A total of 30 bed volumes of eluate strip solution will be cycled through the elution column and electrowinning cells. Upon completion, heating will be ceased, and cooling water will be injected into the circulating stream for a period of one hour. This cooling water will displace a portion of the total strip solution which will be removed as a bleed from the barren solution return circuit, discharging to the CIL feed distribution box. Upon completion of the cool down sequence, the carbon will be transferred to the carbon holding hopper in the Hard Rock plant (with the option to return the carbon directly to the CIL circuit) and will remain there until the Oxide plant carbon regeneration kiln is ready for taking a new batch of carbon for re-activation.

Currently, Oxide plant carbon regeneration kiln is used for only 30% of time. Both the Hard Rock and Oxide plant will be using the same carbon regeneration kiln at the sequence shown in Figure 17.3.1.

Figure 17.3.1 Hard Rock and Oxide Plant Carbon Regeneration Sequence



The stripping circuit includes the following main pieces of equipment:

- Elution column.
- Strip solution heater with heat input and heat recovery heat exchangers.
- Trim Cooling Heat Exchanger.
- Strip solution tank.

- Carbon Holding Hopper.
- Carbon Transfer Pump.
- Anti scalant dosing and descaling sulphamic acid systems
- Elution area sump pump.

Carbon Regeneration

Once the carbon regeneration kiln is available to take stripped carbon from Hard Rock plant, carbon will be fluidized (if required) and then pumped by the carbon transfer pump from the carbon holding hopper to the carbon dewatering screen, allowing excess water to be removed prior to the carbon discharging into the carbon regeneration kiln feed hopper. Dewatering screen undersize will be gravity fed to the carbon safety screen feed box in the Oxide plant.

Carbon will be withdrawn from the kiln feed hopper, by the kiln screw feeder, and fed directly to the carbon regeneration kiln, at a rate of 800 kg/h. The carbon will be heated within the diesel-fired, horizontal rotary kiln up to 750°C, to remove volatile organic foulants from the carbon surface and restore the carbon activity.

Re-activated carbon exiting the kiln will discharge directly to the carbon quench hopper in the Oxide plant, where it will be submerged in water and rapidly cooled. From the quench hopper, carbon will be pumped by a new carbon transfer pump (located in the Oxide plant) to the carbon sizing screen in the Hard Rock plant. Sizing screen oversize will be gravity fed to CIL tank #7 with a bypass option to CIL tank #6. Sizing screen undersize will be discarded to the carbon safety screen feed box. Fresh carbon will be added to the CIL circuit via the carbon quench hopper.

The carbon reactivation circuit includes the following main pieces of equipment:

- Existing carbon dewatering screen.
- Existing regeneration kiln including feed hopper and screw feeder.
- Existing carbon quench tank and new carbon transfer pump 2.
- New carbon sizing screen.

17.3.7 Electrowinning and Gold Room

Gold and silver recovery, from the pregnant eluate, will be achieved by electrowinning onto stainless steel cathodes. The electrowinning circuit will consist of three electrowinning cell / rectifier combinations located in the existing Oxide plant gold room. Two new cells will be installed near the existing three Oxide plant electrowinning cells. The third Oxide plant electrowinning cell will act as a common standby for both plants.

Once the elution pre-heating cycle has been completed, the electrowinning sequence will be initiated by diverting pregnant eluate solution to the electrowinning cells. The electrowinning cell discharge or barren eluate will be returned to the strip solution tank for re-use.

Upon completion of the electrowinning cycle, the cell covers will be removed, and gold and silver sludge will be washed off the cathodes and the bottom of the cell with a hand-held high pressure cathode washer. The gold and silver bearing sludge draining from the cell will be collected into a new sludge hopper located in the Oxide plant gold room and will then be filtered in two pressure filters, one from the existing Oxide plant and one new, and then dried in a new drying furnace located in the Oxide plant gold room.

Dried sludge will be mixed with a prescribed flux mixture (silica, sodium nitre, borax, and soda ash), prior to charging into the Oxide plant diesel-fired gold furnace to produce slag and doré ingots. The doré ingots will be cleaned, assayed, stamped, and stored in a secure vault ready for dispatch. The furnace slag produced will periodically be returned to the Oxide plant grinding circuit, via the ball mill feed box.

The gold room and electrowinning area will be serviced by a new gold trap and dedicated gold room area sump pump. Any spillage collected in this sump will be pumped back to the Hard Rock plant leach feed distribution box.

The electrowinning circuit and goldroom include the following key pieces of equipment serving the Hard Rock plant:

- Two new and electrowinning cells with dedicated rectifiers, plus one of the three existing electrowinning cells shared between the Hard rock and Oxide plants.
- New sludge hopper and sludge pressure filters (new and existing).
- New Drying oven.
- Existing flux mixer.
- Existing diesel smelting furnace with bullion moulds and slag handling system.
- Existing Bullion vault and safe.

- Existing Dust and fume collection systems.
- Existing Goldroom security system.
- New HCN gas monitor.
- New Goldroom sump pump with imbedded gold trap

17.3.8 Tailings Disposal

Slurry from the CIL circuit will gravity fed to the carbon safety screen. The carbon safety screen will recover any undersize carbon exiting the CIL circuit through the last inter-stage screen. The safety screen will also serve to prevent carbon loss to tailings in the event of a defective inter-stage screen.

The safety screen oversize will report to a fine carbon collection bag while the undersize will be gravity fed to the tailings pumpbox which will be pumped to the Oxide plant Tailings Storage Facility (TSF). Hard Rock plant and Oxide plant tailings will be directed to opposite ends of the TSF in a combined TSF manifold system.

An automatic two stage slurry sampler will be installed on the carbon safety screen feed which will collect a representative sample of the CIL tail stream. This sample will be assayed, and the result will be used for circuit monitoring and metallurgical accounting.

The main equipment in this area includes:

- Tailings automatic sampler system.
- Carbon safety screen.
- Tailings pumpbox and tailings pumps.
- HCN gas monitor.
- Tailings area sump pump.

17.3.9 Event Pond

The process plant is designed to operate with zero discharge of process solutions to the local environment. The CIL bund will mainly be used to contain spillage. In case of emergency, spillage will be diverted to the Oxide plant existing lined event pond designed to contain any unforeseen spillage event. A second event pond pump will be installed in the event pond. Solution accumulating in the event pond will be returned periodically either to the Oxide plant process water tank or Hard Rock plant tailings pumpbox.

17.3.10 Reagents Mixing and Storage

The major reagents utilized within the process plant are:

- Quicklime (90% w/w CaO content) for pH control.
- Sodium cyanide (NaCN) for gold dissolution and desorption.
- Caustic soda (NaOH) for desorption.
- Hydrochloric acid (HCl) for carbon acid washing.
- Flocculant for pre-leach thickener solid settling.

In addition, fluxes (silica, nitre, borax, and soda ash) will be required for smelting charge preparation. Antiscalant will also be used as required to reduce scaling in the process water distribution, carbon wash and stripping circuits. Sulphamic acid will be used to de-scale the elution heat exchangers as required.

Quicklime

Quicklime will be delivered to site in 30 t bulk tankers as used currently for the Oxide plant. Bulk tankers will be pneumatically off-loaded, using a blower, directly to two lime silos with a total capacity of 68 t (72 m^3 same as Oxide plant). Quicklime will be withdrawn from the silo by a variable speed rotary valve and deposited directly onto the SAG mill feed belt conveyor.

Sodium Cyanide (NaCN)

Sodium cyanide will be delivered by the full container load (20 t) as double-bagged briquettes in standard 1,000 kg plywood boxes. Cyanide will only be removed from the shipping containers after it arrives on site and the containers can be unloaded in a controlled environment. Site stocks of cyanide will be stored in the Oxide plant reagent shed.

Cyanide mixing will be done in the existing cyanide mixing tank located in the Oxide plant. The cyanide bulk bag will be lifted by the reagents area hoist from the plywood box to a bag breaker above the agitated cyanide mixing tank, which will have been previously partially filled with process water and a small amount of caustic solution to provide a high pH environment. After dissolution of the cyanide briquettes the mixing tank will be topped up with process water to achieve a 20% w/v cyanide concentration. Once mixing is completed cyanide will be transferred to the Hard Rock cyanide storage tank by a new cyanide transfer pump.

During operations cyanide will be drawn from the Hard Rock cyanide storage tank and dosed to the CIL circuit distribution points as required by new duty/standby Hard Rock cyanide recirculation pumps.

The sodium cyanide and caustic soda mixing, and storage areas will be provided with a common bunded area serviced by a common sump pump in the Oxide plant. Any spillage generated within this area will be pumped to the leach feed distribution box in the Oxide plant.

Sodium Hydroxide (NaOH)

Sodium hydroxide (caustic soda or caustic) mixing will be done in the existing Oxide plant. Caustic will be delivered to site as pearls in 1 t bulk bags. The bag will be lifted to the existing bag breaker mounted above the existing caustic feeder.

Caustic will be fed slowly by the caustic feeder to the existing caustic mixing tank which will have been previously filled with sufficient water to prevent localised heat generation during dissolution. Filtered water will be used to top up to the mixing tank to achieve a solution with the desired caustic concentration (20% w/v). The mixing tank will be mechanically agitated to assist with caustic dissolution. A new caustic dosing pump will be used to deliver caustic solution to the strip solution tank in the Hard Rock plant.

Hydrochloric Acid (HCl)

Hydrochloric acid (32% w/w) will be delivered to site in 1,000 L intermediate bulk containers (IBC). A new acid metering pump will supply acid, from the bulk containers to the acid wash column where in-line dilution with water will result in a 3% w/w solution feeding the acid wash vessel.

The new hydrochloric acid storage tank and sump area will be protected with an acid-resistant liner and any spillage will be transferred to the tailings pumpbox by the area's dedicated sump pump.

Antiscalant

Antiscalant will be delivered to the plant in intermediate bulk containers (IBC). Metering pumps will distribute antiscalant directly from the IBC to the required dosing points.

Fluxes

The following fluxes will be delivered to the plant in 25 kg bags and used in the gold room: Borax ($Na_2B_4O_7 \cdot 10H_2O$), Sodium Nitrate ($NaNO_3$), Sodium Carbonate (Na_2CO_3) and Silica (SiO_2).

Activated Carbon

Activated carbon will be delivered to the plant in 500 kg bulk bags for topping up carbon in the CIL circuit. An inventory of two CIL tank's worth of carbon will be kept on site for the Hard Rock and Oxide plant inventory.

Grinding Media

The Hard Rock plant SAG mill will require 125mm diameter grinding media. First fill size range will be 80mm to 105mm. The grinding media will be delivered to the plant in 850 kg drums.

17.3.11 Water Services

The Hard Rock plant will require process water for the grinding circuit, high pressure filtered water for dust suppression, filtered water for SAG mill seal water, acid wash, elution, CIL, carbon regeneration, reagents mixing and gland water for sealing slurry pumps.

Grinding and Process Water

Grinding water will be supplied from the pre-leach thickener overflow. Grinding water will be stored in a grinding water tank, which will provide more than 90 seconds of surge capacity. The total nominal grinding water consumption is estimated to be 671 m^3/h.

Process water be not released to the environment. Process water will be continuously recovered from plant tailings and re-used to slurry fresh feed to the milling circuit. However, during normal operations process water will be lost by:

- Entrainment in the tailings in the TSF.
- Evaporation from the TSF decant pond.

Recovered process water will predominantly consist of water reclaimed from the tailings storage facility. Process water will be recovered to a 1,500 m^3 process water tank, which will provide approximately 1.1 hours of surge capacity. The total nominal reticulation of process water is estimated to be 1,344 m^3/h which includes 671 m^3/h grinding water. Process water make-up will be decant water from the TSF and raw water reservoir No. 2.

Grinding and process water combine in a single suction manifold and will be distributed via headers by duty and standby configuration of single stage water pumps with offtakes supplied for the following predominant users:

- Milling area (mill feed chute, cyclone feed dilution and pebble dewatering screen spray water).
- Pre-leach area (trash screen spray, flocculant static mixer).
- CIL circuit (loaded carbon screen and sizing screen spray).
- Carbon safety screen spray.

Raw Water

Raw water will be pumped from the Oxide plant Off Channel Reservoir (OCR) during the wet season (four months) and stored in the Raw water Reservoir No. 2 until needed. Three new pontoon mounted submersible pumps will be installed in the OCR. During the wet season two of the three pumps will be in operation. For the remainder of the year, one OCR pump will be operated if required. In addition, three new pontoon mounted submersible pumps will be installed in the raw water reservoir No. 2 to deliver water to the plant's process water tank (as a make-up if required). These pumps will provide the necessary capacity and flexibility to meet the varying water demand between wet and dry seasons. During the wet season one pump will be in operation and for the remainder of the year, two of the three pumps will be operated as required.

Low Pressure Filtered Water

Raw water from the Oxide plant raw water tank will overflow to the Hard Rock plant filtered water treatment plant #2 (located in the Oxide plant) and treated water will be stored in a 400 m^3 LP Filtered water tank.

Low pressure filtered water from LP filtered water tank will be distributed by duty/standby filtered water pumps to the following main users:

- Gland water pumps.
- SAG mill seal water.
- CIL area (loaded carbon transfer, CIL high pressure washer, cyanide analyzer).
- Acid wash and elution (transfer water, strip solution make-up, trim heat exchanger cooling)
- Carbon transfer from the Hard Rock plant to the Oxide plant carbon regeneration kiln.
- Flocculant and HCL acid mixing.
- High pressure filtered water tank.

High Pressure Filtered Water

Filtered water from HP filtered water tank will be distributed to the following main user by HP filtered water pump:

- Hard Rock primary crushing dust suppression.
- Coarse ore storage dust suppression.

Gland Seal Water

Gland seal water will be supplied from the LP filtered water tank. Gland seal water will be required for the cyclone feed pumps, pre-leach thickener underflow pumps and tailings pumps.

Potable Water

Potable water will be sourced from the Oxide plant systems and will be distributed to the Hard Rock plant for safety showers.

Fire Water

Firewater will be supplied from the Oxide plant fire water system. A take off and return line connecting the Oxide plant fire water header to the Hard Rock plant fire water header will be provided. A fire water header will run through the Hard Rock plant and will supply water to the hose reels and fire hydrants.

17.3.12 Air and Oxygen Services

Plant air will be provided by the plant air system at a delivery pressure of 750 kPag. A new plant air compressor and plant air dryer will be installed to supply plant and instrument air to the Hard Rock plant. Existing Oxide plant standby plant air compressors and dryers will be reconfigured and will be used as common standby units for both Hard Rock and Oxide plants.

Oxygen for the spargers of the pre-oxygenation tank and CIL tanks will be supplied from a PSA or similar type package oxygen system at a delivery pressure of 700 kPag.

17.4 Water, Power, and Reagent Consumption

17.4.1 Water Consumption

To support the processing of 10.3 Mtpa, water will be supplied to the Oxide and Hard Rock plants from the existing 5.2 million cubic metre OCR pond and a new 2.2 million cubic metre water Reservoir #2 pond. Raw water to the Hard Rock plant will be supplied from the raw water Reservoir No. 2 pumps. Average raw water consumption for Hard Rock plant is estimated to be 300 m^3/h which includes 225 m^3/h process water make-up, 73 m^3/h as filtered water and 2.1 m^3/h as potable water. Approximately 225 m^3/h will be supplied from the raw water reservoir No. 2 and the remainder will be supplied from the Oxide plant.

Water from the pre-leach thickener overflow stream will be recycled within the process plant to reduce external water requirements.

An average 437 m^3/h of decant return water will be recycled from the TSF to the hard rock plant as process water make-up. A maximum 860 m^3/h decant return will be required during times when no raw water make-up is available to the process water tank. This quantity will likely only be available during the wet season when rainfall on the TSF, and site run-off stored on the TSF, increases the size of the decant pond. The shortfall will be made up by pumping raw water from the raw water Reservoir No. 2.

At the end of the wet season, when the Nobsin River is no longer topping up the OCR level, decant water from the TSF will continue to be used in preference to raw water make-up to conserve water in the OCR for use later in the dry season.

17.4.2 Reagent and Consumable Consumption

Table 17.4.1 provide a summary of the major reagents and consumables usage on an annual basis.

Table 17.4.1 Major Reagents and Consumables

Reagent / Consumable	Hard Rock Annual Consumption
SAG Mill Grinding Media	4,664 t
Quicklime (90% w/w CaO)	4,302 t
Sodium Cyanide	3,085 t
Activated Carbon	132 t
Sodium Hydroxide (Caustic)	367 t
Hydrochloric Acid	524 t
Flocculant	35 t
Diesel (elution heater, carbon kiln, smelter)	1,750 m^3

17.5 Plant Control System

17.5.1 General Overview

A single control system will service both the Hard Rock and Oxide plants. The general control philosophy for the plant will be one with a moderate level of automation and central control facilities to allow critical process functions to be carried out with minimal operator intervention similar to that successfully installed and utilized in the Oxide plant. Instrumentation will be provided within the plant to measure and control key process parameters.

The main control room will house two PC-based operator interface terminals (OIT) and a single server. These workstations will act as the control system supervisory control and data acquisition (SCADA) terminals. 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 process control system (PCS).

The process control system that will be adopted for the plant will be a programmable logic controller (PLC) and SCADA based system as per that existing in the Oxide plant. The PCS will control the process interlocks and control loops for non-packaged equipment. Control loop set-point changes for non-packaged equipment will be made at the OIT.

17.6 Sampling and Assaying

Oxide plant titration facilities and an on-line analyser unit will be provided to monitor cyanide concentration in CIL process liquors. Automatic samplers will take composite samples during each shift from the pre-oxidation feed and CIL tailings streams which will provide the primary gold balance for the Hard Rock plant. These samples will be filtered and dried on site with splits then sent for assay.

Manual sampling of slurry and carbon in the CIL circuit will be used to monitor the CIL profile and provide end of month gold in inventory measurement for metallurgical accounting. Manual sampling of loaded and barren carbon and the pregnant and barren eluate streams will monitor the performance of the elution and electrowinning circuits respectively.

The oxide plant metallurgical laboratory will continue to be used to undertake simple bottle roll leach testing and other testwork. This will be used to monitor the metallurgical properties of pre-production mining samples to ensure that plant performance can be predicted in advance.

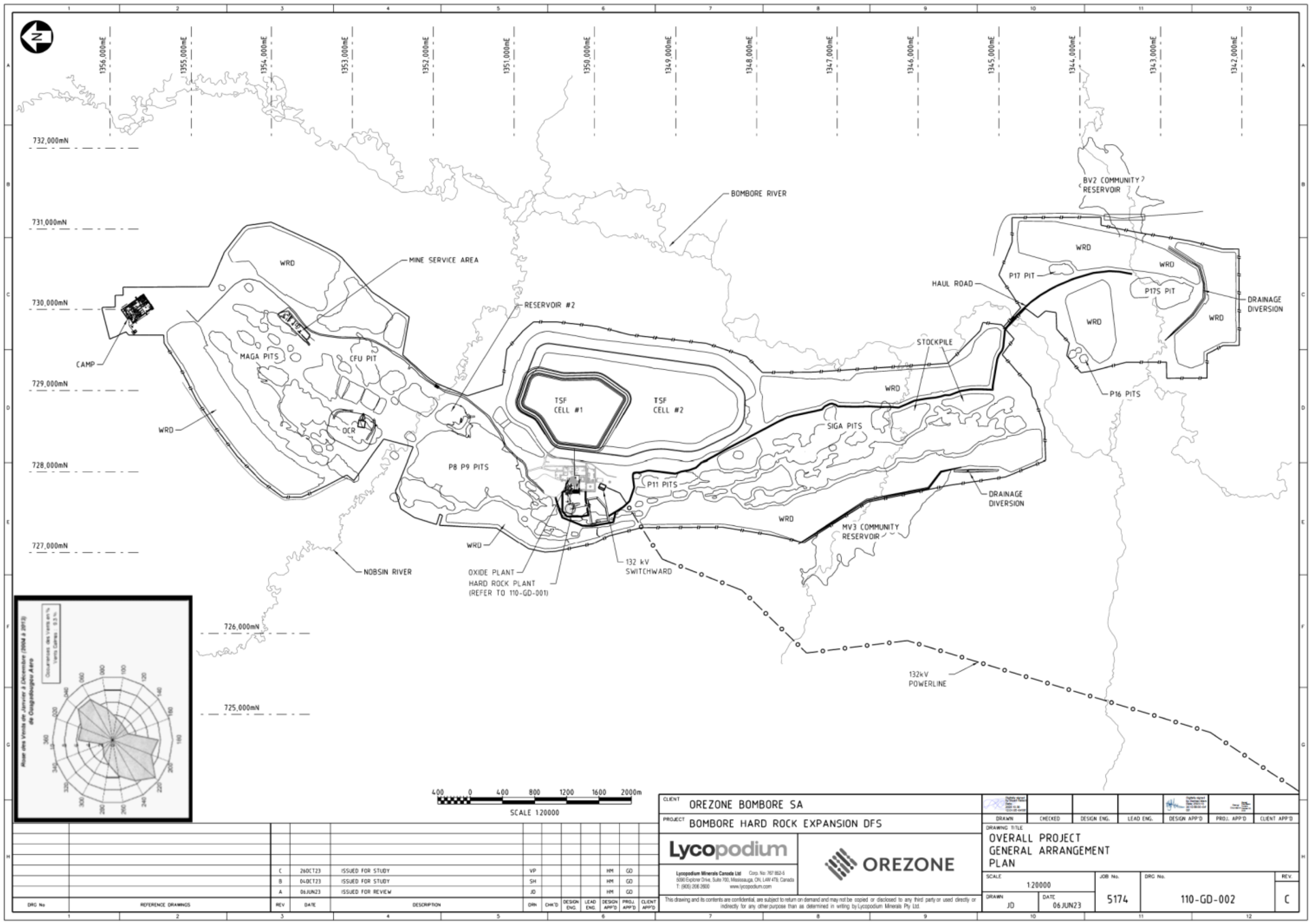
The oxide plant laboratory facilities will also continue to be used for assaying approximately 10,000 mining grade control drilling samples per month. Fire assays are supplemented by LeachWell type rapid cyanide soluble gold assays.

18.0 PROJECT INFRASTRUCTURE

18.1 Project Infrastructure

The overall site plan, shown in Figures 18.1.1, specifies the major facilities for the project and includes the ultimate mine open pits, process plant, TSF expansion, additional raw water storage, waste dumps, 11 kV power distribution and plant access road. Power is provided from a new onsite 132 kV switch yard and transmission line. The site operation is fenced to clearly delineate the mine area and deter access by unauthorized persons and prevent grazing animal access.

Figure 18.1.1 Overall Site Plan



18.2 Site Access

Site access is from the north using an upgraded 5 km public road that connects the site to national highway N4 and enters the north end of the property where the Bomboré camp is located. Private roads provide light and heavy vehicle access around the site facilities. An existing heavy vehicle bridge provides access across the Nobsin River, and a new heavy vehicle bridge will be installed across the Bomboré river for access to the P16/P17 mining areas.

Monitored high-security double fencing is constructed around the process plant and associated infrastructure. Access into the fenced area is through a manned gatehouse with separate personnel and vehicle access.

A helipad is located within the plantsite for gold shipments.

18.3 Accommodation

The existing camp at the northern end of the Project area includes accommodation, dining and recreation facilities, power generation, water treatment, and office facilities. The camp is fully enclosed within a double fence, which is secured and patrolled by armed security personnel. The Project site is within a thirty-minute drive from the regional town of Mogtédo, with a population in excess of 15,000. The town is developing rapidly with many substantial multi story concrete block buildings established or under construction.

Most of the semi-skilled and unskilled labour required for project development and operations are sourced from Mogtédo and surrounding communities. Mogtédo has the capacity to provide rented rooms and leased accommodation for the contractor's skilled workforce, the contractors as per the Oxide Plant construction and operations will make their own accommodation arrangements with local businesses. Contractors arrange for bussing their employees to and from site and for providing a midday meal.

18.4 Mine Service Area

The Mine Services Area (MSA) facilities have been established by the mining contractor and the Company, and includes workshop and warehouse facilities, offices, dining room, and ablutions. Power is provided to the MSA from the site 11kV overhead powerline.

18.5 Site Buildings

OBSA plant site buildings are located east of the processing facilities. The plant site buildings include:

- Process office with control room.

- Assay and Metallurgical Laboratory.
- Gatehouse and Security Building.
- Mess.
- Ablutions.
- Reagent Storage Buildings.
- Plant Maintenance Workshop.
- Warehouse.

18.6 Power Supply

Site power is currently supplied by rental diesel generators. However, connection to the national grid is underway, with forecast completion in Q4 2023. The grid connection infrastructure includes a 132kV switching station tying into an existing 132kV overhead line, a 19km 132kV overhead powerline, and a site 132kV/11kV switchyard. After connection to the grid, diesel generators will be used only for critical load back-up power.

The average site power demand with the stand-alone Oxide plant in operation is estimated to be 6.6 MW. The average additional power demand for the stand-alone Hard Rock plant in operation is estimated to be 19.1 MW with connected power at 31.3 MW.

The new 132kV/11kV switchyard will provide energy to a 11kV switchgear located within the new Hard Rock plant area containing feeders for each of the various distribution transformers.

