NI 43-101 TECHNICAL REPORT SULPHIDES UPDATE ON KOBADA GOLD PROJECT IN MALI

Prepared for
AFRICAN GOLD GROUP, INC.

Prepared by
SENET (PTY) LTD

Effective Date
29 September 2021

DATE AND SIGNATURE PAGE

This Report titled "NI 43-101 Technical Report on Kobada Gold Project in Mali" was prepared by SENET (Pty) Ltd (SENET). The Report is compliant with the Canadian National Instrument 43-101 (NI 43-101) and Form 43-101F, and was signed by the following Qualified Persons:

Nicholas Dempers

MSc Eng (Chem), BSc Eng (Chem), BCom (Man), Pr.Eng (RSA), Reg.No 20150196, FSAIMM (RSA)

SENET (Pty) Ltd

29 September 2021

Johannesburg, South Africa

Uwe Engelmann

BSc (Zoo. & Bot.), BSc Hons (Geol.) Pr.Sci.Nat. (Reg. No. 400058/08), MGSSA (No. 966310)

Minxcon (Pty) Ltd

29 September 2021

Roodepoort, South Africa

Stephanus J.P. Coetzee

B.Sc Hons Environmental Management

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29 September 2021

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MSc Civil Eng., BSc Civil Eng. PrEng (RSA)

Epoch Resources

29 September 2021

Johannesburg, South Africa

Ghislain Prevost

P. Eng., B. Mining Eng, M.Sc.A. Mineral Eng. (OIQ # 119054)

DRA Americas

29 September 2021

Montréal, QC, Canada

The effective date of the NI 43-101 Technical Report is 29 September 2021, which is the cut-off date for all scientific and technical information included in the recently completed Kobada Gold Project Definitive Feasibility Study Report.

CERTIFICATE OF QUALIFIED PERSON – NICHOLAS DEMPERS

I, Nicholas Dempers, do hereby certify that

1. I am a principal process engineer at SENET (Pty) Ltd, Building 12, Greenstone Hill Office Park, Emerald Boulevard, Greenstone Hill, Greenstone 1609, Modderfontein, Gauteng, South Africa.
2. I am a reviewer of the report titled "NI 43-101 Technical Report on Kobada Gold Project in Mali" dated 29 September 2021.
3. I am a graduate of the University of Cape Town, with a BSc in Chemical Engineering. I also hold a MSc in Chemical Engineering from the University of Cape Town and a BCom from the University of South Africa.
4. I am a registered professional member of the Engineering Council of South Africa (Reg. No. 20150196), and I am a fellow of the Southern African Institute of Mining and Metallurgy.
5. I have practised my profession continuously since 2001. I have over 19 years' experience in the minerals industry. I have been involved in the process operation (production) and plant design, from conceptualisation to complete project execution, of more than 10 mineral process projects, as well as more than 12 process plant studies for major commodities including cobalt, copper, gold, uranium, rare earths and platinum group metals (PGMs). I have assisted in or compiled National Instrument 43 101 (NI 43-101) Reports for various projects that have been listed on the TSX stock exchange
6. I have read the definition of "qualified person" set out in NI 43-101 and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
7. I have not visited the Kobada Project Site.
8. I performed consulting services and reviewed files and data supplied by AGG between 07/05/2020 and 9/07/2020.
9. I am responsible for the preparation of Sections 13, 17, 18, 21 and 22 and contributed to Sections 1, 24, 25 and 26 of the Technical Report.
10. I have had no previous involvement with this project or any other project on this property.
11. I am independent of AGG as independence is described in Section 1.5 of NI 43-101. I do not have, nor do I expect to receive a direct or indirect interest in the Mineral Properties of AGG and I do not beneficially own, directly or indirectly, any securities of AGG or any associate or affiliate of such company.
12. I have read NI 43-101 and Form 43-101F1, and the part of the Technical Report for which I am responsible has been prepared in compliance therewith.
13. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to ensure that the Technical Report is not misleading.

Signed at SENET (Pty) Ltd, Johannesburg, South Africa on 29 September 2021.

____

NICHOLAS DEMPERS

MSc Eng (Chem), BSc Eng (Chem), BCom (Man), Pr.Eng (RSA), Reg.No 20150196, FSAIMM (RSA)

CERTIFICATE OF QUALIFIED PERSON – UWE ENGELMANN

I, Uwe Engelmann, do hereby certify that

1. I am a Director of Minxcon (Pty) Ltd, Suite 5, Coldstream Office Park, 2 Coldstream Street, Little Falls, Roodepoort, South Africa.
2. I graduated with a BSc Honours (Geology) degree from the University of the Witwatersrand in 1991.
3. I have more than 23 years' experience in the mining and exploration industry. This includes eight years as an Ore Resource Manager at the Randfontein Estates Projects on the West Rand. I have completed a number of assessments and technical reports pertaining to various commodities, including gold, using approaches described by the National Instrument 43-101, *Standards of Disclosure for Mineral Projects*, Form 43-101F1, and the Companion Policy Document 43-101CP (NI 43-101).
4. I am affiliated with the following professional associations, which meet all the attributes of a Professional Association or a Self-Regulatory Professional Association as applicable (as those terms are defined in NI 43-101):

Class	Professional Society	Year of Registration
Member	Geological Society of South Africa (MGSSA No. 966310)	2010
Professional Natural Scientist	South African Council for Natural Scientific Professions (Pr.Sci.Nat. Reg. No. 400058/08)	2008

5. I am responsible for Sections 4, 5, 6, 7, 8, 9, 10, 11, 12, 14 and contributed to Sections 1, 24, 25 and 26 of the Report titled "NI 43-101 Technical Report on Kobada Gold Project in Mali" prepared for AGG with an effective date of 1 July 2021 (the Report).
6. I have read the definition of "Qualified Person" set out in NI 43-101, and certify that by reason of my education, affiliation with professional associations and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of the Report.
7. I have read NI 43-101 and the Report has been prepared in compliance with it.
8. As of the effective date, to the best of my knowledge, information and belief, the Report contains all the scientific and technical information required to be disclosed to make the Report not misleading.
9. I am independent of AGG as such term is defined in Section 1.5 of NI 43-101. My compensation, employment or contractual relationship with the Commissioning Entity is not contingent on any aspect of the Report.
10. I was critically involved in the planning and execution of the infill drilling campaign undertaken during 2018 and 2019/2020.
11. I undertook personal inspections of the subject properties in June 2018, August 2019 and November 2019 to inspect and oversee the drilling campaign.

Signed at Little Falls, Roodepoort on 01 July 2021.

UEngelmann

UWE ENGELMANN
BSc (Zoo. & Bot.), BSc Hons (Geol.), Pr.Sci.Nat., MGSSA

CERTIFICATE OF QUALIFIED PERSON – GHISLAIN PRÉVOST

I, Ghislain Prévost, do hereby certify that

1. I am employed by DRA Americas in the role of Senior Mining Engineer, with my office located at 555 René Lévesque West, 6th Floor, Montreal, Quebec Canada H2Z 1B1.
2. I am a graduate of École Polytechnique de Montréal, Montréal, Canada with a Bachelor of Mining Engineering in 1996 and a Master Degree of Mineral Engineering in 1999.
3. I am registered as a Professional Engineer in the Province of Quebec (Reg. # 119054).
4. I have worked as a Mining Engineer in various capacities since my graduation from university in 1999.
5. My relevant work experience includes:
    a. Design, scheduling, cost estimation and Mineral Reserve estimation for several open pit studies.
    b. Technical assistance in mine design and scheduling for mine operations in Canada, Asia, South America and Africa.
6. I have read the definition of "qualified person" set out in the National Instrument 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be an independent qualified person for the purposes of NI 43-101.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have participated in the preparation of this Technical Report and am responsible for Sections 15 and 16 relating to the Mineral Reserves Estimate and Mining Methods and parts of Sections 1, 21 and 25.2.
9. I did not visit the property site that is subject to the Technical Report.
10. I have had no prior involvement with the property that is the subject of the Technical Report.
11. I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this Report.
12. Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity, or employee of AGG, or any associated or affiliated entities.
13. Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of AGG, or any associated or affiliated companies.
14. Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three (3) years from AGG, or any associated or affiliated companies.
15. I have read NI 43-101 and Form 43-101F1 and have prepared the Technical Report in compliance with NI 43-101 and Form 43-101F1; and have prepared the report in conformity with generally accepted Canadian mining industry practice, and as of the date of the certificate, 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.

Signed at Quebec, Canada on this 12th day of August 2021.

INGÉNIEUR · ENGINEER
Ghislain Prévost
119054
QUÉBEC

GHISLAIN PRÉVOST
P. Eng., B. Mining Eng, M.Sc.A. Mineral Eng.

CERTIFICATE OF QUALIFIED PERSON – STEPHANUS JP COETZEE

I, Stephanus JP Coetzee, do hereby certify that

1. I am a director at Advisory on Business and Sustainability Africa (Pty) Ltd (ABS Africa), Unit 2 Block C, Carlswald Close Office Park, Carlswald, Midrand, South Africa.
2. I am a co-author of the report titled "NI 43-101 Technical Report on Kobada Gold Project in Mali" dated 08 September 2021.
3. I am a graduate of University of the North West (previously Potchefstroom University for Christian Higher Education), with a B.Sc. Honours Degree in Environmental Management.
4. I am a registered professional member of the Registered Professional Natural Scientist (Pr Sci.Nat) with the South African Council for Natural Scientific Professions (SACNASP), Registration Number 40044/04.
5. I have practised my profession continuously since 1997, and I have experience in environmental science and environmental management. I have assisted in or compiled National Instrument 43 101 (NI 43-101) Reports for various projects that have been listed on the TSX, JSE and AIM stock exchanges.
6. I have read the definition of "qualified person" set out in NI 43-101 and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
7. I visited the Project Site on 29 – 31 October 2019.
8. I performed consulting services and reviewed files and data supplied by AGG between 22/10/2019 and 08/09/2021.
9. I am responsible for the preparation of Sections 20 and contributed to sections 1, 11, 12 and 17.1.6 of the report titled "NI 43-101 Technical Report on Kobada Gold Project in Mali".
10. I have had no previous involvement with this project or any other project on this property.
11. I am independent of AGG as independence is described in Section 1.5 of NI 43-101. I do not have, nor do I expect to receive a direct or indirect interest in the Mineral Properties of AGG and I do not beneficially own, directly or indirectly, any securities of AGG or any associate or affiliate of such company.
12. I have read NI 43-101 and Form 43-101F1, the part of the technical report for which I am responsible has been prepared in compliance therewith.
13. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to ensure that the Technical Report is not misleading.

Signed at Midrand, Johannesburg, South Africa on 08 September 2021.

STEPHANUS JP COETZEE
B.Sc. Hons. Environmental Management

CERTIFICATE OF QUALIFIED PERSON – GUY JOHN WIID

I, Guy John Wiid, of Epoch Resources, 8 Viscount Road, Bedfordview Johannesburg, South Africa, do hereby certify that

1. I am a reviewer of the technical report titled "NI 43-101 Technical Report on Kobada Gold Project in Mali", with an effective date of 29 September 2021.
2. I graduated with a BSc Eng (Civil) from The University of the Witwatersrand in 1988. I also obtained an MSc Eng (Civil) from the University of the Witwatersrand in 1995.
3. I am a Registered Professional Engineer with the Engineering Council of South Africa, Registration Number 940269.
4. I have worked as an engineer in the field of mining waste management and mine closure, continuously, for a total of 29 years since my graduation in 1988. I have been directly involved in the design, construction, and ongoing monitoring of tailings dams.
5. I have read the definition of 'qualified person' set out in National Instrument 43-101 (the Instrument) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a 'qualified person' for the purposes of the Instrument.
6. I have not visited the Kobada Gold Project.
7. I am responsible for the review of Sections 1, 18.1.11, 24, 25 and 26 of the Report.
8. I am independent of the issuer as defined in Section 1.5 of the Instrument.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read the Instrument and Form 43-101F1, and the part of the Technical Report for which I am responsible has been prepared in compliance with that instrument and form.
11. At the effective date of this Technical Report, to the best of my knowledge, information and belief, the part of the Technical Report for which I am responsible contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Epoch Resources, 8 Viscount Road, Bedfordview Johannesburg, South Africa on 29 September 2021.

GJWiid

GUY JOHN WIID
Professional Engineer and Director

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Unit	Description
%	per cent
% m/m	percentage mass per mass
% w/w	percentage weight per weight
µg	microgram
µm	micrometre (micron)
µS	microsiemen
°C	degree Celsius
a	annum
cm	centimetre
d	day
dB	decibel
g	gram
g/t	gram per tonne
Ga	billion years (10^9 years)
h	hour
ha	hectare
Hz	hertz
kg	kilogram
km	kilometre
km^2	square kilometre
koz	thousand ounces
kV	kilovolts
kVA	kilovolt amperes
kW	kilowatt
kWe	kilowatt
kWh	kilowatt hour
L	litre
m	metre
mamsl	metres above mean sea level
min	minute
mm	millimetre
Moz	million ounces
MPa	megapascal
Mt	million tonnes
MW	megawatt
N	newton

Unit	Description
Nm	newton metre
NTU	Nephelometric Turbidity Unit
oz	troy ounce
Pa	pascal
Pas	pascal second
ppb	part per billion
ppm	part per million
s	second
s^{-1}	reciprocal second
t	metric tonne
V	volt

It is noted that, throughout the report, table columns might not add up due to rounding.

LIST OF ABBREVIATIONS

Abbreviation	Description
AACE	Association for the Advancement of Cost Engineering
AAS	Atomic absorption spectroscopy
ABA	Acid-base accounting
ADR	Adsorption, desorption, and recovery
AEP	Annual exceedance probability
AGG	African Gold Group Inc.
Ai	Abrasion index
AISC	All-In Sustaining Costs
AMD	Acid mine drainage
ANFO	Ammonium nitrate fuel oil
AQG	Air quality guideline
BBWi	Bond ball work index
BESS	Battery energy storage system
DFS	Definitive feasibility study
BHID	Drillhole identification number
BOQ	Bill of quantities
BRGM	Bureau de Recherches Géologiques et Minières (Geological Survey of France)
BRWi	Bond rod work index
C&I	Control and instrumentation
CAPEX	Capital cost
CBA	Core bedding angle
CCR	Central control room
CCTV	Closed-circuit television
CDA	Canadian Dam Association
CDP	Community Development Plan
CIL	Carbon in leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	Carbon in pulp
CIS	Carbon in solution
CMMS	Computerised maintenance management system
COD	Chemical oxygen demand
COG	Cut-off grade
CRIRSCO	Committee for Mineral Reserves International Reporting Standards
CRM	certified reference material
CRO	Control room operator

Abbreviation	Description
CWi	Crushability work index
DD	Diamond (drilling)
DNACPN	Direction Nationale de l'Assainissement et du Contrôle des Pollutions et des Nuisances (National Department of Sanitation and Pollution and Nuisance Control)
DTM	Digital terrain model
DUFCF	Discounted unlevered free cash flow
EC	Electric conductivity
ECSA	Engineering Council of South Africa
EDF	Environmental Design Flood
EDM	Énergie du Mali
EGRG	Extended gravity recoverable gold
EHS	Environmental health and safety
EMOP	L'Enquête Modulaire et Permanente auprès des Ménages
EOP	End of period
EP	Equator Principle
EPC	Engineering, procurement, and construction
EPCM	Engineering, procurement, and construction management
ERP	Enterprise resource planning
ESDCO	Environment and Social Development Company
ESIA	Environmental and Social Impact Assessment
ESMP	Environmental and Social Management Plant
EW	Electrowinning
FA	Fire assay
FEL	Front-end loader
FoS	Factor of safety
FS	Feasibility Study
G&A	General and administration
GIIP	Good International Industry Practice
GIS	Geographic information system
GRG	Gravity recoverable gold
GSI	Geological strength index
GSSA	Geological Society of South Africa
HAZMAT	Hazardous material
HAZOP	Hazard and operability
HDPE	High-density polyethylene
HFO	Heavy fuel oil