Transformers will be installed near the areas of use and will step-down the voltages from 11 kV to 415 V. The existing 11 kV overhead power line will be expanded to distribute power across the site, stepped down to 415 V at point of use.

18.7 Potable Water

Two vendor-packaged modular potable water treatment plants including filtration, ultra-violet sterilization and chlorination are installed at the accommodation camp and process plant for reticulation to the camp, site buildings, ablutions, safety showers and other potable water outlets.

18.8 Sewage & Waste Management

Grey water and effluent from the accommodation camp drains to a recently upgraded vendor package sewage treatment plant located adjacent to the camp for treatment.

Treated effluent is discharged into leach drains. Treatment plant sludge is suitable for direct landfill burial in unlined pits.

The process plant, mine services area and other remote facilities use septic tanks for collection of sewage. These are emptied as required and the contents transported to the camp sewage treatment plant for treatment.

Site solid waste is currently sorted into bins and removed by a contractor for recycle or disposal at the municipal facilities in Ouagadougou.

Waste will continue to be sorted and reused or recycled as far as the limited access to recycling facilities in Burkina Faso allows.

Inert solid wastes are deposited into suitable landfill sites at the toe of the waste dumps and promptly covered to deter unauthorized access and re-use. Materials such as cyanide packaging is cleaned and buried, under supervision, on site beneath mine waste to prevent unauthorized use of the packaging. On site incineration is not permitted.

Putrescible waste from the kitchen and general site refuse bins are transported to the municipal facilities for disposal.

Waste lubricating oils are returned to the supplier for recycling.

18.9 Communications

The Internal communications and IT services will be expanded and integrated via an existing site-wide fibre optic network. External communication is via the national telecommunication network, with satellite communication as emergency back-up. Additionally, a radio network is established on site.

18.10 Fuel & Lubricant Supply

The existing on site fuel and lubricant storage and dispensing facilities are administered by the selected fuel distributor. Fuel storage facilities have been installed in the Mine Services Area, camp, and power plant.

18.11 Site Security

Orezone has adopted a multi-layered security framework. Site security is based on concentric lines of fencing / access control.

The current operating area is enclosed within a patrolled chain link fence line. The main point of entry is via a manned gatehouse immediately north of the camp.

The process plant is enclosed by a double chain link security fence with defensive razor wire on top and on ground level and monitored by closed circuit cameras with artificial intelligence and integrated face recognition. Entry is by a single monitored security post with controlled access (biometric access control) and departure.

Access to the goldroom within the plant is restricted and strictly controlled employing a two-factor identification, metal detection and automatic door closures with interlinked locking. Video surveillance is installed, and entry points are monitored and alarmed. A minimum of two authorized staff must be in the goldroom while work is in progress with constant monitoring of the surveillance systems from a central control room.

The accommodation camp is fenced with a manned entry gate to prevent unauthorized access. Security personnel contracted to Orezone are supplemented by law enforcement, military, and police patrols.

18.12 Ouagadougou Facilities

In Ouagadougou, Orezone owns and operates a fully functional office and warehouse facility with auxiliary power, water, and redundant internet connectivity.

The Ouagadougou facility is sufficient to serve as a management and logistics base for the Bomboré operation. Administration functions and services such as procurement, accounting and government relations are based out of the Ouagadougou office reducing the burden on site facilities and accommodation.

18.13 Tailings Storage Facility

The Tailings Storage Facility (TSF) was constructed east of the Plant Site to the Stage 2 design elevation as a single cell paddock type storage facility. The facility will be expanded to form a two-cell arrangement and subsequently combined to form a single cell after the initial years of operation. The TSF Cell 2 area has been expanded eastwards and therefore occupies a larger area than the current TSF design.

The TSF north cell has been constructed to Stage 2 and will be developed to Stage 3 in 2023/2024 to store tailings from the initial three years of operations. The second (south) cell will be developed from Stage 4 (Year 4) to Stage 7 (Year 7). After this, the two cells will be combined to store tailings from Year 8 onwards. The objective of splitting the facility initially was to reduce the up-front capital by reducing the area of the initial basin liner. The overall facility has been designed to store 128 Mt of tailings.

The embankments will be developed downstream with multiple zones, constructed using selected open pit mine waste or local borrow. To allow flexibility to adjust the design to suit ongoing mine planning, the facility will be constructed in stages. The initial Stage 1 storage capacity was designed for 5.2 Mt of tailings (12 months production) and the final storage capacity will be approximately 128 Mt of tailings.

The TSF has a composite liner, comprising compacted soil liner throughout the basin area and upstream face, with a continuous 1.5 mm HDPE liner overlying the compacted soil liner. The liner is 1.5 mm textured HDPE geomembrane liner on the upstream face of the perimeter embankment and 1.5 mm smooth HDPE liner within the basin area.

The design incorporates an underdrainage system within Cell 2. The underdrainage system comprises of basin finger drains and embankment upstream toe drains, in order to reduce seepage losses, improve tailings settled density and increase water recycle to the Process Plant.

Tailings will be discharged as a slurry into the TSF by sub-aerial deposition methods, using a combination of spigots at regularly spaced intervals to maintain the supernatant pond at the decant location. Deposition will be regularly moved around the facility to allow the deposited layer to dry and consolidate to improve settled density.

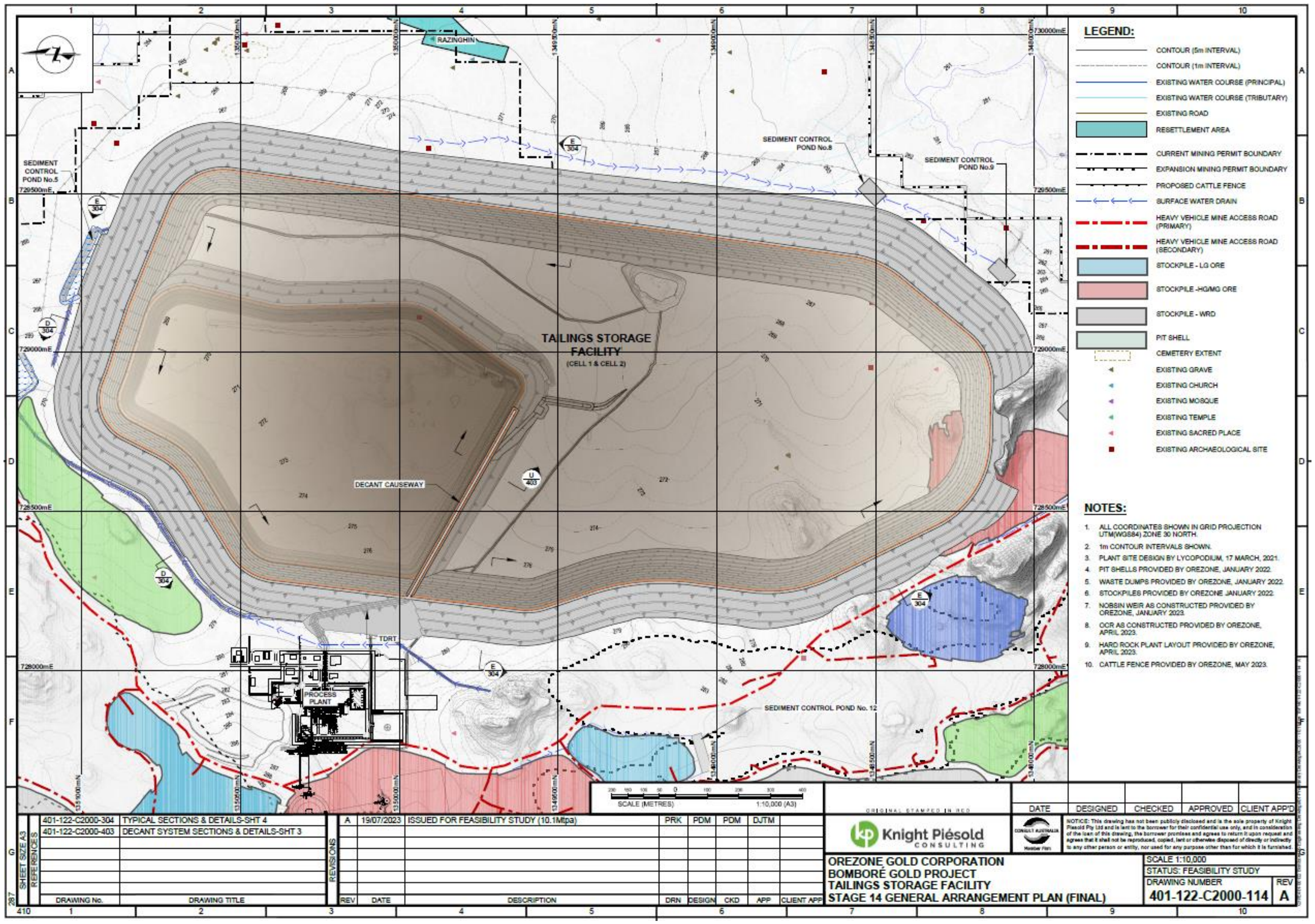
A decant turret system is located at the southeast corner of Cell 1 and the decant will remain at this location until the end of Stage 3. With expansion of the facility to Cell 2 the decant will be positioned in the centre of the facility on the south side of the Cell 1 south embankment. The Cell 1 south embankment alignment will be used as the alignment of the life of mine decant access causeway. The decant system will comprise of floating turrets operating in the supernatant pond and suction pumps located on pump platforms or on the embankment / causeway. The suction pump will be progressively moved to ensure it is positioned above the rising water and tailings levels.

Decant pumps will operate automatically, reclaiming water from the TSF and pumping it via a HDPE pipeline to the process plant. The use of reclaim water will be prioritised over other water sources in order to maximize the conservation of water.

An emergency spillway will be constructed with each embankment raise. The closure spillway will be located to discharge south of the TSF.

The TSF has been designed in accordance with Australian National Committee on Large Dams (ANCOLD), “Guidelines on Tailings Dams – Planning, Design, Construction, Operation and Closure”, July 2019 and the “Global Industry Standard on Tailings Management” (GISTM), August 2020.

Figure 18.13.1 Tailings Storage Facility Layout (Stage 14 – Final)



A summary of the TSF stages is provided in Table 18.13.1.

Table 18.13.1 Tailings Storage Facility Stage Details

Stage #	Cell	Stage Storage Capacity Months	Tailings Storage Cumulative Mt	Embankment Crest Elev. m RL	Maximum Embankment Height m
1	1	12	5.2	283.3	14.7
2	1	12	10.4	289.3	20.8
3	1	12	16.1	295.1	26.6
4	2	9	24.4	279.6	13.6
5	2	12	34.5	285.2	19.2
6	2	12	44.6	290.4	24.4
7	2	12	54.7	295.1	29.1
8	1&2	12	64.8	298.1	32.1
9	1&2	12	74.9	301.1	35.1
10	1&2	12	85.0	304.0	38.0
11	1&2	12	95.1	306.9	40.9
12	1&2	12	105.2	309.7	43.7
13	1&2	12	115.3	312.5	46.5
14	1&2	~15	128.0	316.0	50.0

18.14 Site Water Management

18.14.1 Site Water Balance

The Project is located in an area that has a Köppen climate classification of warm semi-arid. The site has a distinct wet season between May and October with little rainfall occurring outside of this period. The average rainfall for the project area is approximately 800 mm/yr. The annual average lake evaporation is 2,029 mm/yr, more than twice the annual precipitation. The minimum and maximum annual rainfall depths contained in the 54-year historical record are approximately 560 mm and 1,240 mm, respectively.

The operational water management strategy is to utilize water captured within the mine limits to the maximum practicable extent in an efficient manner. This strategy includes significant water storage, recirculation, and reuse efforts. The following summarizes the water management strategy:

- Raw (i.e.-fresh) water from the Nobsin River will be harvested each wet season and stored in two water reservoirs for year-round use. The reservoirs will serve as the main raw water supply source for the Project.

- The water volumes that accumulate in the TSF supernatant pond from direct precipitation and supernatant water liberated from the tailings will be reclaimed via the decant system and will be a primary source of process water to the process plant.

- When the reclaim water volumes from the TSF supernatant pond are insufficient to meet the process plant water demand, additional water will be sourced as needed from water reservoirs.

- Additionally, storm water collected in pits and from run-off from the waste rock dumps (WRDs) and low-grade stockpiles (LGS) will be collected in diversion channels and collection ponds. This water captured in the collection ponds will be pumped to the water reservoirs or plant site (dependent on water quality) for use in the process.

Localized, field-fit temporary sediment control structures are recommended for the areas that are not captured by the infrastructure designed as part of this Technical Report. Specifically, temporary sediment control structures should be constructed prior to full reclamation of environmental barriers and prior to commencing excavation of open pits where stormwater run-off could flow outside of the mine boundary. Typically, temporary sediment control structures include silt fences, hay bales, temporary sediment ponds, leaky rockfill berms, etc. These temporary structures are included instead of permanent collection ponds because (1) the pits will be excavated quite rapidly and thus, the rainfall run-off will be quickly contained within the excavation footprint and (2) the majority of the year is extremely dry, and run-off will generally be negligible during the long dry season. The temporary sediment control structures will be more important during the shorter wet season.

A water balance model was developed to understand the site water flows, estimate water demand related to tailings and to determine the design embankment crest levels to maintain containment. The water balance was completed for various rainfall conditions for selected operational years. A climatological analysis was performed to support the water balance. The following rainfall sequences were modelled:

- Average rainfall conditions.

- 10 year Average Recurrence Interval (ARI), wet season (three months) sequence with no evaporation during the wet season.

- 100 year ARI, dry year sequence.

The model was run for the operational period of the TSF, August 2022 through to 2037. The water balance evaluated the OCR, process plant, TSF and new water reservoir requirements. The following summarizes the key findings of the water balance analyses:

- Under all climatic conditions assessed, the TSF will operate with a water negative condition throughout the life of mine therefore an external water supply is required under all scenarios.

- Under wet conditions, rainfall volumes will not control the required embankment level. This is because the spillway invert set by the tailings deposition level is already higher than what would be required to prevent spillway flow for storms up to the design events assessed.

- Under normal operating conditions the supernatant pond will be maintained at a minimum pond volume by recycling water to the Process Plant.

- The TSF will have sufficient capacity to satisfy the water storage design criteria. This is illustrated on figures 18.14.1 and 18.14.2 for TSF Stages 1 through 3 and Stages 4 through 14, respectively, which show the simulated pond elevations from the water balance.

- The as built capacity of the OCR is estimated to be 5.2 Mm^3. The OCR water storage volume will be filled during each wet season over an 8-9 week period based on a design intake flow rate of $1m^3/sec$.

- An additional Raw Water Reservoir that is 42% the size of the existing OCR is required to prevent shortfalls for a throughput of 10.1 Mtpa.

- There is a relatively low chance that the combined total reservoir water volume would be exhausted by the end of the dry season based on the design criteria assessed. However, calibration of the modelling should be undertaken from the date of commissioning to refine the modelling.

- The OCR and new Raw Water Reservoir water volumes will be affected by the evaporation and seepage losses. Seepage losses were estimated based on the preliminary seepage analysis performed as part of the 2020 FS. Variations or modifications to the seepage loss parameters will affect the required water demand. Therefore, in order to calibrate the water balance model with actual site conditions during operations, it is recommended that a monitoring program be implemented. This will allow better accuracy in estimating water losses and therefore estimation of water requirements over the life of mine. This is already being completed during current Oxide plant operations.

Figure 18.14.1 TSF Water Balance Results, Stages 1 through 3

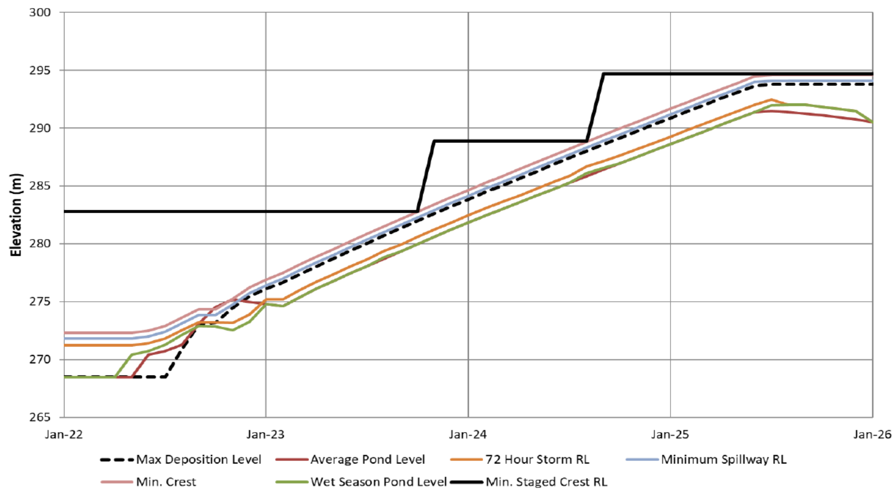
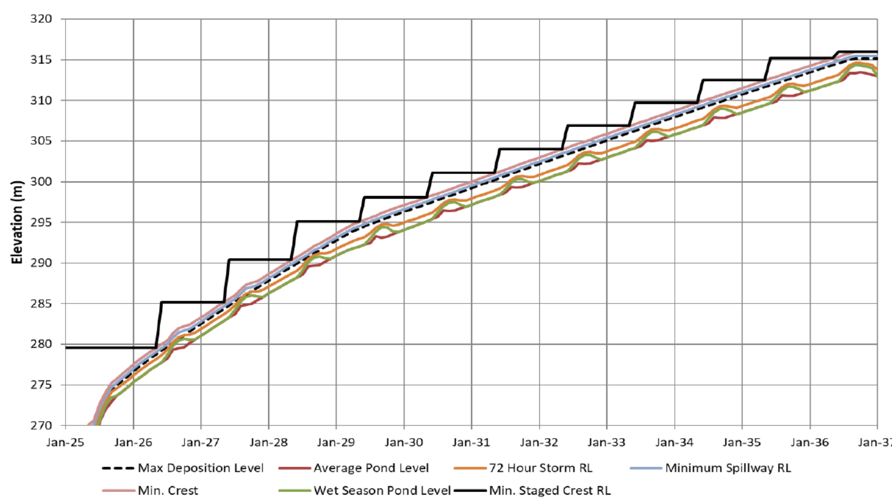


Figure 18.14.2 TSF Water Balance Results, Stages 4 through 14



18.14.2 Off-Channel Reservoir

The OCR is located north of, and adjacent to, the Nobsin River. The as built capacity of the OCR is estimated to be 5.2 Mm^3. The maximum elevation of water storage within the facility is approximately 263 m above sea level (masl).

18.14.3 New Raw Water Reservoir

A 1.8 Mm^3 Raw Water Reservoir No. 2 will be constructed 1km north of the plant site. Raw Water Reservoir No. 2 will be filled by pumping from the OCR during the wet season and will provide raw water to the Hard Rock plant. Exhausted pits will also be used to supplement storage capacity and achieve the 2.2 Mm3 incremental capacity requirement estimated by the water balance to support a 10.1 Mtpa processing rate.

18.14.4 Water Balance Calibration

To calibrate the water balance model with actual site conditions during operations it is recommended that a monitoring and calibration program be implemented which will allow better accuracy in the estimation of water requirements over the life of mine.

The modelling is sensitive to seepage assumptions, and should seepage losses be in excess of the current estimates this may result in the water reservoirs running dry earlier than estimated. It is recommended that a hydrogeological assessment be completed. This should incorporate the mining plan for pit development and pit dewatering as these will likely impact the OCR seepage rates.

18.14.5 Existing Nobsin River Offtake Infrastructure to OCR and Flood Protection Levees

The Nobsin River offtake system to transfer water to the OCR consists of a weir within the Nobsin River to form a backwater and direct river flows into the offtake channel to the OCR. Flow-control gates at the intake sunp location of the channel allow for management of the intake system. A flood protection levee was constructed to prevent the ingress / encroachment of Nobsin River flows to the OCR and associated infrastructure. Raw Water Reservoir No. 2 will be filled by pumping from the OCR during the wet season.

The capacity of the OCR offtake system is designed for a flow rate of $1m^3$/sec to the OCR. This allows for the capture of 5.2 Mm^3 in approximately 9 weeks (subject to rainfall runoff). The offtake channel invert elevation at the location of the flow-control gate is approximately 265 masl.

The OCR flood protection levees were designed with sufficient heights, extent, and erosion protection measures to prevent river flooding of the OCR and associated infrastructure for the 100-year flood. The levees are constructed of compacted fill and will be protected with riprap.

18.14.6 Water Diversions

Seasonal drainages through the Siga and P17 mining areas will be diverted around the pits into the Bomboré river. Community reservoirs will be constructed upstream of the diversions to provide a surface water supply beyond the wet season.

18.14.7 Oxide Plant Throughput Rate Increase

Subsequent to the water balance calculations, it was determined that the oxide plant throughput rate can be increased from 5.7 Mtpa to to 5.9 Mtpa. The increased rate will have a minor effect on the TSF stage designs and water requirements and will be addressed during the detailed engineering phase.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Product Sales

No formal market studies have been undertaken as refined gold is already being sold by the Company to global buyers at prevailing market prices. The final product of the mine is gold doré bars which Orezone has contracted with a European-based refinery to process into refined gold for final sale.

Commercial terms offered by the chosen refinery are market competitive based on a comparison of the tender proposals received. There is no indication of the presence of penalty elements that may impact the refining costs or render the doré untreatable. Custody of the doré is transferred to the refinery 'at the goldroom door' who assume responsibility for in-country transport and export.

19.2 Commodity Price Projections

The gold prices considered for developing the Mineral Resource Estimates and Mineral Reserves are $1,700 and $1,500 per ounce, respectively. For purposes of the project's economic analysis, a base case gold price of $1,750 per ounce has been selected starting in 2024, and a price of $1,900 per ounce in 2023.

19.3 Major Operations Contracts

Major contracts currently in place include:

- Contractor mining.
- Fuel supply.
- Major Reagents supply including carbon, grinding media, cyanide, and lime.
- Camp management & catering.
- Site security.

Contract terms are within industry norms.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

The approach developed by Orezone throughout the various environmental and social studies that have been conducted since 2009, especially in the context of the Environmental and Social Impact Assessment (ESIA), emphasized stakeholder concerns and integrated the environmental and social aspects into the initial stages of the Bomboré Mine design and continues into the Phase II Expansion design. This approach has ensured the integration of environmental and social issues in the design for the Bomboré Mine.

Orezone is committed to open and responsive engagement with local stakeholders. Orezone holds monthly meetings with planning committees comprised of local leaders and government officials and makes regular visits to nearby communities to disseminate information on Bomboré activities and to address any concerns. Orezone has instituted a grievance mechanism whereby residents can lodge any Bomboré-related issues to enable Orezone to respond in a rapid and in fair manner.

As a result, Orezone believes that community support for the Bomboré Mine remains strong.

20.2 Regulatory and International Standards Requirements

Burkina Faso has a regulatory framework for environmental and social management. The relevant policies, laws and regulations of Burkina Faso were all considered during the implementation of the ESIA.

20.2.1 Policies and Strategies for Environmental Protection

Since the early 1990s, Burkina Faso has developed numerous policies and strategies for the protection of the environment and management of its natural resources. A declaration of Mining Policy formulated in 1995 highlighted the importance of the private sector as an engine of economic development in the country. In May 2013, a Mining Sectoral policy was also adopted covering the period 2014-2025. Other policies on environmental protection include:

- National Plan of Social and Economic Development - Plan National De Développement Économique et Social (PNDES).
- National Policy on the Exploitation of Mining Resources - Politique Nationale en matière d'Exploitation des Ressources Minières (PNERM).
- Mining Sector Policy - Politique sectorielle des mines (POSEM) 2015-2022.
- Rural Development Strategy - Stratégie de Développement Rural (SDR) 2015.

- National Policy on Environmental Matters - Politique Nationale en matière d'Environnement (PNE).
- National Sustainable Development Policy - Politique Nationale de Développement Durable (PNDD).
- Environmental Plan for Sustainable Development Program - Plan d'Environnement de Développement Durable (PEDD).
- National Policy on Rural Land - Politique Nationale de Sécurisation Foncière en Milieu Rural (PNSFMR).
- National Action Program for Adaptation to Climate Variability and Change - Programme d'Action Nationale d'Adaptation à la variabilité et aux changements climatiques (PANA).
- National Gender Policy - Politique Nationale Genre (PNG).

20.2.2 Legal Framework

The legal framework with respect to environmental and social aspects related to economic activities is supported by many laws and decrees, including:

- Environmental Code.
- Mining Code.
- Forest Code.
- Public Health Code.
- General Local Authorities Code.
- Act on Rural Land Tenure.
- Act on Agrarian and Land Reorganization.
- Law on Water Management.
- Act on Pastoralism.

Other relevant regulations include:

- Decree No. 2015-1187/PRES-TRANS/PM/MERH/MATD/MME/MS/MARHASA/MRA/MHU/ MIDT/MCT dated 22 October 2015 on conditions and procedures relevant to the realisation and validation of the strategic environmental assessment and the environmental and social impact notice.

- Decree No. 2015-1200 /PRES-TRANS/PM/MERH/MME/MICA/MS/MIDT/MCT dated 28 October 2017 on the terms and conditions of environmental audit.

- Decree No. 2015-1205/PRES-TRANS/PM/MEF/MARHASA/MS/MRA/MICA/MME/MIDT/MATD dated 28 October 2015 on setting standards for discharges of used waters.