Abbreviation	Description
HIV/AIDs	Human immunodeficiency virus, acquired immunodeficiency syndrome
HL	Heap leach
HR	Human resources
HSSEC	Health, safety, security, environmental and community
IBC	Intermediate bulk container
ICMI	International Cyanide Management Code
ICOLD	International Commission on Large Dams
ICP-MS	Inductively coupled plasma mass spectrometry
IDF	Inflow Design Flood
IEC	International Electrotechnical Commission
IEM	Iowa Environmental Mesonet
IFC	International Finance Corporation
ILR	Intensive leach reactor
I/O	Input/output schedules
IPP	Independent Power Producer
IRR	Internal rate of return
ISCP	Impôt Spécial sur Certains Produits (special tax on certain products)
ISO	International Organization for Standardization
ISRM	International Society for Rock Mechanics
IT	Information technology
IT	Interim target
KNA	Kriging Neighbourhood Analysis
LAeq, d	Equivalent continuous sound level, daytime
LAeq, n	Equivalent continuous sound level, night-time
LAN	Local area network
LBMA	London Bullion Market Association
LOM	Life of mine
LOWL	Low operating water level
LW	Leachwell
M&I	Measured and Indicated (resource)
MCC	Motor control centre
MEL	Mechanical equipment list
MMI	Mobile metal ion
MMMA	Mine Metallurgical Managers Association
MMS	Maelgwyn Mineral Services Africa
MOC	Mine operating cost
MTO	Material take-off

Abbreviation	Description
NAAQS	National Ambient Air Quality Standards
NAG	Net acid generation
NAPP	Net acid production potential
NF	Norme Française (French Standard)
NGL	Natural ground level
NI 43-101	Canadian Securities Administrators' National Instrument (NI) 43-101
NOWL	Normal operating water level
NPV	Net present value
NR	Noise receptor
OCHA	Office for the Coordination of Humanitarian Affairs
OEM	Original equipment manufacturer
ON/AN	Oil natural/air natural
OHMS	Open House Management Solutions
OK	Ordinary Kriging
OMC	Orway Mineral Consultants
OPEX	Operating cost
ORP	Operational Readiness Plan
P&G	Preliminary and general
P&ID	Piping and instrumentation diagram
P&S	Peacocke & Simpson
PAS	Process automation system
PDC	Process design criteria
PFD	Process flow diagram
PGA	Peak ground acceleration
PID	Proportional integral derivative
PLC	Programmable logic controllers
PM	Particulate matter
PMF	Probable maximum flood
PMP	Probable maximum precipitation
PPA	Power purchase agreement
PSD	Particle size distribution
PV	Photovoltaic
QA	Quality assurance
QC	Quality control
R&R	Rest and relaxation
RAP	Resettlement Action Plan
RC	Reverse circulation (drilling)

Abbreviation	Description
ROM	Run of mine
RoR	Rate of rise
RPEEE	Reasonable prospects of eventual economic extraction
RWD	Raw water dam
SABS	South African Bureau of Standards
SACNASP	South African Council for Natural Scientific Professions
SAICE	South African Institution of Civil Engineering
SAIMM	Southern African Institute of Mining and Metallurgy
SANAS	South African National Accreditation System
SANS	South African National Standard
SCADA	Supervisory control and data acquisition
SFA	Screen Fire Assay
SG	Specific gravity
SGS	SGS Lakefield
SHEQ	Safety, health, environmental and quality
SIB	Stay in business
SLD	Single-line diagram
SMBS	Sodium metabisulphite
SMPP	Structural, mechanical, plate work and piping
ST-Lab	Specialised Testing Laboratory (Pty) Ltd
STATSA	Statistics South Africa
SWD	Storm water diversion
TCT	Total concentration threshold
TD	Tailings dam
TDS	Total dissolved solids
TSF	Tailings storage facility
TSS	Total Suspended Solids
UCS	Uniaxial compressive strength
UTM	Universal Transverse Mercator
VSD	Variable speed drive
WAC	West African Craton
WAD	Weak acid dissociable (cyanide)
WAEMU	West African Economic and Monetary Union
WAN	Wide area network
WHO	World Health Organisation
WRD	Waste rock dump
XRF	X-ray fluorescence

1 SUMMARY

1.1 INTRODUCTION

This updated NI 43-101 Technical Report was compiled by NEW SENET (PTY) Ltd (SENET) for African Gold Group Inc. (AGG) with contributions from the Qualified Persons as set out in Table 1.1 to support AGG's press release dated 29 September 2021 and to summarise the results of the updated definitive feasibility study (DFS) of the Kobada Gold Project.

Table 1.1: Qualified Persons and their Contributions

Qualified Person	Company	Contribution
Uwe Engelmann	Minxcon Group (South Africa)	Geology and mineral resources
Ghislain Prévost	DRA Americas (Canada)	Mining, mineral reserves
Stephanus JP Coetzee	ABS Africa (South Africa)	Environmental and social assessment
Guy Wiid	Epoch Resources (South Africa)	Tailings facilities
Nicholas Dempers	SENET (South Africa)	Metallurgical test work interpretation Processing plant and project infrastructure Economic evaluation Coordination and compilation of Report

The Kobada Gold Project is located in southern Mali, approximately 126 km in a straight line south-southwest of the capital city, Bamako, and is situated adjacent to the Niger River and the international border with Guinea.

The Kobada Gold Project is based on one mining permit of 136 km^2 and one exploration permit of 80 km^2, which are wholly owned by AGG Mali SARL, the local Malian Company, which is a 100 % owned subsidiary of AGG.

The Kobada Gold Project is based on one mining permit (Kobada, No. PE 15/22) and two exploration permits (Kobada-Est, No. PR 18/957 and Faraba, No. PR 17/921), wholly owned by AGG Mali SARL, the local Malian Company, which is a 100 % owned subsidiary of AGG.

AGG completed 114 357 m of diamond, reverse circulation, and air core between 2005 and 2012. In 2015, AGG completed a further 1 398 m of diamond core drilling over 13 diamond drill holes. The recent AGG exploration completed between 2018 and 2020 added an additional 21 686 m of diamond and reverse circulation drilling.

Gold mineralisation is present in the laterite, saprolite, and quartz veins that comprise the project, and in the sulphidic hard rock underneath. There are also placer-style deposits in the region.

The Kobada Gold Project, when completed, will produce 100 000 oz/a of gold via a conventional gravity carbon-in-leach (CIL) process designed to treat 3 Mt/a of run of mine (ROM) for the first 10 years of production, followed by low gold production as a result of treating low grade ore.

This Report sets out the Mineral Resource Estimate, Mineral Reserve Estimate, production schedules, and the capital and operating expenditures over the life of the Project. The Report culminates in a full economic analysis of the Project's value.

1.2 QUALIFICATIONS OF QUALIFIED PERSONS

The relevant sections of the NI 43-101 Technical Report were compiled by the consultants' Qualified Persons, as this term is defined in NI 43-101. The certificates of the Qualified Persons (QPs) follow the Date and Signature Page of this Report. A summary of their qualifications and responsible sections, and whether they conducted site visits or not, is given in Table 1.2.

Table 1.2: Summary of the Qualifications and Responsibilities of the QPs

QP	Qualification	Company	Site Visit	Responsibility (Section of Report)
Nicholas Dempers	MSc Eng (Chem), BSc Eng (Chem), BCom (Man), Pr.Eng (RSA), Reg.No. 20150196, FSAIMM (RSA)	SENET	No	1,13, 17, 18, 21 and 22, 24, 25 and 26
Uwe Engelmann	BSc (Zoology & Botany), BSc (Geology), BSc Honours (Geology)	Minxcon	Yes	4, 5, 6, 7, 8, 9, 10, 11, 12, 14, and parts of 1, 24, 25 and 26
Guy Wiid	MSc Civil Eng, BSc Civil Eng	Epoch	No	1, 18.1.11, 24, 25 and 26
Staphanus JP Coetzee	B.Sc Hons Environmental Management	ABS Africa	Yes	20 and contributed to sections 1, 24, 25 and 26
Ghislain Prévost	MSc, P.Eng (OIQ # 119054))	DRA Americas	No	15 and 16 ,1, 21 and 25 to 27

1.3 PROPERTY DESCRIPTION

Kobada is an advanced exploration project in Mali, targeting primary lode gold mineralisation and secondary mineralisation in laterite. The Project comprises the targets Kobada Main Shear Zone, Kobada West, Foroko, Foroko North, Diaban, Gosso and Siramana. The Project is located in southern Mali, approximately 126 km due southwest of the country capital, Bamako, as shown in Figure 1.1.