- Decree No. 2007-853/PRES/PM/MCE/MECV/MATD dated 26 December 2007 on specific environmental regulations for the exercise of mining in Burkina Faso.

- Decree No. 2006-590/PRES/PM/MAHRH/MECV/MRA dated 6 December 2006 on the protection of aquatic ecosystems.

- Decree No. 2006-588/PRES/PM/MAHRH/MECV/MPAD/MFB/MS dated 6 December 2006 determining the perimeters of protection for water bodies and streams.

- Decree No. 2001-342/PRES/PM/MEE dated 17 July 2001 on the scope, content and procedure for Environmental Impact Assessment Study and Environmental Impact Instruction.

- Decree No. 2001-185/PRES/PM/MEE dated 7 May 7 2001 on setting standards for discharges of pollutants into the air, water, and soil.

- Decree 2017-0068/PRES/PM/MEMC/MEEVCC/MINEFID/MATDSI dated 15 February 2017 relating to the rehabilitation fund and closure of mines.

- Decree 2017-0047 dated 1 February 2017 on the rehabilitation fund, artisanal mining sites and the fight against the use of prohibited chemicals.

- Decree 2017-0035/PRES/PM/MEMC/MINEFID/MCIA/MATDSI/MJFIP/MFPTPS/MEECVV dated 26 January 2017 pertaining to the mining agreement template (‘Decree 2017-035').

- Decree 2017-0036/PRES/PM/MEMC/MATDSI/MINEFID/MEEVCC/MCIA dated 26 January 2017 on mining permits and authorisations.

- Decree 2017-034 dated 26 January 2017 on the fund for the financing of geological and mining research.

- Decree 2017-0023/PRES/PM/MEMC/MINEFID dated 23 January 2017 on taxes and royalties.
- Decree 2017-0024 dated 23 January 2017 on the mining fund for local development.
- Decree 2007-901/PRES/PM/MCE/MS/MTSS dated 31 December 2007 on health and safety in the mining sector (Decree 2007-901).
- Order 2018-018 dated 20 June 2018 on standard models of specifications for holders of authorisations for artisanal and semi-mechanised mining of quarry substances.
- Order 2018-019 dated 20 June 2018 on specifications for holders of semi-mechanised mining permits and authorisations for artisanal mining.

20.2.3 Mining Code

The Mining Code (Law N°036-2015/CNT pertaining to the Mining Code of Burkina Faso) dated 26 June 2015, is the main statute governing mining in Burkina Faso and the Law on Gold (Law 028-2017/AN dated 18 May 2017) is also applicable to the marketing of gold and other precious metals. These are administered by the Ministry of Mines and Quarries - Ministère des Mines et des Carrières (MMC) and provides the legal framework for the mining industry in the country. The state owns title to all mineral rights and these rights are acquired through a map-based system by direct application to the MMC.

The Mining Code also incorporates various decrees, including but not limited to:

- The promotion of Companies' social responsibility and local community rights; the creation of a Fonds Minier de Développement Local (Decree N°2017-0024 Decree No. 2 on the organization, operation, and methods of collection of the Local Development Mining Fund).
- The mine rehabilitation and closure fund (Decree No. 2017-0068 on the organization, operation, procedures for collecting resources from the mine rehabilitation and closure fund).
- Local content whereby preference will be given to local suppliers, contractors, and national employees whenever possible (Decree N°2021-1142 establishing the conditions of local supply in the mining sector). Pursuant to this local content, annual threshold amounts are required from companies in the mining sector with respect to procurement and services.

There are three types of mining permits and three types of authorizations according to the Mining Code:

- Exploration Permit; Industrial Exploitation Permit; and Semi Mechanized Operations Permit.
- Prospecting Authorization; Traditional Artisanal Mining Authorization; and Quarrying Exploration and Operation Authorization.

Details pertaining to the above permits can be found in Section 20.2.5.

The Mining Code guarantees a stable tax and custom regime for the life of a mine in Burkina Faso. The Mining Code states that no new taxes can be imposed except for mining duties, land taxes and royalties. The title holder can also benefit from any reductions of tax rates during the life of the operating license. The holder of a permit or authorization is subject to the payment of fixed duties and proportional fees including a surface fee and a proportional royalty with the amount, the base, the rate, and the methods of recovery determined by regulation.

Once in production, the holder of an Industrial Operating Permit is required[1] to open, under the holder's name, a fiduciary account called "Fonds de préservation et de réhabilitation de l'environnement minier" (Fund for the Preservation and the Rehabilitation of the Mining Environment) at the Banque Centrale des États de l'Afrique de l'ouest (BCEAO) (Central Bank of West African States). This account must be funded annually on 1 January, by an amount equal to the total rehabilitation budget presented in the ESIA, divided by the number of years of production to cover the costs of mine reclamation, closure, and rehabilitation. The Company has opened an account and will fund such account in accordance with the applicable regulations.

20.2.4 Institutional Framework

The main institutional stakeholders for the environmental aspects of the Bomboré Mine include:

- Ministry of Mines and Quarries - Ministère des Mines et des Carrières (MMC).
- Ministry of Environment, Water and Sanitation - Ministère de l'Environnement, de l'Eau et de l'Assainissement (MEEA).
- National Agency for Environmental Assessments - Agence National des Evaluations Environnementales (ANEVE): this organization is part of MEEA and has the mandate to promote, regulate, and manage the environmental assessment process of the country. ANEVE holds sessions to review the terms of reference submitted by the project promoter. It formulates an opinion on the admissibility of studies and makes recommendations to the MEEA on the environmental acceptability of projects.
- Technical Committee for Environmental Assessments - Comité technique sur les Évaluations Environnementales (COTEVE): this organization was created by Decree No. 2006-025/MECV/CAB in 19 May 2006 establishing the powers, composition, and functioning of COTEVE. COTEVE is the technical and scientific framework to examine and analyse research reports and notices of environmental impacts presented by the project promoters to MEEA.

[1] Decree No. 2007-845/PRES/PM/MCE/MEF

- Direction Générale de Préservation de l'Environnement (DGPE).
- Direction Nationale des Eaux et Forêts (DNEF).
- Laboratoire d'Analyse de la Qualité de l'Environnement (LAQE).
- Ministère de l'Energie, des Mines et des Carrières (MEMC).
- General Management of Mines and Geology - Direction Générale des Mines et de la Géologie (DGMG).
- Bureau of Mines and Geology of Burkina Faso - Bureau des Mines et de la Géologie du Burkina (BUMIGEB).
- Chamber of Mines of Burkina Faso - Chambre des Mines du Burkina Faso (CMB).
- National Commission of Mines - Commission Nationale des Mines (CNM).
- Nakanbé Water Agency - Agence de l'Eau du Nakanbé

Other Ministries involved:

- Ministry of Infrastructure and Development (MID).
- Ministry of Territorial Administration, Decentralization and Security (MATDS).
- Ministry of Health and Public Hygiene (MSHP).
- Ministry of Agriculture, Animal Resources and Fisheries (MARAH).
- Minister of National Solidarity and Humanitarian Action (MSNAH).

20.2.5 Required Material Permits

Various permits and authorizations are required for the Bomboré Mine. Orezone Bomboré SA (OBSA) holds all permits that are required for its current operations and those envisioned in the 2019 FS.

The Phase II Expansion, as envisioned under the 2019 FS, has been approved with the formal decree dated March 23, 2021 (noted below). For the Phase II Expansion, not envisioned under the 2019 FS and included in this Report, OBSA will require an amendment to its Industrial Exploitation Permit. For the amendment, OBSA will be required to obtain an Environmental Compliance Certificate which first requires an approved ESIA.

Several additional permits and authorizations are required for the Bomboré Mine that are not listed in this section such as water authorizations with respect to the management of raw water from the MEEA and the Nakanbé Water Agency. The proposed expanded TSF is within the current permit boundaries however it will require a permit revision. Orezone has been successful in obtaining such permits and authorizations in the past and is confident that it will be able to obtain the required permits and authorizations for the Phase II Expansion.

20.2.6 Industrial Operating Permit / Mining Convention

In Burkina Faso, the Mining Code provides that exploration and exploitation permits shall be accompanied by a mining convention that the State conclude with the titleholder. In February 2019, Orezone signed the mining convention with the State of Burkina Faso. The purpose of the mining convention is to clarify the rights and obligations of the parties and to guarantee Orezone stability, including taxation and foreign exchange regulation. The mining convention is not a substitute for the law but specifies the provisions of the law. It is valid for the initial duration of the operating license and is thereafter renewable for one or more periods of five years at the request of Orezone. This mining convention will also be applicable for the Phase II Expansion. The application for an Industrial Exploitation Permit requires an Environmental Compliance Certificate to be issued by MEEA (which requires an approved ESIA).

- In January 2017, Orezone received the decree dated 30 December 2016, for the Bomboré Industrial Operating Permit.
- In March 2021, Orezone received the decree dated 23 March 2021, for the extension of the geographical perimeter of the industrial exploitation permit based on the 2019 FS.
- In June 2021, Orezone received the order dated 28 June 2021, amending the initial mine development plan based on the 2019 FS.

Following approval of the ESIA (discussed below), Orezone will apply to the MMC for the Industrial Operating Permit required for the Phase II Expansion.

20.2.7 Environmental Compliance Certificate / ESIA

The ESIA must be supported by a Feasibility Study (FS) and must include a Resettlement Action Plan (RAP) that has been accepted by all stakeholders if the project requires the expropriation of land held by any resident.

The ESIA development, review and approval procedure includes the following steps:

- Preparation of a project description and draft ESIA Terms of Reference to be made by OBSA.
- Review, approval, and finalisation of the ESIA Terms of Reference by ANEVE.

- ESIA development to be made by OBSA, supported by a national independent consulting company specialized in ESIA.
- Public consultation and information sessions to be conducted by OBSA.
- Collection of data from with respect to physical, biological, and human aspects of the ESIA and RAP.
- Public inquiry to be made by MEEA.
- Report of the Public Inquiry to be issued by MEEA.
- Consideration of the ESIA by COTEVE.
- Decision on the approval of the project to be taken by MEEA.
- Issuance of the decree on environmental and social compliance by MEEA.

In September 2020, Orezone received the order approving its ESIA based on the 2019 FS.

In early 2023 Orezone engaged Société de Conseil et de Realisation pour la Gestion de l'Environnement (SOCREGE), an independent West African consulting company, to assist Orezone with the ESIA required for the Phase II Expansion. In June 2023 the Terms of Reference were approved by ANEVE and SOCREGE, in consultation with Orezone. Orezone anticipates that the ESIA will be completed in Q4-2023 and approved in Q1-2024.

20.3 Baseline Studies

20.3.1 Baseline Studies Conducted

In July 2009, an Environmental Baseline Study was completed for the Bomboré mine by the Bureau d'Études des Géosciences des Énergies et de l'Environnement (BEGE) (BEGE, 2009a) located in Ouagadougou. The study focused on collecting data from existing sources and new field studies to establish an appropriate baseline for measuring the overall environmental and social impacts of the Bomboré mine. In September 2009, an Environmental Assessment Report was also completed by BEGE (BEGE, 2009b) to conceptually assess the environmental and community impacts related to the possible development of the Bomboré mine within the Bomboré 1 permit.

At the end of 2010, Orezone commissioned G Mining Services Inc. (GMSI) to prepare a National Instrument 43-101 (NI 43-101) Preliminary Economic Assessment (PEA) for the Bomboré mine. The PEA was delivered in June 2011.

In 2011, Orezone commissioned SOCREGE to conduct a socio-economic study of Bomboré, and BEGE to conduct an environmental impact study based on the carbon-in-leach (CIL) project description delivered by GMSI in June 2011. The Terms of Reference for both studies were validated as required by the ANEVE, which classified the Bomboré mine as a Category A project that was subject to an ESIA in accordance with Article 9 of Decree No. 2001-342/PRES/PM/MEE.

SOCREGE delivered an interim report in January 2012 (SOCREGE, 2012) on their demographic and socio-economic studies within the 2011 GMSI Bomboré footprint.

In June 2012, Orezone commissioned GMSI to prepare a NI 43-101 compliant feasibility study for Bomboré project based on an oxide milling CIL scenario. The 2012 FS engineering and geotechnical investigations led to a new proposed site layout, which required an expansion of the ESIA study area to be covered by both SOCREGE and BEGE. This area included the surface mine, infrastructures and access roads but excluded potential resettlement sites.

SOCREGE delivered a second interim report in January 2013 on their demographic and socio-economic studies within the expanded Bomboré footprint. BEGE delivered an interim report in July 2012 on their environmental baseline studies also within the 2011 GMSI Bomboré footprint and another interim report in June 2013 on their botanical and archaeological studies within the expanded footprint. In April 2013, Orezone commissioned Cabinet Archi Consult to conduct a financial evaluation of the replacement cost of the buildings potentially impacted by the Bomboré development and to produce the architectural plans and site layouts for the resettlement sites and buildings. As of December 2013, detailed baseline studies were completed over a study area that covered 83 km^2, and baseline studies, discussed below, were in progress in adjacent areas favorable to host the relocated population.

In 2014, BEGE delivered a report setting out the results of the environmental baseline studies conducted since 2009 on the 2013 GMSI Bomboré footprint. New Terms of Reference were prepared and sent to ANEVE in July 2014 based on the changes to the project. In parallel, updated environmental and social baseline studies were conducted by BEGE, SOCREGE and Cabinet Archi Consult. This baseline characterization of the physical, biological and human components was done through different field missions that occurred during both the 2014 wet and dry seasons and until February 2015 for the human components.

In May 2015, Orezone applied for an Industrial Exploitation Permit to the Ministry of Mines and concurrently submitted to the Ministry of Environment a preliminary version of the ESIA for the project together with the preliminary RAP. ANEVE conducted the Public Enquiry in November 2015.

In January 2016 Orezone presented the ESIA together with the RAP and Feasibility Study (FS) to the Burkina Faso authorities (COTEVE). The ESIA and RAP were reviewed by Burkina Faso authorities and Orezone addressed all the comments with final revised versions of both documents submitted to the Ministry of Environment on 2 April 2016. The Minister of Environment delivered to Orezone on 12 May 2016 a favorable opinion about the project by way of Arrêté n°2016-0295/MEEVCC/CAB.

In January 2019, Orezone commissioned SOCREGE to update the socio-economic study of the project, and BEGE to update the environmental impact study of the project. These updates were required by Orezone to include the hard rock expansion phase of Bomboré project and other additional areas (P17S area located in the south and three areas in the seasonal rivers floodplains).

In March 2020, Orezone presented the ESIA and the updated RAP to COTEVE for the environmental feasibility of the Phase II expansion and was issued a favorable opinion. Orezone then presented this to the National Mining Commission, which validated it and in March 2021 the Minister of Mines issued Orezone the decree for the expanded mining permit (Decree n °2021-0144 of 23 March 2021).

20.3.2 Description of the Main Environmental and Social Components

The Bomboré site is within the Sahelian climate zone and has a semi-arid type of climate subject to a wet season from approximately May to October and a dry season from approximately November to April. The average annual rainfall varies from 700 mm to 900 mm. Monthly minimum and maximum temperatures recorded by Orezone on the property range from 18°C in December-January for the average monthly minimum and 40°C in April for the average monthly maximum. Monthly minimum and maximum humidity levels during the same period ranged from 6% in the dry season (measured in February) to 96% in the wet season (measured in August and September) with high evaporation and evapotranspiration rates. Sunshine is nine hours per day on average. Prevailing winds during the wet season are from the west direction, shifting to the dry Harmattan from the east direction during the dry season.

The land is generally flat with a few scattered hills whose altitude reaches up to 344 m; these hills consist of Birimian rocks typically overlain by ferruginous duricrust formations. The Bomboré site is in a low-risk seismic zone (there have been no historical earthquakes that have been recorded in close proximity to the Bomboré site as noted by the International Seismological Centre. [1] Surface water and stream flow is confined to the wet season up to November. Siltation of any river-based water storage facilities is extremely rapid due to the torrential nature of rainfall in the region. The main water courses include the Bomboré River and the Nobsin River, a Bomboré tributary.

Field hydrogeological drilling results indicate that the water table is present within the first 40 m below surface and most of the groundwater occurs within the first 80 m below surface. Groundwater is generally of good potable quality. The Groundwater quality is within International Finance Corporation (IFC) standards for drinking water except for Arsenic (As) (IFC guideline of 0.1mg/l) for some locations (up to a maximum of 3 mg/l), identified during the baseline monitoring. The As level is not an issue for current operations and therefore it is reasonably expected not to be an issue for the Phase II Expansion as the water will be collected in the open pits and the different sediment ponds through the operation, reused in one of the processes or sent to the sedimentation ponds where it will be monitored and if necessary, treated before discharge.

[1] Bondár, I. and D.A. Storchak (2011). Improved location procedures at the International Seismological Centre, Geophys. J. Int., 186, 1220-1244, doi: 10.1111/j.1365-246X.2011.05107.x

Eight morpho-pedological units were identified and ascribed to four major units with respect to their agroforestry potential, from totally improper for agriculture (Unit A: 12% of the Bomboré area), to marginally apt for pluvial crops (Unit B: 15% of the Bomboré area), apt to moderately apt for pluvial crops, (Unit C: 57% of the Bomboré area) and apt to moderately apt for pluvial crops, pluvial rice and orchards (Unit D: 16% of the Bomboré area). Eroded areas are numerous and are caused by run-off water, wind, and human activities.

Deforestation is widespread over the permit area. Vegetation in uncultivated areas is mainly composed of savannah with denser vegetation, shrubby vegetation, and riverine forests, growing only along the watercourses and the draining system. The main vegetation types comprised inside the study area can be divided into four main types, i.e., cultivated parcels (42% of the Bomboré area), savannah (37% of the Bomboré area), riverine forest and riparian formations (15% of the Bomboré area, essentially along the Bomboré and Nobsin Rivers and tributaries) and barren lands (6% of the Bomboré area).

Due to human pressures and degradation of the natural environment, terrestrial fauna is generally scarce and avian fauna is diversified. Hare, hedgehog, squirrel, rat, wild cats, varan, land turtles, frogs, toads, and adders have been observed. Bird species observed include pigeons, turtledoves, guinea fowls, partridges, herons, ducks, coucals, and brown vultures. Fish species including carp, catfish, sardine, tilapia, and lungfish have been recorded inside the hydrological network. No terrestrial species has a conservation status at either national or international level, except for one bird species, the Hooded Vulture (Necrosyrtes monachus), that is Endangered according to the International Union for Conservation of Nature (IUCN) red list. A Critical Habitat Assessment has been undertaken by Orezone as well as an Integrated Biodiversity Assessment Tool (IBAT). Mitigations and conditions are in place to ensure the Bomboré Mine has no net loss of biodiversity and a net gain of biodiversity for critical habitats. No critical habitat or dangerous species have been recorded within the IBAT buffer zones.

In addition to degradation of vegetation and animal species, the artisanal mining activities have caused a significant change in the soil and surface water quality. Digging of tunnels and accumulation of rocky material, creating small water tanks, the use of chemical products for the treatment of ore and wastewater discharges, are activities that modified the environmental conditions in sectors in which these activities are practiced. These changes resulted in soil erosion and water pollution. Section 14 of this Report provides further details on historic artisanal mining activities.

On the administrative and human terms, the land ownership within the Bomboré area is peculiar due to the coexistence of traditional and modern land tenure schemes. In the traditional system, families have inherited or have been granted the right to use the land by the state but there is no private ownership of the land.

The total population in the Bomboré area is approximately 7,700 inhabitants. Women represent on average 52% of the population and each household has an average of seven people. Youth (age 0-20) account for 60% of the population. In addition, two artisanal mining communities (Kagtanga and Sanam Yaar) have a combined population of approximately 4,600 inhabitants. These two communities have been successfully resettled as part of the RAP Phase I.

Only about 16% of the community within the Bomboré area is literate. Health facilities are very basic with only two public health clinics in Nobsin and Mogtedo V3 that are within the Bomboré area. More than 35% of the population suffers from malaria. The number of AIDS (acquired immunodeficiency syndrome) cases has declined steadily with approximately 12,000 cases in 2000, 6,100 cases in 2010 and 2,200 cases in 2020.

Agriculture is very important in the Bomboré area, but yields are low and declining due to the intensive nature of the activity with sorghum, corn, and millet accounting for the bulk of the production. Most of the producers (86%) are using seeds from their own crop and organic manure as fertilizer but 63% also buy some chemical fertilizers. Most households practice animal husbandry; own livestock consisting of poultry, cattle, sheep, and goats. The collection of non-ligneous forest products is also a source of food and medicinal plants. Depending on location, almost three quarters of households practice artisanal mining, which serves as valuable source of income. Crafts, hunting, and tourism are marginal activities.

Most households do not have latrines and waste is generally disposed using methods as a cathole. Several wells are present in the Bomboré area and include some large diameter traditional wells and new drillings. Currently there is no access to the electricity network, Société Nationale Burkinabè d'Electricité (SONABEL), in the Bomboré footprint with only a few households having private equipment (solar panels or generator) that is used mainly for lighting, to recharge mobile phones and to watch national or satellite television.

Inventories carried out in 2012, 2014 and 2019 identified 156 archaeological sites and 195 ethnographic sites (graves and sacred sites) in the Bomboré area. In accordance with customary traditions, national and local laws, certain objects and artifacts were collected by Orezone and stored for their preservation under the control of the Department of Archeology at the University of Ouagadougou. Following ceremonies led by the local community leaders, some of the impacted sacred sites have been relocated to areas outside of the project area and the sacred sites where relocation is not possible, these sites are now protected by OBSA with full access by the impacted communities to those sites.

20.4 Community Information and Consultation Program

The stakeholder information and consultation process are an integral part of the ESIA. Orezone has put in place mechanisms and communication tools so that all those involved in, or affected by, the Bomboré mine can freely express themselves. Key areas include air quality, water, biodiversity, access to land resources, alternate livelihood and employment and social services / infrastructures. The information collected during these consultations has helped identify issues, risks, benefits, and opportunities for the Bomboré mine to avoid, minimize, or offset impacts associated with mining operations and enhance the positive ones.

As part of the stakeholder information and consultation process, a Stakeholder Engagement Plan (SEP) was developed. To develop the SEP, information about the Bomboré mine was transmitted by Orezone using information sheets. These information sheets were discussed by Orezone at numerous meetings with administrative authorities, technical services, as well as representatives of the surrounding communities and public radio broadcasts.

Many initiatives have been undertaken by Orezone to inform and consult with affected communities. These initiatives have ensured the integration of environmental and social issues in project design and include the following agreed steps:

- Establishment of an Orezone permanent team for environmental and community relations.
- Adoption of a Stakeholder Engagement Plan.
- Establishment of a grievance mechanism procedure.
- Development of local recruitment and supplier capacity development plan.
- Adoption and implementation of a Sustainable Community Development Program.
- Numerous ad hoc meetings with authorities and other stakeholders.

A Provincial Compensation and Resettlement Committee of the people affected by the Bomboré mine was set up by Order No. 2013-010/MATS/RPCL/PGNZ/HC-ZRG dated 28 May 2013, and was officially activated on 4 April 2014. The first public meeting was held in July 2014 to discuss issues related to resettlement. In addition, Orezone has established, at early stages of the study, a community information and consultation mechanism, which has been implemented throughout the various ESIA processes. The main concerns raised during the communication activities included:

- Disturbance of subsistence activities.
- Compensation to be supplied to traditional landowners.
- Air, water, Fauna / Flora, and soil degradation.
- Disruption of sacred sites.
- Promotion of women.
- Assistance to vulnerable people.
- Restoration of livelihoods.
- Access to jobs and training.
- Influx of foreign workers and spread of disease.
- Road safety and accident prevention.

- Closure plan and the safe take-over of the land by the local communities after the mine closure.
- Control and transparency during the implementation of social and environmental compensation measures.

Orezone considered these concerns expressed by stakeholders and incorporated specifications to optimize the Bomboré mine design to avoid and manage any of these constraints. The majority of these efforts are directed toward community health and safety, educational programs, vocational training, food security and regional development in addition to opportunities for local employment and support for small businesses. Examples include local hiring for RAP Phases I, II and III construction, skills training for future job applicants and employees (i.e., heavy equipment) and support for new community businesses and subsidence programs (i.e., soap making, blanket weaving, agricultural gardens, chicken breeding, tree nurseries, and promotion of land reclamation techniques to improve yields and areas of arable land). These actions led to a more balanced approach between the financial objectives of Orezone and the preservation and conservation of the environmental and social components, which are integral part of sustainable development.

20.5 Bomboré Impacts, Risk Analysis, Environmental and Social Management Plan

20.5.1 Bomboré Impacts

The methodology used to identify and analyze the environmental impacts is based on an approach recognized by international funding agencies. This approach identifies the direct interactions between activities at Bomboré and the impact on physical, biological, chemical and human components. These interactions are customized according to project-specific phases (pre-construction, construction, operation, and closure). All interactions identified are then analyzed based on three criteria (intensity, extent and duration) to obtain a global indicator, of the absolute importance of the impact. The importance of the impact is then qualified as; minor, medium, or major.

Most of the impacts on the physical environment are of low or medium absolute importance given the predicted disturbances on air, soil, surface, and groundwater during the construction and operational phases, subject to proper implementation of the mitigation measures. The mining operations have been designed for zero water discharge, which is a clear advantage in terms of minimal impact which ensures that under no circumstances will process water be released to the environment (see section 17 of this Report). Protection of the natural groundwater from process water containing cyanide and from sulphide waste rock dumps run-off water containing leachable metals was properly planned.

Impacts on the biological components are mostly minor and a Biodiversity management plan is under development for better mitigation during the different Bomboré phases. Impacts on the human components have an importance ranging from minor to major depending on the particular issue. The most significant impact caused by the Bomboré mine is the resettlement of certain communities as further discussed in section 20.6.

The economic impact of the Bomboré mine at local, regional, and national levels is extremely positive. Beginning with the construction phase, direct and indirect jobs have been created, resulting in tangible economic benefits for both local and regional communities. As of 30 June 2023, there were 1,262 contractor personnel and 637 permanent and temporary Orezone employees directly involved with or supporting mining, processing, exploration, and capital project activities at Bomboré. Burkinabé citizens comprise 98% of this direct workforce with female representation at 8%. The workforce requirement will increase with the Phase II Expansion.

The procurement of local goods and services have resulted in significant economic benefits to local and regional businesses. Orezone has and will continue to be in compliance with the decree for local procurement and supply. For example, in 2022, the Company's purchase of goods and services, the share of local suppliers represented 74.6% against 25.4% for foreign suppliers.

The revenues generated by the mine operations increase Burkina Faso's internal revenue through taxes and royalties levied by the local and regional authorities (see Chapters 15 and 21). These revenues have a beneficial impact at the local and regional levels through increased investments in social and health services, and local infrastructure. In addition, Orezone supports several social programs for displaced households and in a broader context, local and regional communities (such as agriculture modernization, women empowerment, soap production, woven loincloth production, chicken breeding).

Small scale artisanal mining activities currently exist in the Bomboré area. Orezone has identified within its exploration licenses two economically viable deposits that miners from the two expropriated communities, Sanam Yaar and Kagtanga, could apply to the state for legal artisanal mining permits without objection from Orezone under the supervision of National Agency for the Supervision of Artisanal and Semi-Mechanized Exploitations.

In accordance with a memorandum of understanding signed with artisanal miners in 2015, the artisanal miners agreed to form and govern co-operative amongst themselves. The co-operative is currently being developed with OBSA's support. Pursuant to the memorandum of understanding, Orezone retains a right of first refusal should any permits be abandoned by the artisanal miners or sold in the future.

20.5.2 Risk Analysis

A Preliminary Risk Analysis was conducted to assess the environmental risks of the Bomboré mine. Like any other heavy industrial activities, the Bomboré mine may unintentionally experience critical issues like spills, emissions and fires that could have a direct negative impact on the surrounding environment. The causes and consequences of each of these situations were determined and detailed preventive and emergency implementation measures have been identified. The criteria considered for this risk assessment consider the severity of events, as well as the consequences and the likelihood of an occurrence.

An analysis of the facilities and consumables used on the Bomboré mine site revealed several inherent risks. The main environmental risks associated with the Bomboré mine are as follows:

- Fire.
- Explosion.
- Degradation / failure of walls and ramps in the pits and waste dump areas, berms, and retention structures.
- Acid and Metalliferous Drainage (AMD).
- TSF dam failure, tailing overflow through spillway to ambient environment.
- Spills or leaks of hazardous materials.
- Toxic emissions.
- Natural disasters.
- Insurrection of the population.

To minimize the level of risk related to both personnel and the environment, health and safety and security measures have been implemented in accordance with best practices and the laws of Burkina Faso. These measures are led by experienced managers in all Bomboré departments including, mining, processing, environmental and social, health and safety, and security.

An Emergency Response Plan (ERP) has been implemented for operations. The main objective of the ERP is the management of those risks, which cannot be eliminated by the protection measures already in place so that the ERP will immediately be initiated if any such incident or accident occurs. The intent of the ERP is to define emergencies that could reasonably occur, and the measures of prevention, preparedness, response, and repairs required for such situations, including staff training.

20.5.3 Environmental and Social Management Plan

The Environmental and Social Management Plan (ESMP) describes the environmental and social management measures to be implemented at Bomboré during all project phases. The ESMP covers the avoiding, minimizing, enhancing, or compensating of the various anticipated negative impacts by reducing them to an acceptable level for all stakeholders.

The ESMP identifies the objectives to comply with the regulations in Burkina Faso and international best practices in the mining sector. The ESMP also includes environmental monitoring programs and environmental and social follow-up, providing the basis for assessing the effectiveness of management measures to implement by Orezone. The ESMP also includes several measures to strengthen the capacity of the stakeholders concerned by the application of environmental and social management measures.

Management measures were implemented at the earliest stages of the construction phase and continue into operations. The management measures for the physical, biological, and human components include, among others, the following:

- Protection of biodiversity and soils.
- Management of hydrocarbons and chemicals.
- Control of run-off water, restrictions during heavy rain periods, respecting buffer zones along watercourses, etc.
- Implementation of restrictions regarding land clearance, topsoil and cleared vegetation management, limits for working areas, etc.
- Reduction of noise, gas, and dust emissions.
- Efficient use of water and regular monitoring on quality and quantity.
- Execution of a sustainable land reclamation with rehabilitation strategy (applicable to the bioclimate and social condition with external stakeholders' agreement).
- Control of traffic speed, access roads, the use and maintenance of equipment (fuel and lubricant tanks, vehicles, and motorized equipment, etc.).
- Management of human resources, logistics, mobilization and demobilization of personnel and contractors.
- Management of the arrival of unwanted ‘opportunistic’ populations in the area.

- Maximization of job opportunities for the local workforce, of supplies of goods and services by local stakeholders, and of women’s benefits and management of unrealistic expectations.
- Improvement of local content by capacity development of local workforce and suppliers.
- Population and workers awareness to the risks of transmitting HIV / AIDS and other endemic diseases.
- Precise location and protection of worship and sacred sites.
- Implementation of an alternate livelihood restoration program through sustainable approach accepted by the community and viable in the project area in accordance with socio-economic, cultural, and climatic context.
- Continued implementation of a community relations covering stakeholders' engagements, grievances management, integration of vulnerable community, empowerment of women.
- Protection of cultural heritage.

Additional operating measures at the Bomboré mine include the following:

- Monitoring of the mine tailings storage facility in compliance with the applicable regulations and requirements.
- Management of waste rock dumps and progressive re-vegetation to restore as much as possible to the initial state, minimize wind and water erosion to better mitigate dust emissions and sediment discharge.
- Management of air, water, hazardous materials, wastes, traffic, maintenance of vehicles, etc.
- Mining is carried out according to best practices and with specific attention to occupational health and safety.
- Orezone will further investigate potential arsenic leach from Waste Rock Dump as well as a complementary Geochemical investigation to allow for the elaboration of the AMD (Acid and Metalliferous Drainage) Management Plan. The geochemical studies conducted to date note that arsenic leaching is minimal.

An ongoing monitoring program will be implemented for the construction of the Phase II Expansion. The program will ensure compliance with the commitments agreed to as part of the ESIA and environmental obligations, as well as compliance with the proposed management measures and with laws, regulations, and other environmental considerations.

Although the Bomboré area includes habitats heavily modified by human activities, including degraded critical habitats, it supports some special-status species in terms of biodiversity even though the IBAT conducted indicate that there is no endangered species or area within the 50 km buffer. Bomboré's environmental acceptability by the National Authorities as well by the regional and local communities is related to the consideration of these biodiversity considerations.

The environmental and social follow-up programs include:

- Monitoring changes for sensitive environmental components: soil, water, air, biodiversity.
- Comparing current conditions with pre-Bomboré initial conditions to identify trends or impacts that may result from Bomboré activities or natural events.

The main elements monitored include:

- Surface and ground water quality.
- Ambient air quality.
- Ambient noise.
- Status of the flora and effectiveness of re-vegetation.
- Biodiversity preservation – Fauna / Flora.
- Waste management including hazardous waste.
- Local and regional economy.
- Gender.
- Social cohesion.
- Livelihood and restoration program.

Various management measures are planned for the closure phase and include the following:

- Dismantling of infrastructure and facilities, except for structures that will be kept in place and handed over to the local authorities without compromising the integrity and security of places and people.
- Site rehabilitation and re-vegetation.

- Restoration of alternative livelihood conditions for neighbouring populations and workers as agreed in the stakeholder's engagement plan.

20.6 Resettlement

20.6.1 People and Activities Affected by the Bomboré Mine – Staged Resettlement

The resettlement of communities is progressive and pursuant to the 2019 FS was divided into Phases I, II and III as a function of the mine plan. The resettlement of people is a complex activity but due to a focused effort from Orezone, Phase I was successfully completed in early 2021 and community feedback on the new resettlement has been overwhelmingly positive, with concern of improving the resettlement conditions in the future. As such, Orezone is confident that Phases II and III will be successful as well. A Phase IV will be required to access additional areas contemplated in this Technical Report that were not included in the 2019 FS.

Phase I covered an area of 1,833 ha and included the physical and economic displacement of approximately 428 households (2,879 people) and 516 ha of farmland, specifically portions of the communities of Nobsin, Goingo, Mogtédo V4 and Mogtédo V5. The artisan mining communities of Sanam Yaar and Kagtanga (approximately 1,364 households / 3,100 people) were also resettled. Orezone successfully completed Phase I in early 2021 with 1,102 private homes, 66 public infrastructures (schools, hospital, etc.) and community roads built. The resettlement of certain old communities to new homes significantly reduced the negative impacts to the inhabitants.

The resettlement of the communities for Phases II and III is scheduled to be completed in 2024 and consist of the following:

Phase II earthworks and construction commenced in Q2-2023. Phase II covers an area of 1,055 ha where approximately 491 households (2,789 people) will be relocated and 440 ha of farmland compensated, specifically for the communities of Mogtédo V3, Mogtédo V2 and Bomboré V2. The Phase II socio-economic assessment was updated in 2022.

Phase III covers an area of 180 ha where approximately 169 households (1,007 people) in the vicinity of P17S area will be relocated and 88 ha of farmland compensated, specifically for the inhabitants of Bomboré V2. The Phase III socio-economic assessment was updated in 2022 and is expected to be completed in 2024. See Chapter 4 for the location of the P17S area.

The reports issued by SOCREGE, BEGE and Cabinet Archi Consult, complemented by Orezone field validation and updated inventories as of June 2019 were used as a baseline for the Resettlement Action Plan that is included in the 2019 ESIA. These reports contain an inventory conducted on the households and their properties in their community within the study area, in addition to an overall inventory of the existing public infrastructures.

Phase IV covers an area of 239 ha where approximately 330 households in the vicinity of Bolin, BV2, Razinghin and community water reservoir area will be relocated and 275 ha of farmland compensated, specifically for the communities of Bomboré V2, Bolin and community water reservoir. The Phase IV socio-economic assessment is being updated and it is expected that this update will be completed in December 2023. Phase IV is planned to be completed progressively over 2024 through to 2027.

20.6.2 Scope of Resettlement

Phases of the RAP scope are based on the following assumptions:

- Pursuant to the RAP (see Section 20.2.5), the area impacted is based on the site layout with an agreed to 500 m buffer zone to be expropriated around the mining infrastructure (pits, pads, ponds, haul roads, etc.) to reduce the impact of typical mining operations (i.e., dust, noise) on local populations.

- Census data and property inventories as of February 2022 for Phase II and Phase III. Census data and property inventories as of October 2023 for Phase IV.

- Best-practice strategy to comply with national standards (Loi n° 017-2006 portant Code de l'urbanisme et de la construction, Loi n° 23/94/ADP portant Code de la Santé publique). The budget includes the replacement of all private houses, a latrine for each household, all public infrastructures upgraded to the national standards, plus financial compensation for granaries, sheds, ovens, parks, shops, roosts, and similar small infrastructures. Beyond the replacement of impacted private property, Orezone has set up a livelihood restoration program. This program consists of capacity building for those affected by the Bomboré mine and funding for income generating activities.

- Cash compensation for expropriated farmlands at market value plus crop compensations over five years are included in the estimate. Crop compensations are based on the value of a basket of harvested products, as established from the average yields and market values compiled by the local branch of the Ministry of Agriculture, Animal Resources and Fishery (Ministère de l'Agriculture, des Ressources Animales et Haleutic (MARAH)) for the area, and the surface area of the parcel, as recorded during the field study.

20.7 Acid Rock Drainage and Metal Leaching

For Phase I Oxides, samples of waste rock, potential construction materials and tailings were subjected to independent laboratory geochemical tests to assess their potential to generate Acid Rock Drainage (ARD) and to leach metals (ML).

The results of the ARD assessment were compared to the evaluation criteria presented in the Global Acid Rock Drainage (GARD) Guide (INAP, 2009), a reference document on best practices related to mine waste characterization and ARD prediction, prevention, and mitigation measures. Results of the metal leaching tests, and process water chemistry were compared to the applicable effluent discharge guideline values specified in Section 2.0 of the IFC/World Bank Group EHS Guidelines for Mining (IFC, 2007).

20.7.1 Waste Rock and Construction Materials

The ARD/ML assessment of the waste rock focused on dominant rock types and weathering zones (Oxide and Transition) from the CFU, KT, Maga, P8P9, P11, P16, P17, Siga East, West, and South prospects. Three separate sampling and analytical programs were initiated in 2011 (McClelland Laboratories, Inc.), 2012 (Golder Associates) and 2019 (Knight Piésold), a total of 141 rock samples were collected from exploration drill cores from within the open pit outline while the laterite was collected from a nearby borrow source. The results of this ARD/ML programs to date are presented below.

Approximately 90% of laterite, oxide (saprolite), and transition (saprock) units demonstrate little potential to generate ARD (non-PAG) and are not expected to leach metals at concentrations above the Burkina Faso nor the IFC effluent guidelines. Therefore, laterite, saprolite, and saprock material is considered suitable as potential construction material for most rock types.

However, saprock from the mafic intrusive (MI3) unit at P8P9 and Siga South, as well as the meta-sandstone (S3) unit at P8P9 and Siga South & East as well the meta-pelite (S4) indicated some potential to generate ARD. In addition, the meta-pelite (S4) unit indicate Arsenic leach potential. See Chapter 4 for the location of the various pits noted.

Thirty-nine weeks of data collected from five humidity cell tests being performed in 2019 by KP indicate metal concentrations at or below detection limits in the leachates collected from the P8/P9, Siga South, and P17 pits. All Constituents of Concern are below the IFC and the World Health Organization (WHO) guidelines. The pH of the HCT leachates had an overall downward trend from week 4 to week 39. All the humidity cells remained circum-neutral at the end of 39 weeks of testing. Additional geochemical investigation will be carried out to cover the expansion zone.

As part of the long-term management of these waste materials, an Acid Rock Drainage management plan with the continuous monitoring strategy and plan will be developed. The Potential Acid Forming material will not remain exposed to the atmosphere (i.e., will not be placed near the top or edge of a waste rock pile) to avoid “hot spots” of high ARD potential. They will be encapsulated.

20.7.2 CIL Tailings

The tested tailings samples were sourced from both a non-acid generating (S at 0.1%) and sulphide head sample (S at 1.78%) master composite samples that were subjected to bench-scale laboratory testing with the carbon-in-leach. The Phase I Oxide extraction as well as the Phase II Expansion is 100% CIL circuits. See Chapter 13 for a further discussion on CIL tailings.

Tailings solids and process water subjected to geochemical testing were prepared during the metallurgical testing conducted by McClelland on an oxide Master Composite (MC) sample containing 50% medium-grade and 50% high-grade materials. The oxide MC sample was subjected to a cyanidation process, after which a portion underwent cyanide destruction. The solids used for static testing were obtained from a portion of the materials subjected to cyanide destruction (CY142 and CY149), while the process water chemistry comes from decanted water both cyanided but not treated and cyanided with cyanide destruction (CY149).

Humidity cell testing was conducted in 2019 on one sample that most represents the material that will be processed at the plant (BL402 Test CN48 – Blend Fresh [15]: Oxide [85]) and released into the tailings. Results from 39 weeks of humidity cell testing indicate the tailings sample has neutral pH (ranging from 7.5 to 8.4), with a total and free cyanide below the IFC guidelines. The only CoC that exceeds the WHO guidelines is arsenic, with concentrations ranging from 0.01 mg/l to 0.06 mg/L.

The oxide MC tailings sample has a low potential in generating ARD as demonstrated by neutral kinetic test results, however arsenic exceeded the standards Burkina Faso of 0.1 mg/l. Process water reports indicate cyanide concentrations of 1.9 mg/L, which is above the IFC effluent guidelines for effluent (1 and 0.5 mg/L, respectively) and the Burkina Faso total CN guideline (0.1 mg/L). In addition, the arsenic and copper concentrations all exceed their applicable IFC effluent guidelines while arsenic and copper exceeded Burkina Faso 's standards. However, since the Bomboré tailings storage facility (TSF) will be zero discharge, effluent values do not apply and there will be no issues with effluent quality under dry conditions. In the wet season, water will be managed in accordance with the Law on Water Management and Orezone's Water Management Plan specific to the Bomboré project.

The Bomboré TSF is lined and designed to be a zero-discharge facility. Cyanide levels are low due to natural degradation. Typical experience for similar facilities in West Africa are CNwad (weak acid dissociable cyanide) levels of 1-10 mg/l in the supernatant pond, which is significantly less than the guideline value for protecting bird life (typical threshold value of 50 mg/l). The TSF has been designed and is managed in accordance with Australian National Committee on Large Dams (ANCOLD), "Guidelines on Tailings Dams – Planning, Design, Construction, Operation and Closure", July 2019, and the "Global Industry Standard on Tailings Management" (GISTM), August 2020.

20.7.3 Solid Waste

Solid waste generally includes bags, pallets, empty drums, worn out parts, liners, and other supply packaging. For all non-recyclable wastes, a solid waste disposal site has been created at the Bomboré mine in a suitably enclosed area, restricted to prevent animal access, and located to avoid contamination of water and vegetation. The general and recyclable are transported from site to a certified waste disposal facility located near Ouagadougou by EBTE Tout Nouveau, a certified waste contractor.

A high temperature incinerator with double chamber and pyrolytic system is being considered for solid waste management.

20.7.4 Hazardous Waste

Hazardous wastes, which primarily include waste oils, packaging for process reagents and laboratory chemicals, are disposed of in a safe and environmentally sound manner. The certified supplier recycles waste oils, while most reagents and chemicals that require disposal are disposed of in the lined TSF. Biomedical waste, hydrocarbon and hazardous waste, empty sodium cyanide boxes and inner bags are transported to a certified disposal facility as noted above.

Any occurrence of spills of hazardous materials on site are given the highest operating priority and generally include the excavation of contaminated soils, neutralization of the affected site, disposal and/or neutralization of the impacted soils on site to the bioremediation facility. A soil test is conducted, and the materials transported to the topsoil stockpile once the results are compliant. The mining equipment on site is immediately available for use in the event of a spill.

As noted above, a high temperature incinerator with double chamber and pyrolytic system is being considered for hazardous waste management.

20.7.5 Sanitary Wastewater

At the Bomboré camp, a sewage piping network collects sewage from the various lavatories, showers and laundry facilities and sent to a modular bacterial digester wastewater treatment system to process both black and grey waters. This unit has been designed not to contaminate groundwater or existing watercourses. A sewage treatment plant will be setup for sewage and grey water treatment and reuse for dust suppression and trees watering.

In the process area, satellite workshops and small office buildings, a standard septic tank and soak-away drain-field system are used for each respective building, with sludge trucked to the sewage treatment plant for further treatment.

20.8 Closure, Decommissioning and Reclamation

The closure, decommissioning, and reclamation costs of the Bomboré mine are estimated at US$ 19.1M and was included in the financial analysis for these closure activities related to the environmental and social aspects. This estimate is an update from the estimate included in the 2019 FS.

Once in production, the holder of an Industrial Exploitation Permit is required (pursuant to Decree No. 2007-845/PRES/PM/MCE/MEF) to open, under the holder's name, a fiduciary account called "Fonds de préservation et de réhabilitation de l'environnement minier" (Fund for the Preservation and the Rehabilitation of the Mining Environment) at the Banque Centrale des États de l'Afrique de l'ouest (BCEAO) (Central Bank of West African States). This account must be funded annually on 1 January by an amount equal to the total rehabilitation budget presented in the ESIA, divided by the number of years of production to cover the costs of mine reclamation, closure and rehabilitation. Following OBSA's application to open the account, the Fonds d'Intervention pour l'Environnement (FIE) "Environmental Intervention Fund" has informed OBSA of the opening of BCEAO account no. 261 2200 C000 60715 under the heading "Fonds de réhabilitation et de fermeture de OREZONE BOMBORE SA". The effective opening of this account facilitates the rehabilitation of the mining environment.

The Closure and Rehabilitation Plan includes work to be conducted from the closure of the mine, at the end of operation activities, as well as progressive rehabilitation work. As part of the Closure and Rehabilitation Plan, the following requirements will be adhered to as applicable:

- Local Development Mining Fund (Fonds Minier de Développement Local).
- Mine Rehabilitation and Closure Fund (Fonds de réhabilitation et de fermeture de la mine).
- Fund for the Rehabilitation and Securing of Artisanal Mining Sites and to Combat the use of Prohibited Chemicals (Fonds de réhabilitation, de sécurisation des sites miniers artisanaux et de lutte contre l'usage des produits chimiques prohibés).
- Fund to Finance Geological and Mining Research and Support Training in the Earth Sciences (Fonds de financement de la recherche géologique et minière et de soutien à la formation sur les sciences de la terre).

The goal is to return the site, pursuant to the ESIA, to a satisfactorily agreed state as quickly as possible in terms of:

- Reducing the risks for health and safety.
- Controlling erosion.
- Limiting maintenance and monitoring.
- Developing a compatible profile with the future uses of the site, primarily for the plant site.

The main objectives of the Closure and Rehabilitation Plan include restoring ecosystems and take-over and recovery of land uses. This plan includes:

- Dismantling and removal of plant equipment, machinery and infrastructure (except for structures that will be kept in place and handed over to the local authority without compromising the integrity and security of places and people).

- Progressive rehabilitation to allow rapid recovery of the vegetation cover and the early recovery of the ecosystem.

- Sustainability of rehabilitation work and control of water and wind erosion.

- Take-over and recovery of land uses.

- Maximization of material and equipment recovery.

- Site rehabilitation as part of a participatory approach involving interested communities.

- Implementation of a post-closure monitoring program.

In addition, a waste rock dump development program will be implemented and will notably include the development of agricultural plots. All structures that can be used by communities will be maintained, except for all facilities that may constitute a risk to people or the environment.

Access roads, power lines and other infrastructures built for mining will be released to the municipality, as necessary, for use by communities at the end of mine life. Restricted areas may be defined within the permit to protect the environment, the natural habitat, archaeological sites, or public interest infrastructures all in agreement with community and authorities.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimate

The Project Capital Cost Estimate (CCE) was compiled by Lycopodium process and engineering teams, assisted by Orezone.

The Bomboré Phase II Expansion Study is based on an annual ore feed rate of 10.3Mtpa with delivery of higher-grade ore in the earlier years. The Study maintains the current 5.9Mtpa oxide plant throughput and includes the commissioning of a new and independent 4.4Mtpa hard rock plant in 2025. Construction of the new standalone 4.4Mtpa hard rock processing facility will commence in 2024.

The CCE reflects the Project scope as previously described in this study report and is summarized in Table 21.1.1.

Table 21.1.1 Capital Cost Estimate Summary (US$, 2Q23, +15 / -15%)

Main Area	Capital (US$M)
Construction Distributables	14.5
Treatment Plant	71.7
Reagents & Plant Services	9.4
Infrastructure	13.2
Subtotal	**108.8**
Management Costs	15.1
Owners Project Costs	32.6
Subtotal	**47.7**
Project Total (excl. Contingency)	**156.5**
Contingency	11.0
Project Total incl. Contingency	**167.5**

All costs are expressed in US dollars (US$) unless otherwise stated and are based on Q3 2023 pricing. The estimate is deemed to have an accuracy of + 15 / - 15%.

A total contingency of US$11 million (M) has been included in the financial model which is 7.1% of the project cost before contingency. Total estimated project cost including contingency is US $167.5M.

21.1.1 General Estimating Methodology

General arrangement drawings were 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 including the conveyors and plant related infrastructure within the plant boundary. Where appropriate these quantities were checked and verified against the "as built" quantities from the Phase 1 Oxide project

Capital equipment pricing was based on project specific budget quotation requests (BQRs) for major mechanical and electrical equipment and database pricing for the balance, drawn from similar projects currently under construction or recently completed including the Phase 1 Oxide project.

The capital estimate was split into major disciplines (earthworks, concrete, steelwork, platework, tanks, mechanical equipment, mobile equipment, piping, electrical, instrumentation & control, mining, on-site and bulk off-site infrastructure, owner's costs, first fill and contractor indirect costs). Where possible, quantities were calculated, and budget rates were applied in order to produce the estimates. Otherwise, benchmark prices or typical factors of mechanical equipment prices were applied. In cases where the equipment was the same as Phase 1 (e.g. CIL agitators) updated quotations were obtained from those vendors. In other cases, such as the SAG mill, firm bids were obtained. The approximate percentages by value of each method are shown in Table 21.1.2 below.