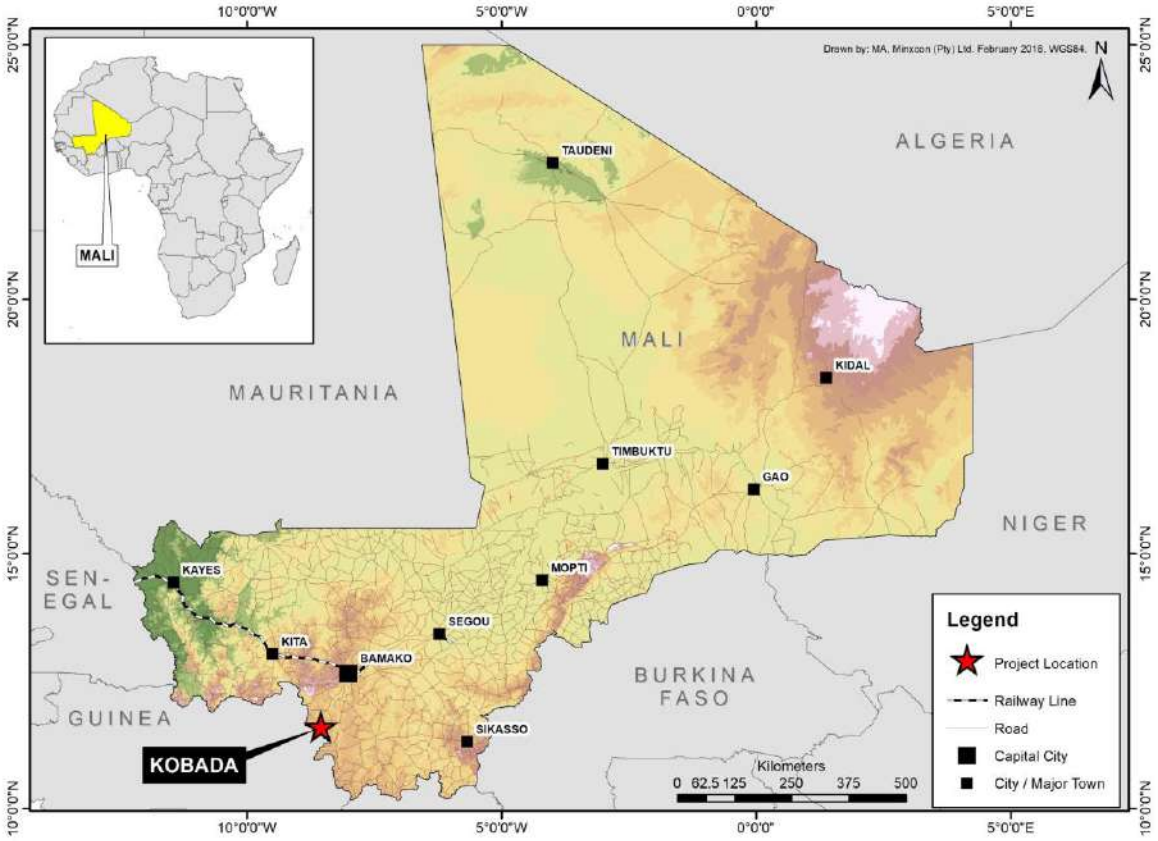


Figure 1.1: Project Location

1.4 OWNERSHIP OF THE PROPERTY

The Project Area is held under a mining permit (Kobada, No. PE 15/22) and two exploration permits (Kobada-Est, No. PR 18/957 and Faraba, No. PR 17/921), issued to AGG Mali SARL and shown in Figure 1.2. AGG Mali SARL is a wholly owned subsidiary of AGG through AGG (Barbados) Limited. The mineralised target areas under consideration are restricted to the boundaries of the mining permit.

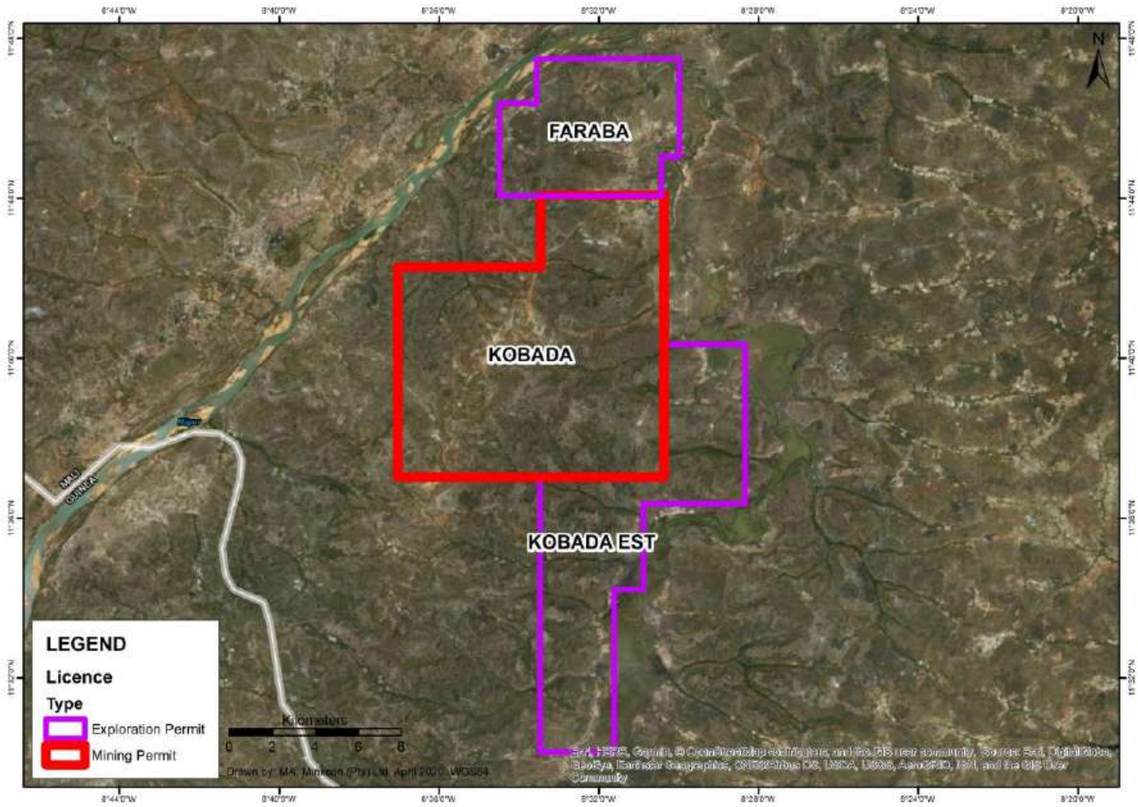


Figure 1.2: Mining and Exploration Permit Areas

1.5 GEOLOGY AND MINERAL DEPOSIT

The Project is located in the Bagoe Formation on the north-central edge of the Birimian rock units that form part of the Leo Rise in the southern part of the West African Craton. The Leo Shield is composed of the Keneman-Man Domain in the southwest, and the Proterozoic Baoule-Moussi Domain. The Baoule-Moussi contains relics of Archaean rocks and Paleoproterozoic Birimian formations. The Birimian consists of narrow elongated belts of mainly epi-metamorphosed volcano-sedimentary formations deformed by a number of regional events. Regional metamorphism reaches greenstone facies with amphibolite facies restricted to the intrusive granitoids contact.

The Project is situated on the western flank of the Bougouni Basin, composed primarily of sedimentary rocks with minor tholeiitic volcano-sedimentary intercalations. The Bougouni Batholith appears approximately 25 km northeast and southwest of the Project Area. Gold at Kobada is present in the laterite, saprolite and quartz veins.