Table 21.1.2 Basis of Pricing for Major Discipline Costs

Discipline	Allowance / Factored %	Historical / Estimated Pricing %	Budget Quote %
A General	-	100	-
B Earthworks	-	-	100
C Concrete	-	2	98
D Steelwork	-	2	98
E Platework	-	1	99
F Mechanical	2	14	84
G Piping	77	23	-
H Electrical	41	29	30
J Instrumentation & Control	-	100	-
M Buildings & Architectural	-	-	-

The process plant was also broken down into unit metallurgical operation areas with quantity take-offs generated based on similar facilities from previous projects to provide an acceptable level of confidence for a definitive estimate.

Unit rates were established for bulk commodities, materials and labour that were drawn from previous studies or construction projects within the region.

The rates used in the estimate have been reviewed and deemed to reflect the current market conditions.

Payment terms from similar projects or mentioned in the budget prices received were used to develop an approximate cash flow schedule for the project implementation capital expenditure.

The capital cost estimate is based on an engineering, procurement, and construction management (EPCM) implementation approach and horizontal (discipline based) construction contract packaging.

21.1.2 Estimate Basis

The capital cost estimate for the process plant was prepared in accordance with Lycopodium's standard estimating procedures and practices. The basis, source of information and methodology applied by Lycopodium are summarized in Table 21.1.2 and Table 21.1.3. Lycopodium compiled the estimate for the entire project but did not audit or review the non-process sections, which were prepared to a similar standard by other consultants.

Table 21.1.3 Capital Cost Estimate Basis

Description	Basis
Site	
Geographical Location	Client advice from Site Plan.
Maps and Surveys	Topographical information and surveys provided by Owner.
Geotechnical Data	Preliminary geotechnical report and geotechnical advice from Orezone provided during the DFS and used as basis for estimate allowances.
Process Definition	
Process Selection	Based on Flowsheets.
Design Criteria	DFS Standard (Based on PDC).
Plant Capacity	4.4 Mtpa.
Flowsheets	DFS.
P&IDs	Not Required.
Mass Balances	DFS.
Equipment List	DFS.
Process Facilities Design	
Equipment Selection	Selection based on duty and budget (or firm where appropriate) pricing provided by vendors.
General Arrangement Drawings	Fixed.
3D model	Preliminary to a level of detail suitable for DFS.

Description	Basis
Piping Drawings	Not required.
Electrical Drawings	DFS.
Specifications / Data Sheets	Preliminary for Budget Quotation Requests (BQRs).
Infrastructure Definition	
Existing Services	Provided by Orezone.
Design Basis	Fixed.
Layout	Fixed.
Bulk Earthworks	Volume estimated from the layout and available topography for bulk earthworks on all sites. Unit rates discussed in narrative sections below.
Detailed Excavation	Allowances for under pad excavation and backfill to prepare site for concrete works
Concrete Installation	Estimated from the layout and similar projects of comparable scale. Concrete (wet) supply rates and installation rates applied from project specific BQRs.
Structural Steel	Quantities estimated from the layout and similar projects of comparable scale. Supply and install rates applied from Lycopodium database.
Platework & Small Tanks	Quantities provided in the mechanical equipment list. Large item quantities estimated from reference projects. Smaller items compared to database. Supply and install rates applied from Lycopodium database.
Plant site roads, bulk earthworks, and site drainage plan	Volume estimated from the layout and available topography for bulk earthworks on all sites. Supply and install rates applied from Lycopodium database.
Tankage Field Erect	Quantities provided in the mechanical equipment list. Supply and install rates applied from project specific BQRs.
Mechanical Equipment	Quantities provided in the mechanical equipment list. Costs from responses to BQRs from reputable suppliers. Costs for low value items taken from the Lycopodium database.
Haul Roads	Excluded.
Mining Fleet	Excluded.
Power Station	Tie-in to 132kV/11kV site substation
Conveyors	Concrete & structural estimated from reference projects and layout. Mechanicals supply pricing from project specific BQRs and installation rates applied from Lycopodium database.
Plant Piping General	Factored from mechanical costs.
Overland Piping	Size and specification based on engineering selection. Quantity based on site layout. Rates based on recent market inquiries.
Electrical General	Quantities derived from engineering design and site layout. Materials pricing and installation costs from a combination of recent database information and responses to BQRs.

Description	Basis
Electrical HV	Quantities derived from engineering design and site layout. Materials pricing and installation costs from a combination of recent database information and responses to BQRs.
Commodity Rates – General	Appropriate rates from a combination of recent database information and responses to project specific BQRs.
Installation Rates – General	Appropriate rates from a combination of recent database information and responses to project specific BQRs based on preliminary contracting strategy.
Heavy Cranes	Requirements estimated based on lifting study or heavy lifts and likely duration.
Freight General	Combination of rates per freight tonne and factors.
Contractor Mobilization / Demobilization	Appropriate rates from a combination of recent database information and responses to project specific BQRs.
Fencing	Costed based on measured length and rate.
EPCM	Scope and deliverables-based estimate based on the EPCM controlled scope.
Vendor Representatives	Assessed by equipment package based on similar projects.
Owner's Costs	
Site Establishment	Requirements estimated using base rates.
Construction Facilities	Excluded.
Opening Stocks, First Fill Reagents and Consumables	Estimated from consumption rates and costs in operating cost estimate.
Spares	Based on Budgetary Quotes.
Owner's Project Team	Client estimate.
Project Insurances and Permits	Client estimate.
Sterilization Drilling	Already completed.
Plant preproduction expenses	Allowance.
Plant preproduction labour	4 weeks labour cost
Training	Allowance.
Owner's Team Expenses	Client estimate.
Duties and Taxes	Custom duties of 8.3% during operation phase to be considered to the cost estimate.
Escalation	Excluded.

21.1.3 Qualifications

The capital estimate is qualified by the following assumptions:

- The estimate base date regarding Project pricing is the second quarter 2023 (2Q23).

- All pricing received has been entered into the estimate using native currency. Prices of materials and equipment with an imported content will be converted to US$ at the rates of exchange stated in Table 21.1.4.

Table 21.1.4 Currency Exchange Rates

Currency	Exchange Rate
CAD	0.77
USD	1.00
EURO	1.09
AUD	0.69
ZAR	0.06
MAD	12
GBP	1.20

- Contractor rates and distributable include mobilization / demobilization, recurring costs, direct and indirect labour, construction equipment, construction crane (up to 100t), materials, materials handling and offloading, temporary storage, construction facilities, off site costs, insurances, flights, construction fuel, tools, consumables, meals, and PPE.

- The bulk commodities for earthworks that include imported material will be based on the assumption that suitable construction / fill materials will continue to be available from borrow pits within 2 km of the work fronts. Imported materials for concrete have been included in the concrete installation rates by the contractor.

- Engineering quantities for the tailings storage facility (TSF) have been provided by Orezone and appropriate rates applied to complete the capital cost of these items. Orezone sub consultant 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 is assumed available locally to the Project site.

- The estimate allows for all reinforced bar and mesh for construction to be free issued to contractor.

- There is no allowance for blasting in the bulk earthworks, which is consistent with the preliminary geotechnical information.

- The estimate allows for supply of structural steel and platework from Southeast Asia.

- Communications network and data for construction facilities to be free issued by the Owner.
- Owner's mobile equipment to be purchased early and made available for construction and operations 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 the owner.
- Import duties included in estimate.

The following items are specifically excluded from the capital cost estimate:

- Working capital (included directly in the financial model).
- Sustaining capital (included directly in the financial model).
- Financing costs or interest costs during construction.
- Schedule delays exceeding four weeks and associated costs.
- Scope changes.
- Further unidentified ground conditions.
- Extraordinary climatic events.
- Exchange rate variations.
- Force majeure.
- Labour disputes.
- Receipt of information beyond the control of EPCM contractors.
- Schedule recovery or acceleration.
- Research and exploration drilling.
- Salvage values.
- Project sunk costs.

- Project escalation.
- Community relations.

21.1.4 Contingency

The purpose of contingency is to make specific provision for uncertain elements of cost within the project scope and thereby reduce the risk of cost over-run to a predetermined acceptable level. Contingencies do not include allowances for scope changes, escalation, or exchange rate fluctuations.

Each line item in the capital cost estimate will have a contingency value allocated based on the apparent risk level. The assessment of the risk level will be based on the present engineering progress, information provided by vendors and contractors and database information.

Contingency is individually applied to quantities, unit rates and costs and compiled as a weighted average to form the overall recommended project contingency.

Estimate contingency has been developed with input from Orezone adapted to fit a brownfield mine and the data, information, and experience from recently completed Phase 1 Oxide plant.

21.1.5 Escalation and Foreign Exchange

Exchange Rates

All items in the capital cost estimate have been expressed in United States dollars (US$) and no allowances for exchange rate variations are included in the estimate.

21.1.6 Preproduction Costs

Preproduction costs that include preproduction labour and vendor representative costs have been included in the estimate.

21.2 Operating Cost Estimate

21.2.1 Summary

The operating costs have been compiled by Lycopodium based on costs developed by:

- AMC – Mining costs.
- Lycopodium - Processing costs.
- Orezone / Lycopodium – Site general and administration costs.

The estimate is considered to have an accuracy of +15 / -15%, is presented in US$ and based on information obtained during the second quarter of 2023 (Q2 2023).

The estimated Life-of-Mine (LOM) operating costs per tonne of ore treated and per ounce of gold produced are summarized in Table 21.2.1.

Table 21.2.1 Life of Mine Operating Costs per Tonne and per Gold Ounce (US$, 2Q 2023)

Cost Components	Total Cost $M	$/ Tonne Processed	$/oz Au
Mining	840.2	8.12	398
Processing	945.6	9.13	448
Site G&A	242.9	2.35	115
Refining and Bullion Transport	5.8	0.06	3
Government Royalties and Dev Tax	222.3	2.15	105
Total Cash Cost	**2,256.7**	**21.80**	**1,070**

21.2.2 Mining Operating Costs

Mining operating costs have been developed by AMC based on the schedules and strategy outlined in Section 16. The costs are based on contractor mining and vary annually depending on the location, depth and ore type of the pits mined.

The estimated annual mine operating costs have been summarized in Table 21.2.2

Table 21.2.2 Summary of Estimated Annual Mine Operating Costs

	LOM	9M 2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Annual Mining Costs (US$M)													
Grade control costs	41.5	2.6	3.3	4.6	4.3	3.8	4.1	4.3	4.0	3.7	3.9	2.6	0.3
Owner's Costs plus Contractor fee	120.8	8.0	10.6	11.2	11.2	11.2	11.2	11.2	11.2	11.2	11.2	11.2	1.9
Load & Haul	458.1	30.3	35.4	48.9	46.2	41.2	45.2	48.8	45.4	41.6	41.6	29.5	4.1
Drill & Blast	143.4	4.2	4.5	9.3	16.6	23.3	17.6	17.3	19.2	12.6	12.0	6.2	0.7
Stockpile Reclaim	29.3	0.6	0.3	0.0	3.2	7.1	2.7	0.6	2.2	0.8	2.2	4.8	4.7
Total Combined Mining Cost	**793.2**	**45.7**	**54.1**	**74.0**	**81.4**	**86.6**	**80.8**	**82.2**	**81.9**	**69.8**	**70.8**	**54.2**	**11.6**
Mining Quantities (Mt)													
Ore	95.7	5.9	8.8	8.8	9.1	6.4	8.8	11.0	10.1	9.8	9.0	7.2	0.9
Total Ore and Waste	283.2	16.5	20.7	30.0	30.0	28.5	28.4	29.8	28.6	25.2	26.1	17.2	2.3
Unit Mining Cost (US$/$t_{mined}$)													
Grade control costs	0.15	0.16	0.16	0.15	0.14	0.13	0.14	0.14	0.14	0.15	0.15	0.15	0.15
Owner's Costs plus Contractor fee	0.43	0.48	0.51	0.37	0.37	0.39	0.39	0.37	0.39	0.44	0.43	0.65	0.82
Load & Haul	1.62	1.84	1.71	1.63	1.54	1.45	1.59	1.64	1.59	1.65	1.59	1.71	1.80
Drill & Blast	0.51	0.25	0.22	0.31	0.55	0.82	0.62	0.58	0.67	0.50	0.46	0.36	0.31
Stockpile Reclaim	0.10	0.04	0.01	0.00	0.11	0.25	0.10	0.02	0.08	0.03	0.09	0.28	2.06
Total Unit Mining cost (US$/$t_{mined}$)	**2.80**	**2.77**	**2.62**	**2.47**	**2.71**	**3.04**	**2.84**	**2.76**	**2.86**	**2.78**	**2.71**	**3.15**	**5.15**

21.2.3 Processing Operating Costs

Processing operating costs have been developed by Lycopodium for a life of mine (LOM) blend ore. It is expected that the plant will operate on a range of mineralized material blends. The LOM processing costs are a weighted average of the various mineralized material type processing costs based on the LOM blend. Future design provisions have not been included in the operational cost estimate.

Processing operating costs have been developed for a plant with an annual throughput equivalent to 10,300,000 tonnes of ore, based on a 24 hour per day operation, 365 days per year. The existing Oxide Plant considers the 2023 operational production rate of 5.9 Mtpa of oxide ore at 80% passing (P_{80}) grind size of 125 microns (µm) (0.7 Mtpa increase from the nameplate capacity) and the new Hard Rock Plant will process 4.4 Mtpa of fresh mineralized material at a P_{80} grind size of 75 µm.

The processing operating costs include all direct costs to allow production of gold bullion. The battery limits for the processing operating costs are from the mineralized material delivered to the ROM bin to tailings discharged onto the tailings storage facility and up to gold bullion delivered to the plant goldroom safe.

The operating costs have been compiled from a variety of sources, including the following:

- Labour rates and manning as advised by Orezone.
- Grid power costs as advised by Orezone at a rate of 0.21 US$/kWh.
- Consumable prices from supplier budget quotations obtained in Q2 2023, Orezone advice on current pricing for the operation, and the Lycopodium database.
- Crushing and grinding consumable prices from supplier budget quotations, OMC modelled crushing and grinding energy, and consumables based on physical mineralized material characteristics determined from comminution testwork for the various mineralized material types.
- Reagent consumptions and gold extractions based on the results from a metallurgical testwork program.
- First principle estimates based on typical operating data / standard industry practice.
- All Oxide circuit costs have been provided by Orezone based on current operational data.

The processing operating cost estimate is summarized in Table 21.2.3.

Table 21.2.3 Process Operating Cost Summary
(US$, Q2 2023 +15 / -15%)

Cost Centre	Operating Cost		Operating Cost		Total Operating Cost		Proportion of Operating Cost
	Oxide Plant		Hard Rock Plant		Process Plants		
	US$ / year	US$ / tonne Ore	US$ / year	US$ / tonne Ore	US$ / year	US$ / tonne Ore	%
Operating Consumables							
Crushing Plant	849,815	0.14	159,133	0.04	1,008,948	0.10	1.0
Milling Plant	5,069,342	0.86	9,458,925	2.15	14,528,267	1.41	15.1
Trash Removal and Thickening	35,071	0.01	166,213	0.04	201,284	0.02	0.2
Pre-Oxidation and CIL	10,186,109	1.73	10,765,167	2.45	20,951,276	2.03	21.7
Acid Wash & Elution	2,762,008	0.47	2,307,394	0.52	5,069,402	0.49	5.3
Miscellaneous	734,654	0.12	420,103	0.10	1,154,757	0.11	1.2
Subtotal Consumables	**19,636,999**	**3.33**	**23,276,935**	**5.29**	**42,913,934**	**4.17**	**44.5**
Plant Maintenance	2,765,547	0.47	2,053,900	0.47	4,819,447	0.47	5.0
Laboratory (Plant)	618,850	0.10	260,605	0.06	879,455	0.09	0.9
Power	11,002,320	1.86	33,772,590	7.68	44,774,910	4.35	46.4
Labour (Plant Operations & Maintenance)	1,955,564	0.33	1,148,425	0.26	3,103,989	0.30	3.2
Subtotal	**16,342,281**	**2.77**	**37,235,520**	**8.46**	**53,577,801**	**5.20**	**55.5**
Total Process Cost	**35,979,280**	**6.10**	**60,512,455**	**13.75**	**96,491,735**	**9.37**	**100**

Qualifications

The operating cost estimate presented in this section excludes the following:

- ROM stockpile rehandling costs.
- Government monitoring / compliance costs.
- General and Administration (included in Section 21.2.4).
- Process / Tailings:

   - Tailings storage costs, including future lifts and rehabilitation
   - Tailings dust suppression costs
   - External government required monitoring and compliance costs
   - No cyanide detoxification costs.
- Environmental:
   - Any rehabilitation or closure costs.
- Labour:
   - Contract labour other than maintenance (generally self-performed by Orezone).
   - Overtime allowances.
- First fill / opening stocks are captured in the CAPEX.

The operating cost estimate includes the following:

- Import duties on consumable unit costs (in the consumables cost).
- Costs for the preparation and assaying of routine laboratory tests on the plant and site water samples (in the laboratory cost).
- Selected G&A costs (travel to site for international and regional expats, international expat recruiting / relocation and camp, catering, and cleaning) for administration and process plant personnel (in the G&A costs).

Power

The power cost estimate has been based on grid power at a unit cost of US$0.21/kWh. The average hard rock plant continuous power draw and power cost for the LOM blend by plant area is summarized in Table 21.2.4.

Table 21.2.4 Hard Rock Plant LOM Blend – Power Cost Summary (US$, Q2 2023, +15/-15%)

Plant Area	Connected Power kW	LOM Average Continuous Power Draw kW	LOM Power Cost US$/year
Area 125 – ROM & Reclaim	1,246	831	1,110,480
Area 135 – Milling and Classification	23,390	14,658	25,017,930
Area 145 – Trash Removal and Thickening	1,281	452	754,320
Area 165 – Leaching and CIL	1,567	1,112	1,794,240
Area 170 – Elution and Goldroom – Oxide Plant Area Expansion	128	55	2,253,900
Area 175 – Elution and Goldroom	183	81	79,800
Area 185 – Tails Handling	1,047	293	486,570
Area 210 – Reagents – Oxide Plant Area Expansion	12	2	630
Area 270 - Reagents	63	17	20,790
Area 220 – Water Services – Oxide Plant Area Expansion	80	30	50,400
Area 230 – Plant Water Services – Oxide Plant Area Expansion	39	26	4,830
Area 240 – Air Services – Oxide Plant Area Expansion	95	76	127,260
Area 245 – Air Services – Oxygen Plant	967	774	1,295,280
Area 330 – Water and Sewerage – Oxide Plant Area Expansion	414	248	150,780
Area 335 – Water and Sewerage	502	274	320,670
Area 355 – Tailings Storage Facility	276	182	304,710
Total	**31,290**	**19,111**	**33,772,590**

The power consumption for the crushers and ball mills has been calculated by OMC based on the physical mineralized material properties determined from comminution testing. The power consumption for the remainder of the individual plant mechanical equipment items has been calculated from the load list derived from the vendor supplied equipment and estimated based on the installed motor size of individual items of equipment, excluding standby equipment, adjusted by efficiency, load, and utilization factors to arrive at the annual average power draw. This total is then multiplied by total hours per annum and the electricity price to obtain the power cost.

Operating Consumables

Costs for the hard rock plant processing operating consumables, including reagents, liners, fuels, and process supplies have been estimated. The consumption of reagents and other consumables has been calculated from laboratory testwork and comminution circuit modelling at average mineralized material properties, calculated from first principles, or has been assumed based on experience with other operations. No additional allowance for process upset conditions and wastage of reagents has been made.

Reagent costs have been sourced from a combination of budget quotations, Client supplied costs, and in-house data relating to similar projects in the region. Transport and freight to site and import duties and taxes have been added based on vendor information. In absence of vendor information, a cost of 17.5% is added to transport, customs, and duties.

A diesel price, delivered to site, of US$1.60 per litre has been used, as provided by Orezone. Diesel will be used in plant mobile equipment, for the carbon elution heater circuit, the carbon regeneration kiln, and furnace. The diesel consumption for plant mobile equipment is based on industry standard vehicle consumption rates and estimated equipment utilisation, while the diesel usage for carbon elution and re-generation has been calculated from equipment anticipated running times and vendor data.

Allowances have been made for water treatment reagents. All costs associated with the Oxide circuit have been provided by Orezone based on current operations.

Maintenance

The plant maintenance cost allowance has been factored from the capital supply cost using factors from the Lycopodium database and is summarized for the LOM blend.

The allowance covers mechanical spares and wear parts, but excludes crushing and grinding wear components, grinding media, liners, and general consumables which are allowed for in the consumables cost.

The maintenance cost excludes payroll maintenance labour which is included in the labour cost. Contract labour has not been allowed for as mill liner changes and plant shutdowns is self-performed by Orezone.

Allowances for plant mobile equipment and general maintenance expenses have been made.

The mobile equipment allowance is based on unit costs for maintenance of the light vehicles, portable generators, and other mobile equipment for the process plant.

General maintenance expenses include specialist maintenance software, maintenance manuals, control system maintenance and licence fees.

Labour

The labour rates, manning levels and rosters used to determine the labour operating cost estimate have been agreed with Orezone and is an extension to the existing Oxide Plant operations.

The plant labour cost includes the additional Hard Rock Plant labour costs associated with site-based administration, plant operations and maintenance personnel. The plant labour cost excludes all mining personnel (included in the Mining cost category) and all support departments (included in the Site General & Administrative costs).

The site laboratory will be operated on a contract basis with the personnel included in the process labour count, but the labour costs included in the contract laboratory cost.

The estimate of the labour contingent has been based on a three-shift operation (two shifts working 12 hours per day, one rotation shift), to provide continuous coverage for the plant operation with allowance for leave and absenteeism coverage. Provision has been made in the manning numbers to accommodate annual and sick leave requirements.

The roster is based on all expatriate personnel working six weeks on site and three weeks off site with all other personnel working 10 days on site and four days off site with four weeks annual leave and two weeks sick leave per year.

Unit rates for labour have been based on Orezone advice and include the base salary excluding any overheads allowance. The overhead cost includes allowances for housing, travel, shift work, medical health insurance, life, and disability cover, and leave provisions. Camp and transportation costs for the workforce are excluded from the labour cost category.

Laboratory Costs

The existing contract assay laboratory will provide sample preparation and assay services for plant and environmental samples daily.

Laboratory costs have been based on Orezone input based on current operations. The laboratory cost allows for the supply of the laboratory equipment, mobilization, 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.

Services and Utilities

Mobile Equipment

Plant mobile equipment requirements have been agreed with Orezone. Mobile equipment costs provide for the fuel and maintenance of the mobile equipment fleet (excluding the mining fleet and mining light vehicles). The purchase cost of this equipment has been included in the capital cost estimate.

The fuel and maintenance costs for the mobile equipment are included in the consumables and maintenance cost centre, respectively.

An allowance has been made for a Front-End Loader (FEL) to be routinely used in the plant to feed stockpiled mineralized material into the reclaim bin when the primary crushing circuit is off-line, as required.

Water Supply

Water supply costs are not included in the water section as it incurs only a power and maintenance cost which is covered by the respective section. Water treatment for filtered water at a unit cost of US$0.10/kl, as well as potable water at a unit cost of US$0.11/kl is considered in the consumable portion of the operating costs.

First Fill Reagents and Opening Stocks

Costs have been allowed in the project capital cost estimate to purchase the mill grinding media and activated carbon needed to commission the plant. Other consumables and reagents required for the process plant first fill and opening stocks are expected to be paid for in the commissioning and ramp up period, from the operating cost budget.

Quantities allowed have been based on either consumption over the minimum period or minimum shipping quantities, considering package size.

Vendor Representatives

This cost has been included in the capital cost estimate and allows for specialist vendor representatives to oversee commissioning of their processing equipment and include allowances for labour, airfares, and expenses.

Training

Training allowances have been included in the capital cost estimate and cover the costs of providing pre-production training for process plant operations and maintenance staff, but not their salaries as these are covered in the pre-production labour costs.

Working Capital

Working capital covers the cost of operating the process plant before the first receipt of revenue from bullion sales. Working capital calculations are included in Orezone's project cashflow analysis but are not included in either the capital or operating costs estimates.

21.2.4 Site General and Administrative Costs

The Site G&A cost estimate is based on Orezone's experience of operating in Burkina Faso for several years, gazetted rates for land tax and similar costs, rates, and quotations from reputable service providers such as Orezone's current catering contractor and insurance providers, and a build-up of the expected G&A organization chart from first principles. G&A costs were estimated to average US$21.3M annually, or approximately US$2.35/t ore over the LOM, not including preproduction.

These costs include insurance, permitting, office supplies, general management, accounting, communications, informational technology, environmental and social management, human resources, purchasing and procurement, health and safety, security, international travel, and camp operating costs. In most cases, these services represent fixed costs for the site with some exceptions such as camp operations and transportation costs of employees.

The Site G&A costs exclude certain costs such as the transport and refining of gold and royalty payments which are included in the economic analysis. Environmental rehabilitation costs, which are treated as separate line items in the financial model, are included in the sustaining capital.

The annual Site G&A costs have been summarized in Table 21.2.5.