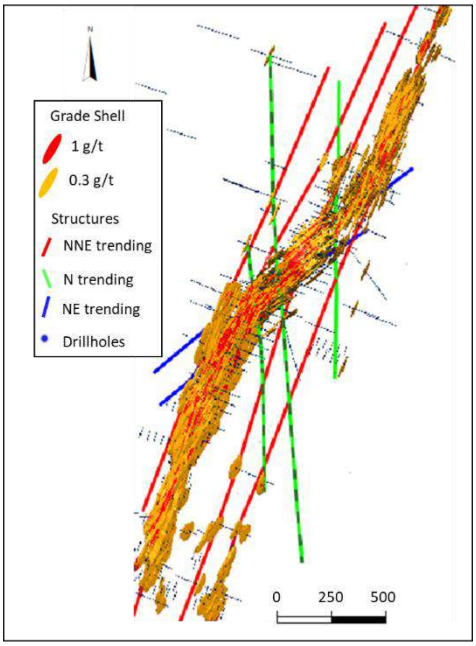
The terrane is intensely lateritised, with large laterite plateaus covering most of the area. The underlying saprolite is exposed below the plateau boundaries. Rare outcrops occur where resistive quartz veins protected the host rocks, and the schistose nature of the sediments can then be observed. Drilling intersected interbedded greywackes, siltstones and mudstones.

The saprolite shows slight variation from what is now a clay (mudstone precursor) to a fine silty clay (fine siltstone precursor). There are no marker horizons, and no sedimentary features are preserved. The deformation intensity of these metasediments is moderate. Regional foliation is moderate and often not recognised in the saprolite, and while shear zones occur at Kobada, these tend to be discrete structures 5 cm to 50 cm wide. The laterite horizon is typically 3 m to 4 m thick and generally presents a stark contrast to the saprolite.

The veins occur as quartz-carbonate veined mesothermal, orogenic gold hosted within a greenstone belt. They are located in arenites affected by a geological structure that is oriented northeast along the border of an intermediate intrusive that has basic components. Veins are usually less than 2 m wide and often occur in parallel sets.

1.5.1 Property Geology

A geological model representing the grade distribution and trend was generated at a 0.3 g/t cut-off. These grade shells defined the domains that were utilised to estimate the Mineral Resources. The trend and orientation of the shells were guided by existing structural knowledge and work that was performed on the property. The structural model on which the modelling process was based is a dextral shear system with the principal shear orientations trending NNE, roughly E-W striking Riedel shears and extensional veins, and approximately north-striking conjugate sets (see Figure 1.3). In addition, a weathering profile was defined over the orebody to represent the weathering intensity that was observed in the core. New drillholes drilled in 2019 and 2020 corresponded well with the grade and orientation of the mineralisation that was modelled from historical data and served to confirm the geological interpretations and structural trends observed over the orebody.



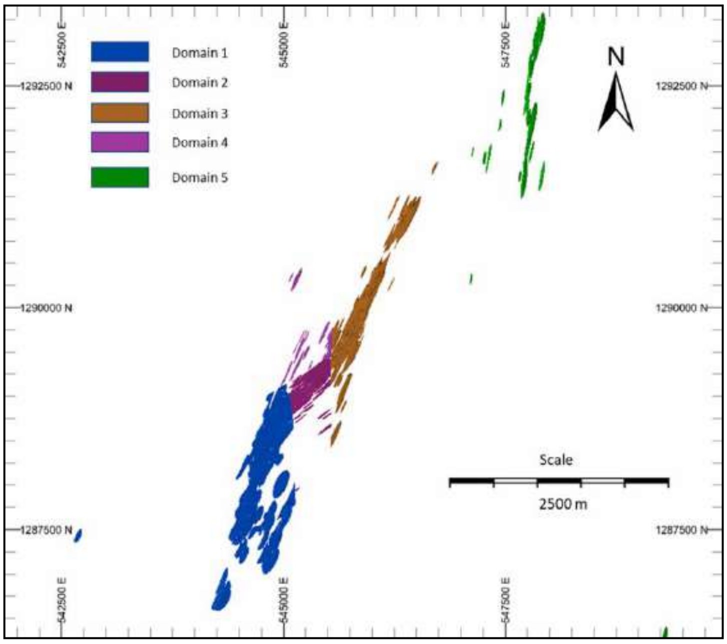


Figure 1.3: Structural Model and Domains

1.5.2 Mineral Resource Estimates

The updated Mineral Resource estimation was conducted using Ordinary Kriging based on best practice techniques and utilising all the historical and recent drilling from 2020. The Mineral Resources are declared only within the Mineral Resources optimised pit (applying reasonable prospects of eventual economic extraction (RPEEE)) based on a gold price of US$1,800/oz. The economic cut-off grade that has resulted from the pit optimisation is 0.25 g/t. However, the cut-off grade that has been applied to the updated Mineral Resources has remained unchanged at 0.35 g/t.

The Mineral Resources have been depleted by means of the new digital terrain model (DTM) to account for artisanal mining operations. An additional 1 % tonnage loss has been applied to the oxides/saprolites to account for artisanal mining losses not accounted for by the new DTM. Further discounts applied to the Mineral Resources include geological losses of 5 % for the Measured Mineral Resources, 10 % for the Indicated Mineral Resources, and 15 % for the Inferred Mineral Resources.

The gold content conversion calculations utilise a conversion of 1 kg = 32.15076 oz and all tonnages are reported in metric tonnes.

Inferred Mineral Resources have a low level of confidence, and while it would be reasonable to expect that the majority of the Inferred Mineral Resources would be upgraded to Indicated Mineral Resources with continued exploration, due to the uncertainty of the Inferred Mineral Resources, it should not be assumed that such upgrading will occur.

Table 1.3 shows the Kobada Project Mineral Resources as at 1 July 2021.

Table 1.3: Kobada Project Mineral Resources as at 1 July 2021

Mineral Resource Classification	Tonnes	Tonnes Less Geological Loss	Au	Au	Au
	Mt	Mt	g/t	kg	koz
Measured	22.65	21.40	0.83	17,784	572
Indicated	44.81	40.15	0.88	35,425	1,139
Measured and Indicated Total	**67.46**	**61.54**	**0.86**	**53,209**	**1,711**
Inferred Total	**49.60**	**42.03**	**1.06**	**44,564**	**1,433**

NOTES:
1. A Mineral Resource cut-off grade of 0.35 g/t Au was applied.
2. A gold price of US$1,800/oz was used for ultimate optimisation.
3. Columns might not add up due to rounding.
4. Mineral Resources are stated as inclusive of Mineral Reserves.
5. Mineral Resources are reported as total Mineral Resources and are not attributed.
6. Geological losses have been applied.

1.5.3 Status of Exploration

The Kobada Project has been explored since 1988 by various companies, with most of the exploration being completed by AGG from 2005 to 2012, with additional drilling campaigns in 2015, 2018, 2019 and recently in 2020. In total, 1,609 drillholes have been drilled with the

2020 infill drilling campaign, which was an extension of the 2019 Phases 1 and 2, contributing 43 of these drillholes (Phase 3 = 4 drillholes at the Gosso Target and Phase 4 = 39 drillholes in the gap area and the northern domain of the Kobada Main Shear). The recent 2020 drilling was undertaken to test the Gosso Target, to improve and confirm the understanding of the geological model, and to improve the Mineral Resource estimation in conjunction with converting some of the Mineral Resources to the more confident Mineral Resource categories.

There is still an upside potential at the Kobada Main Shear to upgrade some additional Inferred Mineral Resources to Indicated Mineral Resources. In addition to this, the Kobada Project has a significant upside potential in the 55 km strike of potential mineralised shear zones. These shear zones are shown in Figure 1.4. Of these, the Gosso Target is the more advanced, with limited drilling completed (21 drillholes). The latest 2020 drilling (4 diamond drillholes) confirmed the mineralisation at this target. Initial field investigations by the AGG geologists, in early 2021, have highlighted the potential at the Kobada Est targets where artisanal mining has exposed mineralised structural features.

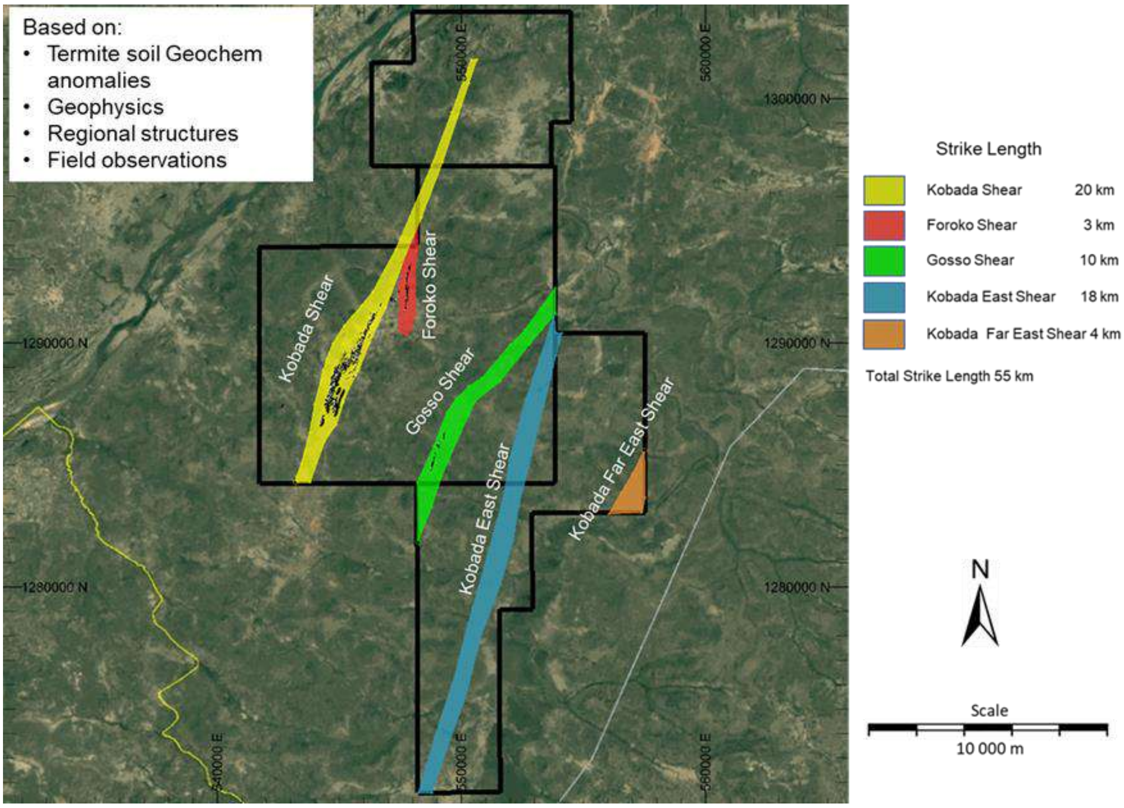


Figure 1.4: Further Exploration Potential

1.5.4 Development and Operations

The Kobada Gold Project is an advanced exploration project and is currently not in operation. The new camp is, however, already in the construction phase in preparation for the future operation.

1.5.5 Conclusions

It is the opinion of Mr Uwe Engelmann, the QP responsible for the Mineral Resource Estimate, that the database is reliable enough and sufficient to support the estimation of a Mineral Resource. There is a possibility that the Mineral Resource grade may be understated as a result of the analytical methodology, i.e. Leachwell, which is a partial gold and not total gold analysis. It is therefore recommended that the Leachwell analysis be followed by a fire assay on the tails to achieve a larger sample size and obtain a total gold grade. An exercise comparing the pre-2018 data set (predominantly Leachwell) to the post-2018 data set (diamond drilling) has shown a 9 % difference (the Leachwell sampling grade was higher), and an exercise comparing the Leachwell only with total gold has shown a difference of 9 %. Based on the current data, it is not possible to reliably quantify what that understatement is. As an indicative range, the above exercises would suggest that it is between 5 % and 10 %. This could be seen as a possible upside potential, i.e. the recovered grade could be higher.

The updated geological model, which forms the basis of the Mineral Resource estimation, utilises more geological parameters than the historical model, based on the additional information gathered during the recent 2019 and 2020 drilling campaign. The new model is better understood and has increased confidence. The recent 2019 and 2020 drilling campaigns have not only confirmed the updated geological model, however, have also resulted in an increase in the Mineral Resource.

There is still a potential to increase the Measured and Indicated Mineral Resource even further at the Kobada Main Shear, with additional infill drilling in the Inferred Mineral Resource, especially for the sulphides, which have only recently received focus. New focus on the deeper sulphides may extend the life of the potential mine even further.

Furthermore, another upside potential exists at the Kobada Project to increase the Mineral Resource significantly by exploring the targets further and converting the exploration targets to Mineral Resources with drilling.

1.5.6 Recommendations

Although the gap area and the northern domain of the Kobada Main Shear were drilled in the 2020 drilling campaign, additional drilling is required at the Kobada Main Shear to convert more of the remaining Inferred Mineral Resources into Indicated Mineral Resources, especially for the deeper sulphides, which make up 850 koz of the 1.4 Moz Inferred Mineral Resource. The lateral extensions of the Kobada Main Shear can also be explored for possible extensions.

Recent field work and drilling have highlighted the significant potential at the Gosso Target as well as the Kobada Est target areas. These should be explored further with a reconnaissance trenching and drilling campaign to optimise and prioritise the exploration targets for future additional Mineral Resources for the Kobada Project.

1.6 MINERAL RESERVES AND MINE PLANNING

1.6.1 Pit Optimisation

The Kobada Open-Pit Project shows an approximately 16-year life-of-mine (LOM) operation including the pre-production period to support a nominal mill feed of 3 Mt/a ROM.

The mineralised material contained within the final pit design has been determined based on a 0.35 g/t gold cut-off grade (COG). All Inferred Resource material are treated as waste in the pit optimisation and in the Mine Schedule. The mill feed is made up completely of Measured and Indicated Resource materials (laterite, saprolite, transition and sulphide) above the calculated COG.

Open-pit optimisation was conducted on the deposit to determine the economic limits of the pit. Table 1.4 presents the pit optimisation parameters used during the DFS of the Kobada Project.

Table 1.4: Pit Optimisation Parameters

Description	Unit	Value
Mining Cost (Laterite and Saprolite)	US$/t	2.50
Mining Cost (Transition and Sulphide)	US$/t	3.00
Processing Cost (Laterite and Saprolite)*	US$/t	9.41
Processing Cost (Transition)*	US$/t	12.64
Processing Cost (Sulphide)*	US$/t	15.08
Mining Recovery	%	95.0
Mining Dilution	%	5.0
Process Recovery (Laterite and Saprolite)	%	96.5
Process Recovery (Transition)	%	90.5
Process Recovery (Sulphide)	%	95.4
Gold Price	US$/oz	1,610
Refining Cost	US$/oz	7.59
Royalties	%	3.0
Overall Pit Slope	Degrees	40
* This cost includes the G&A and tailings operating costs.		

Any inaccuracy in separating the mineralised material and the waste material was accounted for by assuming a mining recovery of 95 % and a mining dilution of 5 %.

1.6.2 Mine Design

The Kobada Gold Mine lends itself to a standard open-pit mining method using articulated trucks and hydraulic loaders (hydraulic shovel or excavator).

The main production schedule consists of 6 months of pre-production stripping followed by 10 years of open-pit production, and an additional 5 years dedicated to stockpile rehandling, for a total LOM of approximately 16 years.

The first six months will be exclusively dedicated to pre-production stockpiling of ore without feeding the process plant.

From Year 1 to Year 10, the process plant will be fed with material from the pit and supplemented by the ROM stockpile.

From Year 11 to Year 15, the process plant will be exclusively fed with rehandled stockpile material using a front-end loader and articulated trucks.