Table 21.2.5 Summary of Site G&A Costs

Department	LOM	9M 2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
General Administration	10.37	0.70	0.93	0.93	0.93	0.93	0.93	0.93	0.93	0.93	0.93	0.93	0.35
Project Management	5.78	1.60	0.00	0.00	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.19
Guest House (Ouaga)	1.45	0.10	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.05
Camp	18.93	1.23	1.64	1.64	1.72	1.72	1.72	1.72	1.72	1.72	1.72	1.72	0.65
Community Relations / CSR	9.13	0.62	0.82	0.82	0.82	0.82	0.82	0.82	0.82	0.82	0.82	0.82	0.31
Finance	9.38	0.63	0.84	0.84	0.84	0.84	0.84	0.84	0.84	0.84	0.84	0.84	0.32
Health & Safety	19.19	1.25	1.66	1.66	1.75	1.75	1.75	1.75	1.75	1.75	1.75	1.75	0.66
Environment	12.31	0.83	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	1.11	0.42
Human Resources	6.22	0.42	0.56	0.56	0.56	0.56	0.56	0.56	0.56	0.56	0.56	0.56	0.21
Information Technology	12.49	0.84	1.12	1.12	1.12	1.12	1.12	1.12	1.12	1.12	1.12	1.12	0.42
Supply Chain Management	12.04	0.81	1.08	1.08	1.08	1.08	1.08	1.08	1.08	1.08	1.08	1.08	0.41
Security	66.13	4.14	5.53	5.53	6.08	6.08	6.08	6.08	6.08	6.08	6.08	6.08	2.30
Insurance	36.16	2.01	2.68	2.68	3.43	3.43	3.43	3.43	3.43	3.43	3.43	3.43	1.30
Permits and Taxes	14.92	0.99	1.06	1.06	1.16	1.44	1.44	1.44	1.44	1.44	1.44	1.44	0.55
Studies	0.11	0.11	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
G&A Labour bonus	8.28	0.78	0.64	0.64	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.28
G&A Total (US$M)	**242.87**	**17.05**	**19.81**	**19.81**	**21.97**	**22.26**	**22.26**	**22.26**	**22.26**	**22.26**	**22.26**	**22.26**	**8.41**
G&A costs per tonne processed	**2.35**	**3.97**	**3.36**	**2.82**	**2.13**	**2.16**	**2.16**	**2.16**	**2.16**	**2.16**	**2.16**	**2.16**	**2.16**

21.3 Growth Capital, Sustaining Capital, and Closure Costs

Growth capital includes the grid power connection project that will be completed in Q4-2023, RAP Phases II & III, that are currently underway and will be completed in 2024, and RAP Phase IV that will be performed progressively over 2024 through to 2027. Sustaining capital costs include ongoing tailings storage facility raises, haul road extensions, grade control drills, and mine dewatering and surface water management equipment. Closure cost includes the remediation work required to return the site to meet all conditions of the Environmental and Social Impact Assessment. Growth Capital, Sustaining Capital and Closure Costs are summarized in Table 21.3.1

Table 21.3.1 Growth Capital, Sustaining Capital, and Closure Costs ($M)

Description	Total Costs	9M 2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Growth Capital														
Grid Power	16.3	16.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
RAP Phase 2 & 3	23.0	10.4	12.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
RAP Phase 4	18.4	0.0	10.2	3.6	4.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Growth Capital total	**57.7**	**26.7**	**22.8**	**3.6**	**4.7**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**
Sustaining Capital														
Plant	2.1	2.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Infrastructure	87.0	2.8	4.2	13.0	16.5	7.5	8.6	7.3	7.5	8.1	8.6	2.8	0.0	0.0
Mining	8.4	1.1	2.7	2.7	1.3	0.0	0.0	0.1	0.1	0.1	0.2	0.1	0.0	0.0
G&A	3.6	2.1	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.0	0.0
Sustaining Capital total	**101.0**	**8.1**	**7.0**	**15.9**	**17.9**	**7.7**	**8.8**	**7.5**	**7.7**	**8.4**	**8.9**	**3.1**	**0.0**	**0.0**
Closure Costs														
Reclamation and Closure	19.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	19.1	0.0
Salvage Value	-9.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-9.9	0.0
Total Closure Cost	**9.3**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**9.3**	**0.0**
Total Growth, Sustaining & Closure Cost	**168.0**	**34.8**	**29.8**	**19.4**	**22.6**	**7.7**	**8.8**	**7.5**	**7.7**	**8.4**	**8.9**	**3.1**	**9.3**	**0.0**

22.0 ECONOMIC ANALYSIS

22.1 Introduction

An economic assessment of the Bomboré Gold Project (Project) including the current scope of the Phase II Hard Rock expansion has been conducted using a pre-tax and after-tax cash flow model prepared by Lycopodium on behalf of Orezone.

Input data were provided from a variety of sources, including the various consultants' contributions to this Technical Report, pricing obtained from external suppliers and contractors, and exchange rates and Project specific financial data such as the expected Project taxation regime received from Orezone. The assessment was based upon:

- Capital cost estimates and expenditure schedules prepared by Lycopodium with input from Orezone.
- Mine schedule and mining operation cost estimates based on contract mining operations, as developed by AMC with input from Orezone.
- Process operating and site general and administrative cost estimates prepared by Lycopodium, with assistance from Orezone.
- Metallurgical performance characterized by testwork completed for the study.
- Sustaining capital, growth capital, closure and salvage cost estimates were developed by Orezone and reviewed by Lycopodium.
- Royalty, tax, discount rates and other model inputs provided by Orezone.
- The cash flow analysis excludes any effects due to inflation and all dollars are expressed in real US$ as at Q3 2023.
- Base case gold price of $1,900/oz for the nine month period of 2023 and $1,750/oz for the remaining life of mine were agreed with Orezone.
- The cash flow analysis is based on full equity funding and any cost of borrowing is excluded.
- The existing Oxide plant will process the Oxide and Upper Transition ores at a nominal rate of 5.9 Mtpa, which is 0.7 Mtpa above the nameplate capacity (5.2 Mtpa nameplate) as reported by the current Bombore plant operations. A new 4.4 Mtpa Hard Rock plant will process the Lower Transition and Fresh ores.

- A Net Present Value (NPV) is calculated using a 5% per annum discount rate.

22.2 Assessment Results

Following are key results of the assessment:

- After-tax NPV5% of $636M.
- Mine life of 11.3 years with gold production totalling 2.11Moz.
- Average annual gold production of 231,000oz in the first three full years after expansion at an AISC of $1,081/oz.
- Average LOM gold production of 186,000oz/yr over 11.3 years at an AISC of $1,122/oz.

Table 22.2.1 presents a summary of the annual gold production information on which the cash flow model is based.

Table 22.2.2 shows the Project cash flow summary. At a discount rate of 5%, the after-tax NPV is estimated at $636 M. The Project economics are summarized in Table 22.2.3.

Table 22.2.1 LOM and Annual Production

	Units	LOM	Annual											
			9M 2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Oxide Production														
Mill Feed	Mt	64.6	4.3	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	5.9	1.3
Grade, Au	g/t	0.54	0.78	0.74	0.71	0.53	0.46	0.45	0.47	0.44	0.51	0.47	0.49	0.37
Recovery	%	90.9	90.8	91.1	90.8	91.0	91.4	90.8	90.3	91.0	90.9	91.0	90.8	91.1
Oxide Gold Production	**koz**	**1,020**	**98**	**128**	**123**	**92**	**79**	**77**	**81**	**75**	**88**	**81**	**84**	**14**
Hard Rock Production														
Mill Feed	Mt	38.9			1.1	4.4	4.4	4.4	4.4	4.4	4.4	4.4	4.4	2.6
Grade, Au	g/t	1.02			1.21	1.33	1.14	1.12	1.02	1.08	1.08	0.95	0.70	0.54
Recovery	%	85.0			85.7	87.6	86.8	87.5	83.6	82.3	83.0	84.0	84.1	84.8
Hard Rock Gold Production	**koz**	**1,089**			**38**	**165**	**140**	**139**	**121**	**126**	**127**	**113**	**83**	**38**
Combined Production														
Mill Feed	Mt	103.5	4.3	5.9	7.0	10.3	10.3	10.3	10.3	10.3	10.3	10.3	10.3	3.9
Grade, Au	g/t	0.72	0.78	0.74	0.79	0.87	0.75	0.73	0.71	0.71	0.75	0.68	0.58	0.48
Recovery	%	87.8	90.8	91.1	89.5	88.8	88.4	88.7	86.1	85.4	86.1	86.8	87.3	86.4
Combined Gold Production	**koz**	**2,109**	**98**	**128**	**161**	**257**	**219**	**216**	**201**	**202**	**215**	**194**	**167**	**52**

Table 22.2.2 Cash Flow After Tax (LOM Summary – Combined Oxide and Hard Rock)

	$M	$/Ore Processed	$/oz Au
Revenue	$3,704.4	$35.78	$1,756.1
Mining Cost	$793.2	$7.66	$376.0
Processing Cost	$945.6	$9.13	$448.3
G&A Cost	$242.9	$2.35	$115.1
Refining & Transport Costs	$5.80	$0.06	$2.7
Government Royalties	$222.3	$2.15	$105.4
Cost of Opening Stockpiles	$47.1	$0.45	$22.3
Total Cash Cost	$2,256.7	$21.80	$1,069.8
EBITDA	$1,447.6	$13.98	$686.3
Phase II Capital	$167.5	$1.62	$79.4
Growth Capital	$57.7	$0.56	$27.4
Sustaining Capital	$101.0	$0.98	$47.9
Rehabilitation & Closure (net of salvage)	$9.3	$0.09	$4.4
Total Capital Costs	$335.6	$3.24	$159.1
Working Capital Movement (incl. stockpiles)	-$31.1	-$0.30	-$14.8
Cash Flow before tax	$1,143.2	$11.04	$541.9
Corporate Income Tax	$258.0	$2.49	$122.3
Cash Flow after tax	$885.2	$8.55	$419.6

Table 22.2.3 Financial Summary

	Value
LOM gold production	2.11 Moz
Revenue from Gold (99.95% Payable)	$3,704M
Operating Costs	$2,257M
Phase II Capital	$167.5M
Growth Capital	$57.7M
Sustaining Capital and Closure Costs	$110.3M
Pre-Tax Cash Flow	$1,143M
After-Tax Cash Flow	$885M
After-Tax economics:	
NPV (5%)	$636M

22.3 Sensitivity Analysis

The Project NPV was assessed by undertaking sensitivity analyses on the gold price, gold recoveries, operating costs, and capital costs. The Project is most sensitive to changes in the gold price and then operating costs. The results of all pre-tax sensitivity analyses are presented in Table 22.3.1and in Figure 22.3.1. The results of all after-tax sensitivity analyses are presented in Table 22.3.2 and in Figure 22.3.2.

Table 22.3.1 NPV Sensitivity Analysis (Pre-Tax)

	Lower	Base Case	Higher
Gold Price (±10%)	$593,133,894	$844,173,483	$1,095,295,069
CAPEX (±10%)	$870,101,976	$844,173,483	$818,244,991
OPEX (±10%)	$986,763,159	$844,173,483	$701,637,375
Recovery (±2%)	$790,593,007	$844,173,483	$897,753,959

Figure 22.3.1 NPV Sensitivity Analysis (Pre-Tax)

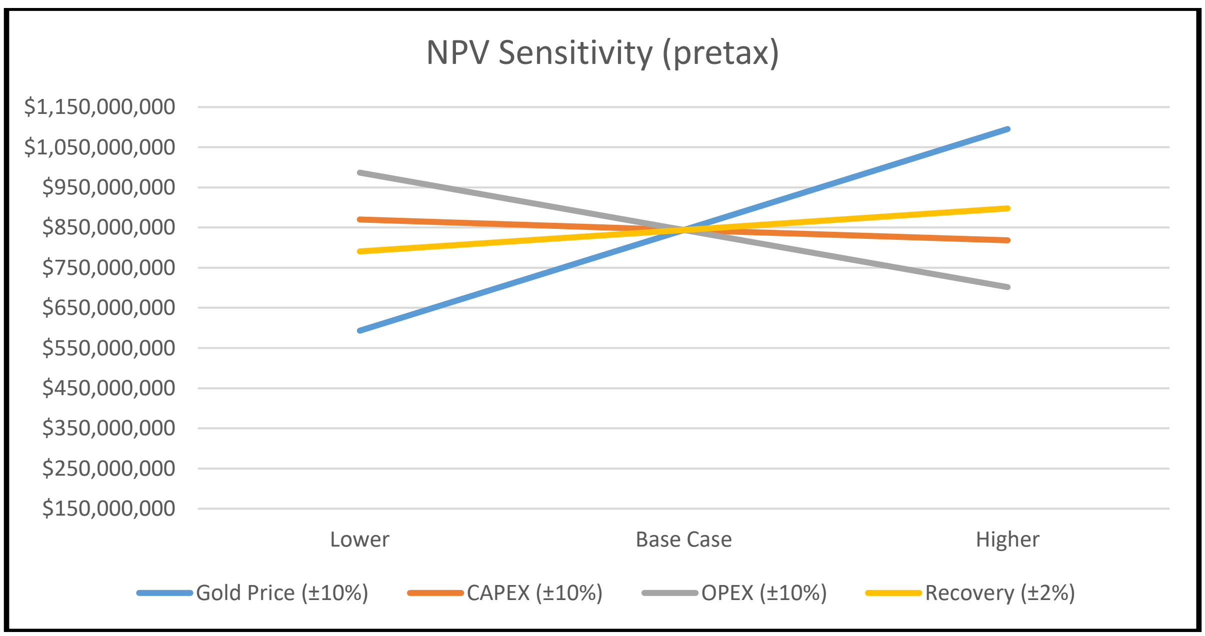
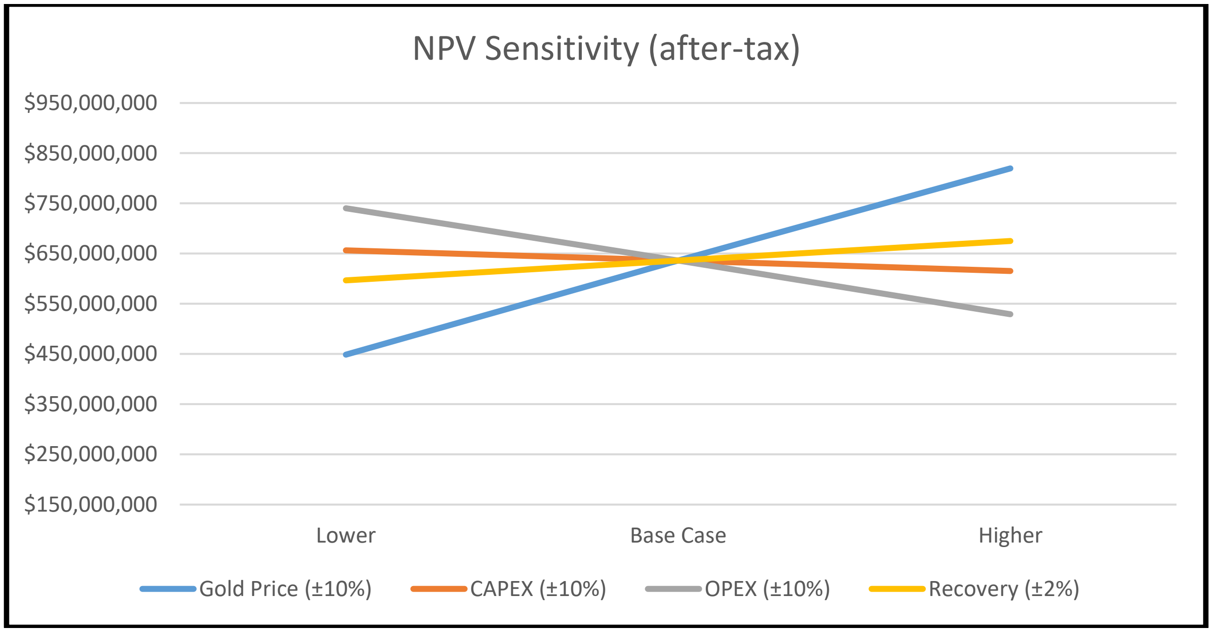


Table 22.3.2 NPV Sensitivity Analysis (After-Tax)

	Lower	Base Case	Higher
Gold Price (±10%)	$448,677,694	$635,896,925	$819,754,284
CAPEX (±10%)	$656,468,995	$635,896,925	$615,324,854
OPEX (±10%)	$740,366,174	$635,896,925	$529,231,420
Recovery (±2%)	$596,687,432	$635,896,925	$675,106,417

Figure 22.3.2 NPV Sensitivity Analysis (After-Tax)



On 27 October 2023, the President of Burkina Faso signed a decree to increase royalty rates on gold sales. The decree increases the royalty from the previous 5.0% on all gold sales at or above $1,500 per ounce to a new rate of 6.0% on gold sales at or above $1,500 and under $1,700 per ounce, 6.5% on sales at or above $1,700 and under $2,000 per ounce and has been capped at 7.0% for gold sales at or above $2,000 per ounce. Certain legislative procedural matters are required before the new royalty rates become law and as of the date of filing this Technical Report, these had not yet occurred. Although these new rates will not have a material impact on the cash flow model, readers are cautioned that the new royalty rates have not been included in the economic analysis.

Assuming the royalty increase is officially adopted into law at the beginning of 2024, the after-tax NPV of the Project would be reduced from $636M to $607M.

23.0 ADJACENT PROPERTIES

The author of this Report section has been unable to verify the information presented below and cautions that the mineralization on adjacent properties is not necessarily indicative of the mineralization on the Bomboré Property that is the subject of this Feasibility Study Report.

23.1 West African Resources Sanbrado Gold Mine

West African Resources (WAF) Sanbrado Mine is an operating gold mine located approximately 10 km southeast of the Bomboré Property. Sanbrado commenced commercial production on 1 May 2020. According to their website www.westafricanresources.com, WAF's unhedged 2023 gold production guidance for Sanbrado is 210,000 to 230,000 oz at an all-in sustaining cost of less than US$1,175/oz.

The Sanbrado Mine consists of several open pits within 1 to 2 km of the processing plant site and an underground mine accessed through a box-cut and portal immediately to the southwest of the M1 South Open Pit. The processing plant consists of a conventional SABC milling circuit, gravity, and carbon in-leach processing with a nominal throughput capacity of 2.5 Mtpa. In April 2023, WAF reported (as of December 31, 2022) Measured Mineral Resources of 5.2 Mt at 2.9 g/t Au for 0.48 Moz gold, Indicated Mineral Resources of 31.9 Mt at 1.7 g/t Au for 1.70 Moz gold, Inferred Mineral Resources of 18.5 Mt at 1.9 g/t Au for 1.15 Moz gold, and Mineral Reserves of 17.1 Mt at 2.3 g/t Au for 1.29 Moz gold.

23.2 West African Resources Toega Gold Project

In April 2020, WAF announced the acquisition of the Toega Gold Deposit (Toega) from B2Gold Corp (B2Gold). Toega is located 7 km to the south of the Bomboré Property. The Deposit has a strike length of >1,200 m, is up to 400 m wide, and has been traced in drilling to a depth of 400 m below surface. In April 2023, WAF reported (as of 31 December 2022) Indicated Mineral Resources of 13.1 Mt at 1.7 g/t Au for 0.70 Moz gold, Inferred Mineral Resources of 8.4 Mt at 2.1 g/t Au for 0.57 Moz gold, and Mineral Reserves of 9.5 Mt at 1.9 g/t Au for 0.57 Moz gold.

According the WAF's April 5, 2023, 10-year production plan, Toega is included in the Sanbrado Mine's production profile from 2026.

23.3 West African Resources Kiaka Gold Project

In November 2021, WAF announced the completion of the acquisition of the Kiaka Gold Project from B2 Gold and partner GAMS-Mining F&I Ltd. Kiaka is located 45 km south of Sanbrado Mine. Kiaka is a large-scale, permitted project at the construction stage that is planned to commence production in 2025. In April 2023, WAF reported as of December 31, 2022, Indicated Mineral resources of 211.5 Mt at 0.9 g/t Au for 5.93 Moz gold, Inferred Mineral Resources of 67.7 Mt at 0.8 g/t Au for 1.80 Moz gold and Mineral Reserves of 154.7 Mt at 0.9 g/t Au for 4.51 Moz gold.

23.4 West African Resources MV3 Gold Project

In April 2023, WAF announced a shallow open-pit maiden gold resource at MV3, consisting of Indicated Mineral resources of 1.6 Mt at 2.2 g/t Au for 0.11 Moz gold and Inferred Mineral Resources of 1.9 Mt at 2.4 g/t Au for 0.14 Moz gold. MV3 is located approximately 2 km to the east of the Bomboré Property. The Deposit has a strike length of 800 m and has been traced in drilling to an average depth of 70 m below surface. According to their website www.westafricanresources.com, further drilling to test mineralization at depth and along strike is planned to be completed at MV3 in 2023. WAF also intends to undertake Feasibility Studies in parallel with extensional drilling, with the aim of fast-tracking MV3 into production by 2024.

23.5 Arrow Minerals Limited Boulsa Gold Project

Land tenements related to Arrow Minerals Limited (ASX: AMD) Boulsa Gold Project are located approximately 30 km northeast from the Bomboré Property. Multiple mineralized prospects have been identified within the Boulsa tenements and are at various stages of early-stage exploration.

According to their website www.arrowminerals.com.au, the most recent work completed by AMD on the Boulsa Property was a regional stream sediment sampling survey. Analysis of the survey work is ongoing.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Implementation and Schedule

The expansion from the current 5.9 Mtpa to 10.3 Mtpa is expected to be complete in September 2025. Mining and processing of oxide material in the 5.9 Mtpa Oxide Plant will continue during the construction of the new 4.4 Mtpa Hard Rock Plant.

The implementation strategy for the Expansion Project will be to award an EPCM contract to complete the detail engineering, procurement, and project construction management for installation of the process facilities. Orezone will continue to self-manage the construction for the grid power connection, community resettlements, road upgrades, camp upgrade, tailings lifts and other similar activities.

The fabrication and installation of the SAG mill is the critical path activity. Mill vendors have advised that the duration of the SAG mill and motor fabrication will be approximately 52 weeks. The priority is to place an award for the SAG mill in October 2023 to be followed by procurement of other long lead items such as the jaw crusher, thickener, apron feeders and liner handler. Basic engineering (Front End Engineering and Design or FEED) will commence in Q4 2023 and continue into Q1 2024 with selection of all major equipment and procurement of engineering drawings to advance concrete and steel design.

The connection to grid power is scheduled to be complete before the end of 2023. SONABEL, Burkina Faso's state-owned electricity company, has approved the required drawings and designs for the powerline and substations. As of November 2023, land compensation for the transmission line corridor is complete and construction of the transmission towers, substation and switching station are well advanced.

Raw water is currently sourced from the seasonal Nobsin River and diverted by a permanent weir into an existing 5.2 million cubic metre reservoir. A second 1.8 million cubic metre reservoir is planned to store sufficient water for the expansion. A pit in the P8P9 orebody has been selected for early excavation starting in 2024 and completion in Q1 2025 to store the additional raw water.

The Resettlement Action Plan (RAP) Phases II and III are underway and includes the construction of three new resettlement communities (MV3, MV2, and BV2). Orezone has sequenced MV3 as the first community to construct in order to gain access to mining areas that are currently contemplated in the 2024 mine plan. A RAP Phase IV is also planned to accommodate an expanded property footprint, as well as areas outside the property to resettle households in the footprints of future community reservoirs. The combined RAP Phases II, III and IV will be completed progressively over 2023 to 2028.

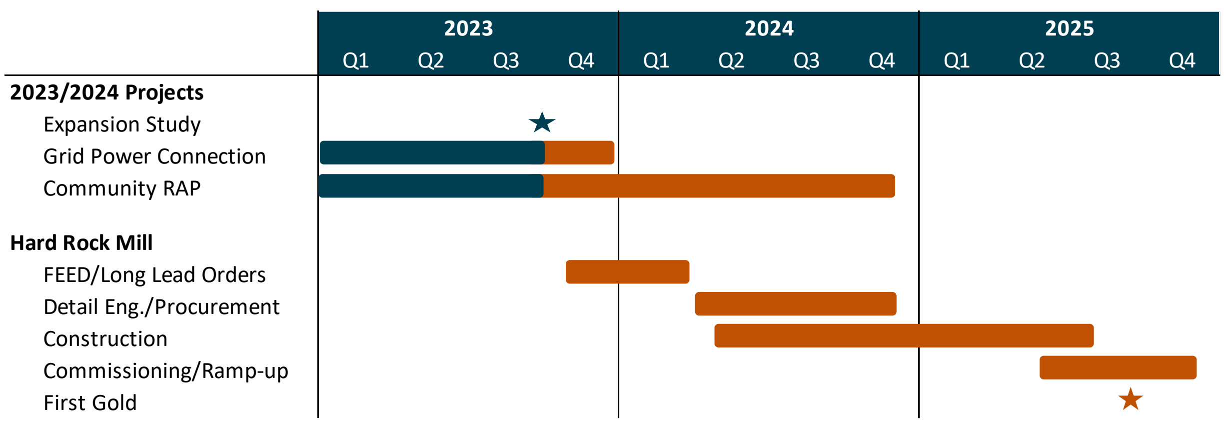
A Phase IV RAP study and an ESIA study related to the property expansion are well advanced and will be submitted to the authorities in December 2023, with approvals anticipated in Q1 2024.