The key design criteria used to develop the pit design are summarised as follows:

- Nominal bench height: 5 m, double benched, when possible, for final pit walls
- Overall slope angle: 40°
- Road width: 15.6 width to accommodate two-way traffic
- Ramp gradient: 10 % or less
- Working areas: a minimum of three independent working areas, based on the mine schedule

1.6.3 Mineral Reserves

A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource, including diluting materials.

A Probable Mineral Reserve is the economically mineable part of an Indicated and, in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower than that applying to a Proved Mineral Reserve.

A Proved Mineral Reserve is the economically mineable part of a Measured Mineral Resource and implies a high degree of confidence in the Modifying Factors.

Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors. Table 1.5 presents the mineral reserve estimate.

Table 1.5: Kobada Mineral Reserves Estimate

Category	Material	Ore (kt)	Grade (g/t)	In Situ Ounces (koz)	Recovered Ounces (koz)
Proven	Laterite	336	0.77	8.4	8.1
	Saprolite	12,279	0.84	326.9	315.4
	Transition	1,777	0.83	46.8	42.4
	Sulphide	5,870	0.74	139.6	133.2
Subtotal		**20,263**	**0.81**	**521.7**	**499.1**
Probable	Laterite	1,296	0.88	36.7	35.4
	Saprolite	16,123	0.91	468.2	451.8
	Transition	1,979	0.90	56.6	51.2
	Sulphide	5,495	0.94	165.1	157.5

Category	Material	Ore (kt)	Grade (g/t)	In Situ Ounces (koz)	Recovered Ounces (koz)
Subtotal		**24,893**	**0.91**	**726.6**	**696.0**
Total Proven and Probable	Laterite	1,633	0.85	45.1	43.5
	Saprolite	28,402	0.88	795.1	767.3
	Transition	3,756	0.87	103.4	93.6
	Sulphide	11,365	0.84	304.8	290.7
Total		**45,139**	**0.87**	**1,248.4**	**1195.1**

Footnotes:
1. The effective date of the Mineral Reserve Estimate is July 28, 2021.
2. Mineral Reserves are reported in accordance with CIM guidelines.
3. A marginal cut-off of 0.35 g/t Au for all material is applied.
4. Mineral Reserves were estimated at a gold price of $1,610 per oz and include modifying factors related to mining cost, and dilution and recovery, process recoveries and costs, G&A, and royalties.
5. Figures have been rounded to an appropriate level of precision for the reporting of Mineral Reserves.
6. Due to rounding, some columns or rows may not compute exactly as shown.
7. The Mineral Reserves are stated as dry tonnes processed at the crusher. All figures are in metric tonnes.
8. The in situ and recovered ounces are in troy ounces.

1.6.4 Mine Planning

A 16-year LOM plan was developed for the Kobada Project using a combination of software packages, including, Hexagon MinePlan™ and TALPAC 3D. MinePlan was used to determine the ultimate pit limit, develop the extraction schedule as well as pit and stockpile design work while TALPAC 3D was used to determine equipment haulage times.

The final mining plan scheduling exercise was based on the requirement for a minimum recovery of 100 koz of recoverable gold per year for the first ten years. The production schedule consists of 6 months of pre-production stripping followed by 10 years of open-pit production, and an additional 5 years dedicated to exclusive stockpile rehandling, for a total LOM of 16 years.

The mining equipment will consist of 120 t class conventional hydraulic shovels (PC1250 or similar) operating in a back-hoe configuration and 55 t class (Volvo A60H or similar) articulated off-highway trucks hauling on the designed access roads.

Front-end loaders (CAT988 or similar) will be used for the stockpile rehandling and to work directly in the mine if necessary.

The track dozers are assigned to maintain the production areas and waste stockpiles, and to clean up the benches. Wheeled dozers, road graders and water trucks complete the remainder of the auxiliary equipment fleet.

A comprehensive network of haul roads, minor roads, ramps, working areas and waste tipping areas will be maintained at a high standard of road repair by the two motor graders. The graders will concentrate on the main arterial haul roads, whilst the track dozer will clean up the areas around the excavating units and the tipping areas on the waste stockpiles.

1.6.5 Grade Control

It has been assumed that dedicated reverse circulation (RC) drilling and sampling will be used for grade control. This activity will be contracted to a drilling contractor with operations in Mali.

The grade control samples will then be sent to the laboratory located at the process plant and will be prepared and analysed using fire assay methodology.

Data will be processed, and blocks will be marked out as per the grade control guidelines, which will be developed prior to mining.

1.7 METALLURGICAL TEST WORK AND PROCESS DESIGN

1.7.1 Metallurgical Test Work

Gold recovery test work performed on oxide and sulphide ore from the Kobada deposit indicated that both ore types are free milling and respond well to gravity recovery followed by cyanidation achieving overall gold dissolutions above 90 % with low cyanide and lime consumptions. Overall gold dissolution refers to gold going into solution and does not include other losses incurred in the plant during operations. Table 1.6 shows a summary of values achieved via the gravity followed by cyanidation route

Table 1.6: Gold Recovery Values

	Unit	Oxide	Sulphide
Gravity Recovery	% Au	42.7	41.5
Grind to CIL	P_{80} - µm	80 %-150 µm	80 %-75 µm
Overall gold dissolution	% Au	93.55	95.38
Cyanide consumption	kg/t	0.62	0.35
Lime consumption	kg/t	0.82	0.30

Comminution test work indicated that the oxide ore has low Bond Ball work index (1.09 kWh/t). Bond Ball Work index on the sulphide ore (14.80 kWh/t) showed that the ore is medium hard for ball milling. Both the oxide and sulphide ore have low abrasion index values (<0.20 g) showing that both ores have low abrasiveness. Therefore, liner and media wear are not expected to be significant. Crushability work index was low for both the oxide and sulphide. Table 1.7 shows a summary of the comminution results.

Table 1.7: Comminution Results Summary

	Unit	Oxide	Sulphide
BBWi	kWh/t	1.09	14.80
Ai	g	0.1236	0.1716
CWi	kWh/t	6.6	15.0

Thickening and rheology test work performed on the oxide (worst-case scenario) showed that the ore did not settle easily, achieving a flux rate of 0.2 t/h/m^2 and underflow density of 49 %.

An arsenic balance showed that for both the oxide and sulphide ore, an arsenic precipitation plant will be required to satisfy environmental requirements to lower the arsenic levels in the solution tailings to less than 0.1 ppm As.

1.7.2 Flowsheet Development

The Kobada process plant was developed from the interpretation of the results of various test work programmes conducted by Gekko, MSA, SGS Canada laboratories on oxide and sulphide ore. This section describes the process flowsheet development of the plant that will treat 100 % oxide ore and various blends of oxide to sulphide ore.

In developing the process flowsheet, trade-off studies were conducted to optimise the process route selection. Specialised consultants, such as Orway Mineral Consultants (OMC), Peacocke & Simpson (P&S), and Kemix, were engaged to help with the flowsheet development in the areas of their expertise. The trade-off studies and simulations conducted included the following:

- Optimum process route selection (SENET)
- Comminution circuit (OMC)
- Gravity circuit (P&S)
- CIL and elution circuit (Kemix)

The conclusions from these trade-off studies and simulations led to the development of the flowsheet, which includes the following plant areas:

- Single-stage crushing facility utilising a mineral sizer to treat the soft saprolite and laterite ores, three stage crushing for sulphide ore.
- Single-stage ball mill operating in an overflow arrangement with a ball-retaining ring to retain the media and reduce scatting for oxide ore, and an additional secondary mill for sulphide ore
- Gravity recovery and intensive cyanidation
- CIL (six CIL stages with no pre-leach tank to mitigate the preg-robbing properties of the ore)
- INCO air/SO_2 cyanide detoxification, arsenic precipitation and pumping of the detoxified tailings to the dam
- Acid wash
- Elution (pressurised Zadra method)
- Carbon regeneration
- Electrowinning and smelting
- Consumables and reagents
- Air and water services

1.7.3 Process Design

The Kobada gold processing plant is designed to process oxide ores from the three main deposits: the north, south and central ores during the first years of production followed by sulphides when the high-grade oxides are depleted. Towards the end of the mine life (Years 9-16) a mix of oxides and sulphides will be processed from stockpiles. Further exploration upside may result in additional oxides to be treated, which may defer the treatment of sulphides until later in the mine life.