The existing lined tailings storage facility is designed to be raised in stages over the mine life with downstream embankment construction techniques using run-of-mine waste rock as is current practice. The tailings storage facility footprint will be expanded with a second cell to provide 128Mt total tailings storage capacity. The earthworks for cell 2 construction will commence in Q3 2024 at the end of the rainy season.

The Phase II Expansion construction activities are expected to commence at site in Q1 2024 with the bulk earthworks. Mobilisation of the concrete contractor and installation of the concrete batch plant will be the priority in Q2 2024. Construction of the welded CIL tanks is expected to commence in Q3 2024, and SAG mill components will start to arrive at site towards the end of 2024. Construction works in the plant area are anticipated to be substantially complete by the end of Q2 2025, leading to commissioning in Q3 2025 and the first gold pour by the end of Q3 2025.

A summary level schedule is provided in Figure 24.1.1.

Figure 24.1.1 Construction Schedule for the 4.4 Mtpa Hard Rock Plant



IMPORTANT NOTICE

Please also see the forward-looking information Important Notice at the end of Section 22.

25.0 INTERPRETATIONS AND CONCLUSIONS

25.1 Summary

Based on the work undertaken to date, as summarized in this Technical Report and the conclusions listed below from the individual Authors, the Bomboré Project is a viable development opportunity centred around the initial mining and processing of the oxide and upper transition zones of the mineralized material on the Bomboré tenements followed by the supplemental mining and processing of higher-grade lower transition and fresh rock material after the staged Phase II expansion to the processing plant.

25.2 Geology and Mineral Resources

The Bomboré Gold Deposit is a large, structurally-controlled, orogenic gold deposit similar to deposits found elsewhere in late Proterozoic Birimian terranes of West Africa and globally.

Drilling has outlined mineralization with three-dimensional continuity, and size and grades that can potentially be extracted economically. Orezone's protocols for drilling, sampling, analysis, security, and database management meet industry standard practices. The drill hole database was verified to be suitable for Mineral Resource estimation work.

The 2023 Mineral Resource estimation work is reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (the 2014 CIM Definition Standards). At a cut-off grade of 0.25 g/t Au for oxide and transition material and 0.45 g/t Au for hard-rock material, Measured + Indicated Mineral Resources are estimated to total 179.3 Mt at an average grade of 0.78 g/t Au for 4.5 Moz of contained gold. At the same cut-off grades, Inferred Mineral Resources are estimated to total 20.0 Mt at an average grade of 0.95 g/t Au for 0.6 Moz of contained gold.

Many mineralized drill hole intervals, though included within the database, remain beyond the limits of the mineralization wireframes. No tonnage or grade estimates were provided for these intersections at this stage.

Potential risks relating to confidence in the grade, vein thickness and corresponding volume of material above a cut-off grade that could influence the Mineral Resource Estimate have been identified. Confidence in the grade, vein thickness and corresponding volume of material above a cut-off grade is influenced by several factors, including the distance between drill holes, the direction of mineralized continuity between samples, and proximity to high-grade shoots within the vein.

Lower confidence is therefore associated with widely-spaced drilling and higher confidence is associated with closer-spaced drilling. The actual drill sample spacing varies by vein and the classification of the Mineral Resource Estimate was assigned based on the level of confidence from drill hole sample spacing.

Risk is associated with all classifications of Mineral Resource Estimate, most particularly with an Inferred Mineral Resource Estimate.

- Wireframes - There is a risk that the Mineral Resource Estimate wireframes (greater than MRE cut-off) may be moderately high-biased with respect to the representative volume, and subsequent estimated tonnage and metal content. This potential bias could be where the wireframes extend somewhat too far into lower-grade (less than MRE cut-off) assay areas of influence.

- Accuracy of Localized High-Grade Samples - Localized high-grade samples when encountered in drill core sampling are part of the mineralization system. The raw assay grade can behave as nuggety gold mineralization, which can be described from the observed coefficients of variation. Locally, this represents a risk in the accuracy of grade estimation for Mineral Resource and subsequent Mineral Reserve estimation.

- Grade Capping - Where only widely-spaced sampling is available, the spatial extent of the high-grade mineralization may be uncertain. The selection of grade estimation methods to extrapolate high-grade samples can strongly influence regionalized vein grades and volumes. Grade capping was implemented to manage the local influence of individual high-grade samples. However, there is a risk that the grade capping methodology used may be too liberal during its attempt to preserve local grade variability. This risk can be reduced through future close-range sampling to delineate high-grade shoots within the vein systems, thereby allowing the highest-grade material to be separately sub-domained to constrain spatial influence of these samples within delineated shoots. Closer-spaced, pre-production definition drilling should be implemented to mitigate excessive extrapolation of high-grade values and to inform the local, short-range, grade variability.

- Mineral Resource Modelling Methodology - During the course of constructing the seven block models that comprise the total MRE, two Mineral Resource modellers (one internal and one external) and two different software packages were utilized along with other individuals manipulating data and optimizing pit shells. This level of varied involvement has the risk of introducing inconsistencies in Mineral Resource modelling approach and therefore the final MRE numbers may contain over or under estimation that is not easily recognizable or auditable. Future MREs should be conducted either entirely in-house or entirely by an external consultant.

Potential opportunities have been identified for expansion and increasing confidence of existing Mineral Resources, in addition to current exploration. The upgrading of Inferred Mineral Resources to Indicated Mineral Resources offers an opportunity for potential immediate Mineral Reserve expansion. Brownfields along strike and down-dip extension exploration offers a secondary opportunity to initially define additional Inferred Mineral Resources that would be subsequently converted to Indicated Mineral Resources. The use of high-grade restrictive search ellipse parameters may present an opportunity to allow higher-grade capping and result in higher-grade in a restricted volume.

- Upgrading Inferred Mineral Resources - The upgrading of Inferred Mineral Resources to Indicated Mineral Resources is the most obvious opportunity to increase Mineral Resources for subsequent conversion to Mineral Reserves. Many of the Mineral Reserve optimized pit shells will bottom out at the depth limit of Indicated Mineral Resources and could easily go deeper with more Indicated Resources. Since the conversion of Inferred Mineral Resources to Indicated Mineral Resources is an in-fill operation within existing wireframes, and not a discovery operation, there is a high probability of success.

- Delineating Additional Inferred Mineral Resources - The delineating of additional Inferred Mineral Resources is an opportunity that requires more effort than in-fill drilling, however, it presents the likelihood of shallow, along strike Mineral Resource expansion at potentially lower future Mineral Reserve strip ratios. Additionally, drilling down-dip extensions beyond known wireframes is an opportunity to expand Mineral Resources at depth and result in a larger open pit excavation and subsequent larger Mineral Reserves.

- High-Grade Restrictive Search Ellipse Parameters - The use of high grade restrictive search ellipse parameters along with more liberal capping values is an opportunity to delineate high-grade shoots without sub-domaining inside existing wireframes. This type of grade interpolation strategy would allow the preservation of local high grade variability without undue influence over nearby lower grade values.

- Exploration Opportunities - Potential opportunities have been identified for increasing the existing Mineral Resources, through expansion of current Inferred Mineral Resources and generation of future Inferred Mineral Resources. There is potential for Mineral Resource expansion drilling programs at the P17 deposit in 2024. Initial Mineral Resource estimation for the Kiin Tanga and P13 satellite deposits (at the north and south ends of the Bomboré gold mineralization trend) in 2024 should identify expansion opportunities along strike and at depth. Planned resistivity surveys at P13 could identify priority opportunities for drill testing that may support future Mineral Resource estimation.

25.3 Mining

The Mineral Reserve Estimate is based on the updated Mineral Resource Estimate (MRE) prepared by P&E and Orezone with an effective date of 28 March 2023. The Bomboré Mineral Reserves are estimated to contain 103.5 Mt at a grade of 0.72 g/t Au containing 2,403 koz Au. Mineral Reserves are composed of open pit Mineral Reserves of 95.7 Mt at an average grade of 0.75 g/t Au containing 2,301 koz Au and oxide stockpiles of 7.9 Mt at an average grade of 0.40 g/t Au containing 102 koz Au.

This Technical Report considered all available Measured and Indicated material in the MRE within the oxide and hard rock horizons. Inferred Mineral Resources were treated as waste. Orezone developed diluted mine models by re-blocking the resource block models.

AMC conducted pit optimizations on the re-blocked block models using Gemcom's Whittle™ 4.X software (Whittle). The pit optimization included both oxide and hard rock horizons, with inputs varied depending on the respective cost and processing characteristics and haulage profiles. A gold price of US$ 1,500/oz and associated off-site charges were provided by Orezone in calculating the Mineral Reserve Estimate. Royalties are applied to the totality of the gold produced. The pit optimization RF1 shells were then used as the basis for producing practical mine designs. The mine block models were evaluated against the mine designs with appropriate modifying factors, to estimate the Mineral Reserves.

The Bomboré mine has been in commercial production since December 2022 and will be further developed as an open pit operation mining oxide and hard rock material from over 70 separate pits of variable size and depth across a mineralized zone approximately 14 km long and 3 km wide. Mining of ore and waste is conducted by contractors with an owner's team responsible for site management, grade control, and mine planning activities. Mining of oxides is currently undertaken with 50 to 80 t diesel hydraulic excavators equipped with 3 to 5 m^3 buckets. Similar equipment will be employed in the hard rock.

The haulage requirements for oxide and hard rock material have been estimated based on rigid frame highway trucks with 26 t payload as currently deployed in the mining operations. Orezone is considering the application of trucks with higher payload of 30-60 t as part of the hard rock expansion.

An optimal mine schedule was developed using Minemax software. Pits were sequenced in order of value within assorted constraints such as wet seasons, access, TSF construction and plant ramp up. Mining rate was smoothed, and descent rates controlled. The target feed throughputs of both oxide and fresh ore were achieved. Equipment numbers peak at 18 excavators and 132 trucks employing approximately 1,100 contractors and 250 owner's team personnel at the mine (excluding plant personnel).

The key project life-of-mine (LOM) highlights are:

- 283.2 Mt total material mined:
- 103.5 Mt of ore:
  - 95.7 Mt of ore at 0.75 g/t Au mined and processed, including 56.7 Mt of oxide and upper transition at 0.56 g/t Au and 38.9 Mt of lower transition and fresh at 1.02 g/t Au.
  - 7.9 Mt of existing oxide stockpiles at 0.40 g/t Au.
- 2.1 Moz of Au produced.
- 187.6 Mt waste.

- 2.0 strip ratio.
- 11.3-year mine life.

This feasibility study demonstrates that the Bombore project has positive economics and the proposed expansion to include hard rock processing is well warranted.

25.4 Mineral Processing, Metallurgical Testing, and Recovery Methods

The process design pertains to the feasibility test work undertaken in 2023 on composite samples of fresh / hard rock material from drill cores supplied from the Bomboré mineral resource. The testwork was conducted in South Africa at Maelgwyn for gold extraction and MacOne for thickening. The 2023 testwork campaign expanded from a feasibility test work program undertaken in 2019 on composites of lower transition and fresh ores from drill cores supplied from the Bomboré mineral resource. The 2019 testwork was conducted in Canada at Base Met Labs for comminution, mineralogy, gold extraction, and thickening.

From this testwork and the processing schedule provided by Orezone, the expected average gold recovery over the whole life of mine is 88%.

The process plant design is based on a metallurgical flowsheet developed for maximizing recovery while minimizing initial capital expenditure and life of mine (LOM) operating costs. The flowsheet is based on unit operations including crushing, milling, pre-oxidation, Carbon-in-Leach (CIL), Zadra elution, gold electrowinning and carbon regeneration that are well proven in the industry.

Orway Milling Consultants (OMC) used the comminution results and evaluated a primary crush SABC and SS-SAG circuit design with the SS-SAG option chosen as the preferred option to achieve an 80% passing 75-micron product size after classification.

Cyclone overflow from the milling circuit will report to the pre-leach thickener. Underflow from this thickener will be processed by pre-oxidation and a CIL circuit with oxygen sparging throughout, to increase gold leaching kinetics to achieve ultimate gold dissolution in 24 hours of residence time.

Elution, electrowinning and smelting of gold will be applied to generate the final product. In addition to the metallurgical processing circuits, reagents and utilities will all be included within the plant boundary, as well as typical plant infrastructure.

25.5 Tailings Disposal and Site Water Management

Design of the TSF was based on fill material being sourced from the open pit mining operations. If waste is not readily available additional borrows will be required and may result in construction cost increases. The life of mine embankment fill requirements should be cross-checked with the mine waste production estimates over the same timeframe. The design can be reviewed and potentially amended to suit the waste production if quantities and/or types of waste do not meet the TSF construction requirements. Filters and drainage materials will be sourced elsewhere.

The TSF embankment crest elevations are based on the physical tailings characteristics and design throughput. Changes in these characteristics and/or throughput may result in changes in the tailings settled density. This may impact the tonnage capacity of the facility and therefore may require adjustment of the construction schedule. As such, measuring throughput, ore blend, rate of rise, beach slopes and achieved densities should be undertaken as part of the monitoring process in order to plan and stage future embankment raises to suit operational practices. Physical testing to confirm tailings properties should be undertaken regularly to verify the design parameters.

Initially, the very high rate of rise will result in lower densities being achieved during the early years of deposition. With the increase in throughput to 10.3 Mtpa the development of additional area is critical to reduce the rate of rise of the tailings. When sufficient footprint is established, the rate of rise will reduce. The pond extents will be limited to a small proportion of the tailings basin and the operators will have developed a continuous rotation of the deposition area, thereby enhancing drying of the tailings and subsequently increasing tailings density and strength.

Geochemical testing of the tailings should be undertaken regularly throughout the life of the facility, particularly when ore sources vary or changes to ore blending occurs, to ensure that design assumptions remain valid. It is expected this work will continue as part of ongoing operations.

Geochemical testing of the waste materials should be undertaken regularly during the life of the facility, particularly when different mineralisation is encountered, to ensure that any potentially acid forming materials are identified and managed appropriately. It is expected this work will be conducted as part of ongoing operations.

Based on the stability assessment, the TSF embankment will have factors of safety equal to or higher than the recommended minimum factors of safety recommended by ANCOLD and therefore should be considered as stable.

During deposition the cyanide level within the supernatant pond will reduce by volatilisation and other destructive processes as the supernatant runs down the beach. On reaching the supernatant pond, decay of the cyanide in the pond will continue as a result of the pH change within the pond water and the exposure to ultraviolet light. Without cyanide destruction measures in place, KP experience in West Africa is that supernatant ponds are typically in the range of 1 – 10 mg/L CN_{wad}. However, this should be regularly monitored during operations to ensure the levels are maintained below the required thresholds.

The required elevation of the TSF spillway invert is a function of the required wet season storage allowance, extreme storm storage and contingency freeboards. The total freeboard required from the average pond volume is the combination of the following freeboard provisions:

- 10 year ARI, wet 3 month sequence with no evaporation and recycle to the process plant allowed.
- 100 year ARI, 72 hour duration storm event with no evaporation, no decant return.
- 10 year ARI wave run-up; plus 0.3 m additional contingency freeboard.

The required minimum crest level is a function of the spillway flow depth during the design flood event for the various stages of construction. The spillway depth has been set at 0.5 m. As such the minimum embankment crest level is 0.5 m plus the 10 year ARI wave run-up higher than the design spillway invert. Diligent monitoring and management of the TSF supernatant pond will be critical throughout the operational life.

25.6 Environmental, Social, and Permitting

The approach developed by Orezone throughout the various environmental and social studies that have been conducted since 2009, especially in the context of the Environmental and Social Impact Assessment, emphasized stakeholder concerns and integration of the environmental and social aspects into the initial stages of the Bomboré Mine design and its continuation into the Phase II Expansion design. This approach has ensured the integration of environmental and social issues in designing Bomboré Mine.

Various permits and authorizations are required for the Bomboré Mine. Orezone holds all permits that are required for its current operations and those envisioned in the 2019 FS. Orezone has been successful in obtaining such permits and authorizations in the past and is confident that it will be able to obtain the required permits and authorizations for the Phase II Expansion.

RAP Phases II and III follow the successful completion of Phase I RAP and involves the construction of three new resettlement communities (MV3, MV2, and BV2). Phase II is well-advanced with the construction of MV3 sequenced as the first community to construct in order to gain access to mining areas that are currently contemplated in the 2024 mine plan.

A RAP Phase IV is planned to accommodate an increased footprint to the mining lease. This resettlement will be performed progressively over 2024 through to 2027.

26.0 RECOMMENDATIONS

26.1 Summary

The Bomboré Property hosts significant gold deposits with estimated Mineral Resources and Mineral Reserves. Orezone has undertaken considerable exploration, front-end engineering, and development work. The near-term primary objective is to further advance engineering work on the Project as discussed herein. Investigation of the Project optimizations noted in this Technical Report should also be considered. The overall recommendations are presented below.

26.2 Geology and Mineral Resources

Recommendations for geology and Mineral Resources work to be done follow below.

- Hyperspectral alteration analysis (ASTER / Sentinel) should be considered to complement existing alteration analysis.
- Migrate and reconcile the P&E block models to the in-house Orezone modelling software systems.
- Extend model space in Leapfrog GEO™ and Surpac™ to accommodate any possible pit expansions.
- Ensure that all of the model spaces are completely populated with lithology, bulk density and oxidation state models.
- Ensure that up-to-date topography covers the entire model space.
- Formalize Mineral Resource cut-offs to the consensus gold price or two-year trailing average.
- Complete a tight pattern of drill holes in the better-defined veins to better model the nugget effect.
- Include a quarterly block-model reconciliation program that incorporates the Mineral Resource models, grade control models, and production results.

26.3 Mining

The following items are recommended as mining continues at Bomboré:

- Implement and continue detailed regular reconciliation to validate the Mineral Resource and Reserve Estimates and confirm actual dilution and losses.

- Update resource block models to incorporate consistent block model origin levels and ensure better alignment of blocks with site reference bench elevations to assist in future mine planning and design. Similarly, incorporate consistent block model construction and classification methods across all models.

- Continue resource delineation drilling and incorporate it into LOM mine plans. Potential to have pits coalesce and expand.

- Re-evaluate starter pits to ensure alignment with pit optimization and strategic targets rather than material movement constraints.

- Optimization of in-pit and ex-pit haul road designs to minimize haulage distances and construction requirements.

- Consider further opportunities to use in-pit dumps and stockpiles while being cognizant of potential ore sterilization.

- Reconcile and re-evaluate on site mining contractor costs.

26.4 Mineral Processing, Metallurgical Testing, and Recovery Methods

It is recommended that the following additional testwork be considered to improve confidence in the hard rock plant design:

- Optimization testwork with an additional gravity gold recovery study with gravity tails leaching at the current established optimum leach conditions to evaluate future inclusion of a gravity recovery circuit.

In addition, it is recommended that during plant operations:

- Natural cyanide attenuation (free and WAD) continues to be tested and monitored in the tailings storage facility.

- Site water quality (raw and process) continues to be tested and monitored during the wet and dry seasons to document any seasonal impact of water quality.

26.5 Tailings Disposal and Site Water Management

The design is based on an average tailings beach slope of 1.0% (1V:100H). However, the beach slope is heavily dependent on the grind size and the ore blend. Thus, small changes in plant performance, ore type, or ore blend have the potential to change the tailings beach slope. As such, tailings parameters should be continuously monitored.

One advantage of staging the embankment construction is the ability to modify the design at each stage based on measured data obtained from the TSF. Thus, in cases where the measured beach slope is different to the design beach slope, the timing and height of the subsequent embankment raise can be modified to account for the change in beach slope.

- Steeper Beach Slope - If the measured beach slope is steeper than the design slope, the tailings rate of rise against the TSF embankments will be faster than expected and the TSF will reach capacity earlier than scheduled. If this was identified as a significant issue the response would be to bring construction of the next embankment stage forward. The tailings storage capacity would be reduced as a consequence of a steeper beach slope, and the stormwater storage capacity would be increased.

- Flatter beach slope - If the measured tailings beach slope is flatter than the design slope, the capacity of the TSF would be increased. The overall TSF stormwater storage capacity would be reduced but should still be sufficient unless construction of the subsequent stage was deferred beyond the original design schedule.

Achieved Densities - The TSF embankment crest elevations are based on the physical tailings characteristics and design throughput. Changes in these characteristics and/or throughput may result in changes in the tailings settled density which in turn may impact the tonnage capacity of the facility. Similar to variations in tailings beach slope, this may require adjustment of the construction schedule. As such, monitoring of throughput, ore blend, rate of rise and achieved densities should be undertaken as part of the audit process in order to plan and stage future embankment raises to suit operational practice. Physical testing to determine current tailings properties should be undertaken regularly to verify the design parameters.

Tailings Geochemistry - Geochemical testing of the tailings should be undertaken continuously throughout the life of the facility, particularly when mineralisation sources vary or changes to ore blending occurs, to ensure that design assumptions remain valid. It is expected that work will continue as part of ongoing operations.

Life of Mine Planning - Changes to the life of mine plan or throughput may impact the tailings management requirements for the site. Any significant increases in total throughput will require a review of the TSF design.

Raise Construction - The ability to construct earthworks during wet seasons can be limited, so construction over the life of the project needs to be carefully planned and monitored so that approval, budgeting, and logistics are in place to allow works to be completed promptly and prior to the onset of the wet season.

Construction Materials Availability - Design of the TSF was based on fill material being sourced from the open pit mining operations. If waste is not readily available additional borrows will be required and may result in construction cost increases. KP has not undertaken any cross-checks of the life of mine embankment fill requirements versus the mine waste production estimates over the same timeframe. The design can be reviewed and potentially amended to suit the waste production if quantities and/or types of waste do not meet the TSF construction requirements.

Mine Waste Geochemistry - Geochemical testing of the waste materials should be undertaken regularly during the life of the facility, particularly when different mineralisation is encountered, to ensure that any potentially acid forming materials are identified and managed appropriately. It is expected that work will be conducted as part of ongoing operations.

Survey - Inaccurate survey is the most common cause of variations between expected and actual quantities, particularly in reference to bulk fill earthworks volumes. Topographical contours should be validated prior to the design and construction of each stage.

Engineered Soil Cover - The current design for closure and decommissioning of the TSF includes an engineered fill cover constructed over the tailings beach as the most suitable long-term solution. On-going characterisation testing (completed by others) during operations may consider alternative options.

The water balance modelling showed that the existing OCR and new Raw Water Reservoir water volumes will be affected by the evaporation and seepage losses. Variations or modifications to the seepage loss parameters will affect the required water demand. Therefore, in order to calibrate the water balance model with actual site conditions during operations, it is recommended that monitoring of all water storage areas and water movement be recorded. This will allow better accuracy in estimating water losses and therefore estimation of water requirements over the life of mine by calibration of the water balance model.

The water balance modelling is extremely sensitive to seepage assumptions and should seepage losses be greater than the current estimates this may result in the reservoirs running dry earlier than estimated. Therefore, it is also recommended that a hydrogeological assessment be completed to incorporate the mining plan for pit development and pit dewatering as these will likely impact the OCR seepage rates.

An excessive loss of storage volume to sediment build up could impact the ability of the reservoirs to provide adequate water in extreme dry conditions. As such it is recommended that, when practicable, the sediment should be regularly monitored and removed from the reservoirs to ensure that adequate storage for water is maintained.

The average annual rainfall is estimated to be 767 mm and the average annual lake evaporation 2,029 mm. The majority of rainfall (approximately 92%) occurs between June and October. It is recommended that the earthworks are undertaken during the dry season so that the moisture content of earthworks materials can be better controlled and less temporary surface water management is required.

The site surface water management design should be reviewed with any development or adjustments to the overall site plan.

26.6 Environmental, Social and Permitting

In early 2023, Orezone engaged Société de Conseil et de Realisation pour la Gestion de l'Environnement (SOCREGE), an independent West African consulting company, to assist Orezone with the ESIA required for the Phase II Expansion. In June 2023 the Terms of Reference were approved by ANEVE and SOCREGE, in consultation with Orezone. Orezone anticipates that the ESIA will be completed in Q4-2023 and approved in Q1-2024.

26.7 Recommended Exploration Program and Budget

The Phase IV exploration programme on the Bomboré mining lease will include the definition drilling that is required prior to the 2024 grade control drilling campaign. Initial Mineral Resource models are also scheduled in 2024 on the Kiin Tanga and P13 satellite deposits, as is completion of the resistivity survey at P13.

The recommended Phase IV exploration work programme is presented in Table 26.7.1. The cost is estimated to be US$5.4M.

Table 26.7.1 Recommended Phase IV Exploration Programme and Budget

Tenement	Phase IV Programme	Quantity	Budget US$
Mining Lease	Advanced Grade Control RC Definition Drilling - Phase III (2024)	57,000 m	3,705,000
	P17 Core Definition Drilling Phase IV	2,560 m	409,600
	Advanced Grade Control Core Definition Drilling - Phase I (2024)	7,600 m	1,216,000
Bomboré II	KT Initial Resource Model		7,500
Bomboré V	P13 Initial Resource Model		25,000
	P13 Resistivity Survey	25 km	13,750
TOTAL Phase IV			**5,376,850**

The Phase V exploration programme on the Bomboré mining lease will include the definition drilling that is required prior to the 2025 grade control drilling campaign. Mineral Resource expansion drilling programmes at P17, Kiin Tanga and P13 are also expected, given the expansion potential already identified at P17, and assuming that initial Mineral Resource models at Kiin Tanga and P13 will identify viable expansion targets.

The recommended Phase V work programme is presented in Table 26.7.2. The cost is estimated to be US$5.4M.

Table 26.7.2 Recommended Phase V Exploration Program and Budget

Tenement	Phase V Programme	Quantity	Budget US$
Mining Lease	Advanced Grade Control RC Definition Drilling - Phase IV (2025+ LOM)	26,000 m	1,690,000
	P17 Core Definition Drilling Phase V	2,590 m	414,400
	Advanced Grade Control Core Definition Drilling - Phase II (2025+)	9,500 m	1,520,000
Bomboré II	KT-Mineral Resource Expansion RC Drilling	2,500 m	162,500
	KT-Mineral Resource Expansion Core Drilling	750 m	120,000
Bomboré III	P17 Core Definition Drilling Phase IV	3,275 m	524,000
	P17 Core Definition Drilling Phase V	4,385 m	701,600
Bomboré V	P13-Mineral Resource Expansion RC Drilling	2,500 m	162,500
	P13-Mineral Resource Expansion Core Drilling	750 m	120,000
TOTAL Phase V			**5,415,000**

27.0 REFERENCES

Buro, Y. and Saucier, G. 2008. *Technical Report on the Mineral Resource of the Bomboré Gold Project*. Prepared by Met-Chem Canada Inc. for Orezone Inc.

Cole, G., El-Rassi, D., Chartier, D., and Leuangthong, O., 2012: *Technical Report on the Bomboré Gold Project, Burkina Faso, West Africa, SRK Project Number 3CO006.002*. Report prepared by SRK (Canada) Inc. for Orezone Gold Corporation. 360p.

Cole, G. and El-Rassi, D. November 2008. *Technical Report on the Bomboré Gold Project in Burkina Faso, West Africa*. Prepared by SRK Consulting (Canada) Inc.

Cole, G. and El-Rassi, D. November 2010. *Technical Report on the Bomboré Gold Project in Burkina Faso, West Africa*. Prepared by SRK Consulting (Canada) Inc.

Cole, G., El-Rassi, D., Chartier, D. and Leuangthong, O. 2012. Technical Report on the Bomboré Gold Project, Burkina Faso, West Africa. Prepared by SRK (Canada) Inc.

Defilippi, C., Ehasoo, G., Cole, G., Minard, T., Rojanschi, V., Masengo, E., Major, G. and Houle, J.S., 2015. *Feasibility Study NI 43-101 Technical Report on the Bomboré Gold Project, Burkina Faso, West Africa.* Prepared by Kappes, Cassiday & Associates and dated 28 April 2015. NB: withdrawn 2016.

Groves, D.I. and Santosh, M. 2016. *The giant Jiadong gold province: The key to a unified model for orogenic gold deposits.* Geoscience Frontiers 7, 409-417.

Goldfarb, R.J. and Groves, D.I. 2015. *Orogenic gold: common or evolving fluid and metal sources through time*. Lithos 233, 2-26.

Gourde, R., Gignac, L.P. and Menard, R., 2014. *Preliminary Economic Assessment, Bomboré Gold Project, Burkina Faso*.

Hagemann, S.G. and Cassidy, K.F. 2000. *Archean orogenic lode gold deposits*. Reviews in Economic Geology 13, 9-68.

Knight Piésold 2023, *2023 Feasibility Study, Tailings Storage Facility Expansion and Associated Infrastructure, Bomboré Gold Project, KP Job No. PE401-00122/29, KP Report No. PE401-00122/19*.

Maelgwyn South Africa (Pty) Ltd, 2023. Bombore Study Final Report, report No. 22-139-3 revision 0. Unpublished Company report.

Maes, E. 2014a. *Technical report of the sample preparation and analytical quality control of the Bomboré Gold Project, 01 July 2013 – 31 August 2014, RC and DD drilling, geotechnical and check sampling programs*.

Maes, E. 2014b. *Technical report of the sample preparation and analytical quality control of the Bomboré Gold Project, 01 July 2013 – 31 August 2014, RC and DD drilling, geotechnical and check sampling programs.*

Maes, E. 2015. *Technical report of the sample preparation and analytical quality control of the Bomboré Gold Project, 01 September 2014 – 02 February 2015, Geotechnical RC drilling and trenching and DD and check sampling programs.*

Orway Mineral Consultants Pty Ltd. 2023. Definitive Feasibility Study Comminution Circuit Design, revision 0, report no. 7327-001. Unpublished Company report.

Pressacco, R.., Carlsson J.T., Ciucelescu, T., Ehasoo G., Scott, T., Houle, J-S., 2017. *Technical report on the updated mineral resource estimate for the Bomboré Gold Project, Burkina Faso, West Africa*. Prepared by RPA and dated 12 January 2017.

RPA. 2017. *Technical report on the updated mineral resource estimate for the Bomboré gold project, Burkina Faso, West Africa*. Technical report dated 12 January 2017, Roscoe Postle Associates Inc., Toronto.

RPA. 2019. *Bombore Gold Project P17S Mineral Resource Update*. Memorandum dated 8 February 2019, Roscoe Postle Associates Inc., Toronto.

Tukey, J.W., 1977. *Exploratory Data Analysis*. Addison-Wesley, Reading, MA. 1st Edition.

28.0 QP CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON

I, Georgi Doundarov, M.Sc., P.Eng., PMP, CCP, as an author of this technical report entitled "Bomboré Phase II Expansion, Definitive Feasibility Study, 43-101 Technical Report" dated effective March 28, 2023 (the "**Technical Report**"), which was prepared for the issuer, Orezone Gold Corporation ("Orezone"), do hereby certify that:

1) I am a Senior Study Manager with Lycopodium Minerals Canada Ltd. My office address is Suite 700, 5090 Explorer Drive Mississauga, ON L4W 4T9, Canada.

2) I am a graduate of the University of Mining and Geology, 1996 with a M.Sc degree in Mineral Processing and Metallurgy as well as a graduate from the Yokohama National University, Yokohama, Japan, 2005 with a M.Sc. degree in Infrastructure Management - Mineral Processing and Metallurgy.

3) I am a Member of the Professional Engineers Ontario (PEO) and registered as a Professional Engineer in the province of Ontario with a number 100107167. I have worked as a metallurgical engineer and project manager for a total of over 25 years since my graduation. My relevant experience for the purpose of the Technical Report is:

   - review and report as a consultant on numerous process facilities and mining projects around the world for due diligence and regulatory requirements;
   - Study Manager on a number of feasibility studies and detailed designs in the gold industry in Africa, Australia and Asia;
   - Lead metallurgist at a number of gold mines in Africa, Australia and Asia; and
   - development, execution and interpretation of a number of testwork programs in gold, copper, base metals and iron ore.

4) 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 fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I have not visited the site of the Bomboré Gold Project.

6) I am responsible for all of the preparation of Sections 1.1, 1.2, 1.3, 1.13, 1.15, 1.16, 1.17, 2, 3, 4, 5, 6, 18.1 to 18. 12, 19, 20, 21.1, 21.2.1, 21.3, 22, 24, 25.1, 26.1, and 27. of the Technical Report.

7) I am independent of the issuer as described in section 1.5 of NI 43-101.

8) I have no prior involvement with the properties that are the subject of this Technical Report.

9) I have read NI 43-101, and the parts of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.

10) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for, contain all scientific and technical information that is required to be disclosed and to make the Technical Report not misleading.

Dated this 24th day of November 2023.

"*signed*"

Georgi Doundarov, P.Eng., PMP, CCP

CERTIFICATE OF QUALIFIED PERSON

I, Olav Mejia, P.Eng., as an author of this technical report entitled "Bomboré Phase II Expansion, Definitive Feasibility Study, 43-101 Technical Report" dated effective March 28, 2023 (the "**Technical Report**"), which was prepared for the issuer, Orezone Gold Corporation ("Orezone"), do hereby certify that:

1) I am a Manger of Process with Lycopodium Minerals Canada Ltd. My office address is 5090 Explorer Drive, Suite 700, Mississauga, Canada L4W 4T9.

2) I am a graduate of the University of San Marcos with a B.Eng. degree in Chemical Engineering and graduate of the University of British Columbia with a MASc degree in Mineral Processing.

3) I am a member of the Professional Engineers of Ontario and registered as a Professional Engineer in Canada with registration number 100602612. I have worked as a Chemical engineer and mineral processing engineer for over 25 years since my graduation. My relevant experience for the purpose of the Technical Report is:

   - Metallurgical and operational experience in copper and molybdenum concentrators and gold plant operation and gold plant designs within South America and North America

   - Lead process engineer on copper and gold projects ranging from testwork management, all phases of studies and detail design up to commissioning of numerous projects in Canada and Americas.

   - Lead process engineer on copper and molybdenum plants in Canada.

4) 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 fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I have not visited the site of the Bomboré Gold Project.

6) I am responsible for all of the preparation of Sections 1.6, 1.10, 13, 17, 21.2.3, 25.4, 26.4 of the Technical Report.

7) I am independent of the issuer as described in section 1.5 of NI 43-101.

8) I have no prior involvement with the properties that are the subject of this Technical Report.

9) I have read NI 43-101, and the parts of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.

10) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for, contain all scientific and technical information that is required to be disclosed and to make the Technical Report not misleading.

Dated this 24th day of November 2023.

"*signed*"

Olav Mejia, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

EUGENE PURITCH, P. ENG., FEC, CET

I, Eugene Puritch, P. Eng., FEC, CET, residing at 44 Turtlecreek Blvd., Brampton, Ontario, L6W 3X7, do hereby certify that:

1. I am an independent mining consultant and President of P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Bomboré Phase II Expansion, Definitive Feasibility Study, 43-101 Technical Report", (The "Technical Report") with an effective date of March 28, 2023.
3. I am a graduate of The Haileybury School of Mines, with a Technologist Diploma in Mining, as well as obtaining an additional year of undergraduate education in Mine Engineering at Queen's University. In addition, I have also met the Professional Engineers of Ontario Academic Requirement Committee's Examination requirement for a Bachelor's degree in Engineering Equivalency. I am a mining consultant currently licensed by the: Professional Engineers and Geoscientists New Brunswick (License No. 4778); Professional Engineers, Geoscientists Newfoundland and Labrador (License No. 5998); Association of Professional Engineers and Geoscientists Saskatchewan (License No. 16216); Ontario Association of Certified Engineering Technicians and Technologists (License No. 45252); Professional Engineers of Ontario (License No. 100014010); Association of Professional Engineers and Geoscientists of British Columbia (License No. 42912); and Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (No. L3877). I am also a member of the National Canadian Institute of Mining and Metallurgy.

   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 fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

   I have practiced my profession continuously since 1978. My summarized career experience is as follows:

   - Mining Technologist - H.B.M.& S. and Inco Ltd., 1978-1980
   - Open Pit Mine Engineer – Cassiar Asbestos/Brinco Ltd., 1981-1983
   - Pit Engineer/Drill & Blast Supervisor – Detour Lake Mine, 1984-1986
   - Self-Employed Mining Consultant – Timmins Area, 1987-1988
   - Mine Designer/Resource Estimator – Dynatec/CMD/Bharti, 1989-1995
   - Self-Employed Mining Consultant/Resource-Reserve Estimator, 1995-2004
   - President – P&E Mining Consultants Inc, 2004-Present
4. I have not visited the Bomboré Gold Project that is the subject of this Technical Report.
5. I am responsible for co-authoring Sections 1.7, 14, 25.2, 26.2, 26.7 and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Bomboré Gold Project that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1. This Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: March 28, 2023
Signed Date: November 24, 2023

{SIGNED AND SEALED}
[Eugene Puritch]

Eugene Puritch, P.Eng., FEC, CET

CERTIFICATE OF QUALIFIED PERSON

ANTOINE YASSA, P.GEO.

I, Antoine Yassa, P.Geo. residing at 3602 Rang des Cavaliers, Rouyn-Noranda, Quebec, J0Z 1Y2, do hereby certify that:

1. I am an independent geological consultant contracted by P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled “Bomboré Phase II Expansion, Definitive Feasibility Study, 43-101 Technical Report”, (The “Technical Report”) with an effective date of March 28, 2023.
3. I am a graduate of Ottawa University at Ottawa, Ontario with a B. Sc (HONS) in Geological Sciences (1977) with continuous experience as a geologist since 1979. I am a geological consultant currently licensed by the Order of Geologists of Québec (License No 224) and by the Association of Professional Geoscientist of Ontario (License No 1890);

   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 fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:
   - Minex Geologist (Val d’Or), 3-D Modeling (Timmins), Placer Dome 1993-1995
   - Database Manager, Senior Geologist, West Africa, PDX, 1996-1998
   - Senior Geologist, Database Manager, McWatters Mine 1998-2000
   - Database Manager, Gemcom modeling and Resources Evaluation (Kiena Mine) 2001-2003
   - Database Manager and Resources Evaluation at Julietta Mine, Bema Gold Corp. 2003-2006
   - Consulting Geologist 2006-present
4. I have visited the site of the Bomboré Gold Project that is the subject of this Technical Report from October 9 to 14, 2017.
5. I am responsible for co-authoring Sections 12, 14 and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101. I am independent of the Vendor and the Property.
7. Other than the 2017 site visit noted in Item 4 above, I have had no prior involvement with the Bomboré Gold Project that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1. This Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: March 28, 2023
Signed Date: November 24, 2023

{SIGNED AND SEALED}
[Antoine Yassa]

____
Antoine Yassa, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

WILLIAM STONE, PH.D., P.GEO.

I, William Stone, Ph.D., P.Geo, residing at 4361 Latimer Crescent, Burlington, Ontario, do hereby certify that:

1. I am an independent geological consultant working for P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Bomboré Phase II Expansion, Definitive Feasibility Study, 43-101 Technical Report", (The "Technical Report") with an effective date of March 28, 2023.
3. I am a graduate of Dalhousie University with a Bachelor of Science (Honours) degree in Geology (1983). In addition, I have a Master of Science in Geology (1985) and a Ph.D. in Geology (1988) from the University of Western Ontario. I have worked as a geologist for a total of 35 years since obtaining my M.Sc. degree. I am a geological consultant currently licensed by the Professional Geoscientists of Ontario (License No 1569).

   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 fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:
   - Contract Senior Geologist, LAC Minerals Exploration Ltd. 1985-1988
   - Post-Doctoral Fellow, McMaster University 1988-1992
   - Contract Senior Geologist, Outokumpu Mines and Metals Ltd. 1993-1996
   - Senior Research Geologist, WMC Resources Ltd. 1996-2001
   - Senior Lecturer, University of Western Australia 2001-2003
   - Principal Geologist, Geoinformatics Exploration Ltd. 2003-2004
   - Vice President Exploration, Nevada Star Resources Inc. 2005-2006
   - Vice President Exploration, Goldbrook Ventures Inc. 2006-2008
   - Vice President Exploration, North American Palladium Ltd. 2008-2009
   - Vice President Exploration, Magma Metals Ltd. 2010-2011
   - President & COO, Pacific North West Capital Corp. 2011-2014
   - Consulting Geologist 2013-2017
   - Senior Project Geologist, Anglo American 2017-2019
   - Consulting Geoscientist 2020-Present
4. I have not visited the site of the Bomboré Gold Project that is the subject of this Technical Report.
5. I am responsible for authoring Sections 1.4, 1.5, 7, 8, 9, 10, 23 and co-authoring Sections 25.2, 26.2, 26.7 and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had no prior involvement with the Bomboré Gold Project that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: March 28, 2023
Signed Date: November 24, 2023

{SIGNED AND SEALED}
[William Stone]

William Stone, Ph.D., P.Geo.

CERTIFICATE OF QUALIFIED PERSON

JARITA BARRY, P.GEO.

I, Jarita Barry, P.Geo., residing at 9052 Mortlake-Ararat Road, Ararat, Victoria, Australia, 3377, do hereby certify that:

1. I am an independent geological consultant contracted by P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Bomboré Phase II Expansion, Definitive Feasibility Study, 43-101 Technical Report", (The "Technical Report") with an effective date of March 28, 2023.
3. I am a graduate of RMIT University of Melbourne, Victoria, Australia, with a B.Sc. in Applied Geology. I have worked as a geologist for over 17 years since obtaining my B.Sc. degree. I am a geological consultant currently licensed by Engineers and Geoscientists British Columbia (License No. 40875) and Professional Engineers and Geoscientists Newfoundland & Labrador (License No. 08399). I am also a member of the Australasian Institute of Mining and Metallurgy of Australia (Member No. 305397);

   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 fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:
   - Geologist, Foran Mining Corp. 2004
   - Geologist, Aurelian Resources Inc. 2004
   - Geologist, Linear Gold Corp. 2005-2006
   - Geologist, Búscore Consulting 2006-2007
   - Consulting Geologist (AusIMM) 2008-2014
   - Consulting Geologist, P.Geo. (EGBC/AusIMM) 2014-Present
4. I have not visited the site of the Bomboré Gold Property that is the subject of this Technical Report.
5. I am responsible for authoring Section 11 and co-authoring Sections 12 and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had no prior involvement with the Project that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: March 28, 2023
Signed Date: November 24, 2023

{SIGNED AND SEALED}
[Jarita Barry]

Jarita Barry, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

FRED H. BROWN, P.GEO.

I, Fred H. Brown, of PO Box 332, Lynden, WA, USA, do hereby certify that:

1. I am an independent geological consultant and have worked as a geologist continuously since my graduation from university in 1987.
2. This certificate applies to the Technical Report titled "Bomboré Phase II Expansion, Definitive Feasibility Study, 43-101 Technical Report", (The "Technical Report") with an effective date of March 28, 2023.
3. I graduated with a Bachelor of Science degree in Geology from New Mexico State University in 1987. I obtained a Graduate Diploma in Engineering (Mining) in 1997 from the University of the Witwatersrand and a Master of Science in Engineering (Civil) from the University of the Witwatersrand in 2005. I am registered with the Association of Professional Engineers and Geoscientists of British Columbia as a Professional Geoscientist (171602) and the Society for Mining, Metallurgy and Exploration as a Registered Member (#4152172).

   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 fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:
   - Underground Mine Geologist, Freegold Mine, AAC 1987-1995
   - Mineral Resource Manager, Vaal Reefs Mine, Anglogold 1995-1997
   - Resident Geologist, Venetia Mine, De Beers 1997-2000
   - Chief Geologist, De Beers Consolidated Mines 2000-2004
   - Consulting Geologist 2004-2008
   - P&E Mining Consultants Inc. – Sr. Associate Geologist 2008-Present
4. I have not visited the site of the Bomboré Gold Property that is the subject of this Technical Report.
5. I am responsible for co-authoring Section 14 and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have no prior involvement with the with the Bomboré Gold Property that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: March 28, 2023
Signing Date: November 24, 2023

{SIGNED AND SEALED}
[Fred H. Brown]

Fred H. Brown, P.Geo.

AMC Mining Consultants (Canada) Ltd.
BC0767129

200 Granville Street, Suite 202
Vancouver BC V6C 1S4
Canada

T +1 604 669 0044
E vancouver@amcconsultants.com
W amcconsultants.com

CERTIFICATE OF AUTHOR

I, David A. Warren, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as Senior Principal Mining Engineer with AMC Mining Consultants (Canada) Ltd., with an office at Suite 202, 200 Granville Street, Vancouver, British Columbia V6C 1S4.

2 This certificate applies to the technical report entitled "Bomboré Phase II Expansion, Definitive Feasibility Study, 43-101 Technical Report" dated effective March 28, 2023 (the "**Technical Report**"), which was prepared for the issuer, Orezone Gold Corporation ("Orezone").

3 I graduated with a degree of B.A.Sc. in Mining Engineering in 1978 from the University of British Columbia (UBC), Canada, and a degree of M.Sc. In Technology, Materials Science and Rock Engineering, Helsinki University of Technology (HUT), Finland, 1997.

4 I am a registered member in good standing of the Engineers and Geoscientists of British Columbia (License #15053) and L'ordre des ingénieurs du Québec (OIQ) (License #121481).

I have worked in the resource industries for 44 years since my graduation and have 35 years of relevant experience in open pit mining, feasibility studies, and technical report preparation for mining projects.

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 fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5 I have not visited the Bomboré Gold Project

6 I am responsible for Sections 15, 16, and parts of 1, 21, 25, and 26 of the Technical Report.

7 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of NI 43-101.

8 I have had prior involvement with the property that is the subject of the Technical Report in that I was involved in conducting work and preparing the previous AMC Technical Reports on the Bomboré property in 2018 (filed 23 August 2018, effective date 9 July 2018) and 2019 (filed 13 August 2019, effective date 26 June 2019).

9 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101.

10 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 24'th day of November 2023

signed

David A. Warren, P.Eng.
Senior Principal Mining Engineer
AMC Mining Consultants (Canada) Ltd.

CERTIFICATE OF QUALIFIED PERSON

I, David J T Morgan, M.Sc., MIEAust, CPEng, APEC Engineer, IntPE(Aus) , as an author of this technical report entitled "Bomboré Phase II Expansion, Definitive Feasibility Study, 43-101 Technical Report" dated effective March 28, 2023 (the "**Technical Report**"), which was prepared for the issuer, Orezone Gold Corporation ("Orezone"), do hereby certify that:

1) I am the Managing Director of Knight Piésold Australia Pty Ltd. My office address is Level 1, 184 Adelaide Terrace, East Perth, WA, 6004, Australia.

2) I am a graduate of the University of Manchester, 1980 with a B.Sc degree in Civil Engineering as well as a graduate from the University of Southampton, 1981 with a M.Sc. degree in Irrigation Engineering.

3) I am a member 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). I have worked as a Civil Engineer and Project Director for a total of over 40 years since my graduation. My relevant experience for the purpose of the Technical Report is:

    - review and report as a consultant on numerous tailings management facilities and mining projects around the world for due diligence and regulatory requirements;

    - Project Director on a number of feasibility studies and detailed designs for multiple commodities in Africa, Australia and Asia.

4) 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 fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I have visited the site of the Bomboré Gold Project.

6) I am responsible for all of preparation of Sections 18.13, 18.14, 25.5, and 26.5 of the Technical Report.

7) I am independent of the issuer as described in section 1.5 of NI 43-101.

8) I have no prior involvement with the properties that are the subject of this Technical Report.

9) I have read NI 43-101, and the parts of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.

10) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for, contain all scientific and technical information that is required to be disclosed and to make the Technical Report not misleading.

Dated this 24th day of November, 2023.

"*signed*"

David J T Morgan, MIEAust, CPEng, APEC Engineer, IntPE(Aus), MAusIMM

AFRICA LABEL GROUP S.A.

CERTIFICATE OF QUALIFIED PERSON

I, Bright Oppong Afum, Ph.D., M.Sc., P.Eng., MAusIMM(CP), as an author of this technical report entitled "Bomboré Phase II Expansion, Definitive Feasibility Study, 43-101 Technical Report" dated effective March 28, 2023 (the "**Technical Report**"), which was prepared for the issuer, Orezone Gold Corporation ("Orezone"), do hereby certify that:

1) I am a Consultant with Africa Label Group Inc. My office address is Mining Engineering Department, University of Mines and Technology (UMaT), Tarkwa, Western Region, Ghana, and Secteur 51 Karpala, Ouagadougou, Burkina Faso.

2) I hold the following academic qualifications:
   - Ph.D., Mining Engineering, Laurentian University, Canada, 2021
   - Citation, Applied Geostatistics, University of Alberta, Canada, 2018
   - M.Sc., Environmental Monitoring and Analysis, Aberystwyth University, Wales, 2012
   - B.Sc., Mining Engineering, Kwame Nkrumah University of Science and Technology, Ghana, 2008

3) I am a Member of the Professional Engineers Ontario (PEO) and registered as a Professional Engineer in the province of Ontario with a number 100555055. I am a Member of The Australasian Institute of Mining and Metallurgy Australasian (The AusIMM) and registered AusIMM Chartered Professional with a number 317952. I have worked as an environmental geo-analyst and mining engineer for a total of over 15 years since my graduation. My relevant experience for the purpose of the Technical Report is:

   - review and report as a consultant on numerous environmental and mining projects around the world for due diligence, validatory, and regulatory requirements;

   - study manager on a number of environmental baseline studies and prediction for feasibility studies in the gold, granite and limestone industries in Africa, Wales, Ireland, and Canada;

   - study manager on several of mine closure, reclamation, and rehabilitation gold and limestone mining projects in Africa; and

   - development, execution, and interpretation of a number of test work programs in gold and limestone.

4) 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 fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I visited the site of the Bomboré Gold Project on November 19, 2023.

6) I am responsible for Sections 1.12, 20, 25.6, and 26.6 of the Technical Report.

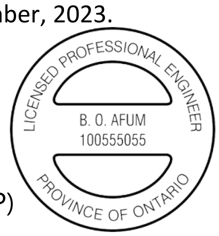
7) I am independent of the issuer as described in section 1.5 of NI 43-101.

8) I have no prior involvement with the properties that are the subject of this Technical Report.

9) I have read NI 43-101, and the parts of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.

10) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for, contain all scientific and technical information that is required to be disclosed and to make the Technical Report not misleading.

Dated this 24th day of November, 2023.



Bright Oppong Afum
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