The proposed process plant design is based on a well-proven and established gravity/carbon-in-leach (CIL) technology, which consists of crushing, milling, and gravity recovery of free gold, followed by leaching/adsorption of gravity tailings, elution and gold smelting, and tailings disposal. Services to the process plant will include reagent mixing, storage and distribution, and water and air services.

The plant will treat 3 Mt/a of saprolite/laterite/sulphide ore. The crushing circuit will be constructed in two phases: the first phase will be to treat oxide ore only, followed by a second phase later in the life of mine (LOM) to treat sulphides and/or a blend of sulphides and oxides. When treating oxide ore, only the primary crushing stage will be in use while three crushing stages will be required when treating the sulphide ore.

The milling circuit will also be implemented in two phases: the first phase will consist of a single-stage ball mill to treat oxides while the second phase will consist of an additional secondary mill, which will be installed in the later part of the LOM to treat sulphides. The circuit will consist of milling in closed circuit with a classification cyclone The discharge from the mill will discharge into the cyclone feed sump and will be pumped to the cyclone cluster for classification. A portion of the cyclone underflow will be bled to the gravity circuit for recovery of gravity gold, with the balance gravitating to the ball mill for further size reduction. Gold will be recovered from the gravity concentrate through a combination of intensive cyanidation and electrowinning facilities. The gravity recovery tailings will be transferred back to the mill feed for further gold liberation. Gold that is not gravity recoverable is recovered through the CIL process.

The overflow from the cyclone cluster will feed the six-stage CIL circuit, where gold will be dissolved and adsorbed onto carbon. The Kobada ore exhibits varying degrees of preg-robbing as confirmed by the test work. The preg-robbing nature of the ores negates the use of a pre-leach tank. The resultant CIL tailings slurry will be subjected to a cyanide destruction and arsenic precipitation process, prior to being pumped to the tailing storage facilities.

Loaded carbon from the CIL circuit will be acid-washed prior to elution, followed by reactivation of the eluted carbon. The solution from the elution circuit will be subjected to electrowinning, where gold will be deposited onto cathodes as sludge. Periodically, the sludge will be washed off the cathodes and dried. The dried gold sludge will then be smelted to produce gold bullion, which will be shipped to the refinery.

A simplified flowsheet of the Kobada process plant is shown in Figure 1.5.

A summary of the key process design criteria is given in Table 1.8.

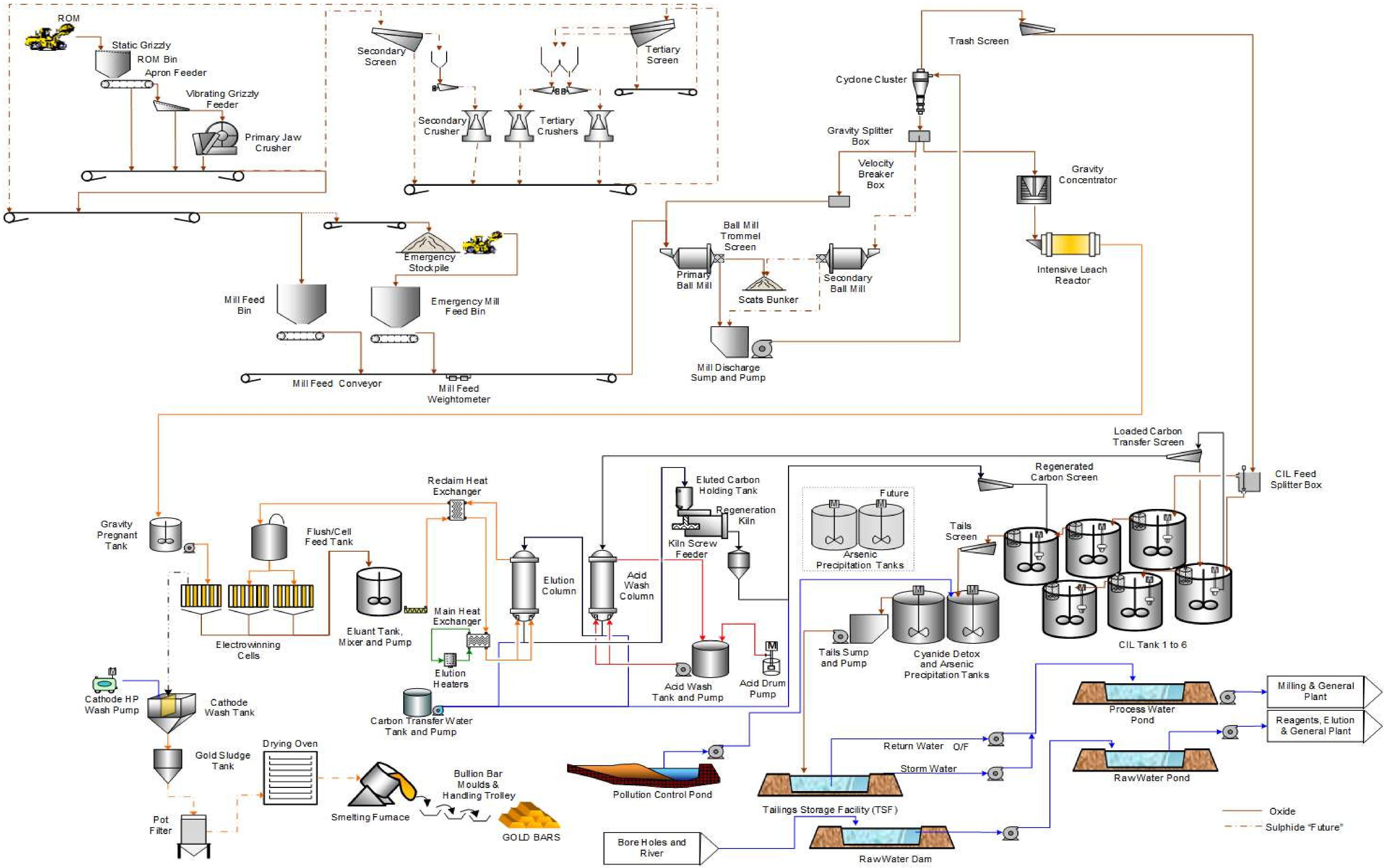


Figure 1.5: Process Flowsheet

Table 1.8: Summary of Key Process Design Criteria

Parameter	Unit	Oxide/Transition	Sulphide
Plant Throughput	Mt/a	3	3
Head Grade: LOM	g/t Au	0.87	1.3
Head Grade: Design	g/t Au	1.36	1.5
Recovery: Oxide (Saprolite/Laterite)	%	96.5	-
Recovery: Transition	%	95	-
Recovery: Sulphide	%	-	97.7
Ai	g	0.103	0.287
CWi	kWh/t	6.6	15
BRWi	kWh/t	6.6	17.8
Scrubber Energy	kWh/t	0.7	
BBWi – Whole Ore	kWh/t	1.1	14.8
BBWi – Whole ±1 mm	kWh/t	12.6	-
Percentage of +1 mm in the ROM Feed	%	55	-
Specific Gravity (SG)		2.5	2.78
Grind Size P_{80}	µm	150	75
Cyclone Overflow Solids	% w/w	36	40
Leach Residence Time	h	16	16
Cyanide Consumption	kg/t	0.7	0.35
Lime Consumption	kg/t CaO	0.6	0.3
Oxygen Uptake	mg/L/min	0.03	Not tested

1.8 TAILINGS STORAGE FACILITY

In 2020, African Gold Group (AGG) commissioned a Definitive Feasibility Study (DFS) of the Kobada Gold Project, under the study lead of SENET. Epoch Resources (Pty) Ltd (Epoch) was in turn appointed by SENET to undertake the DFS design of the Tailings Storage Facility (TSF) associated with Kobada. The facility has since increased in size, and the TSF design has been updated to accommodate the revised required storage capacity, as documented in the DFS report.

The DFS design process is based on the design criteria summarised in Table 1.9 and the following guidelines:

- International Commission on Large Dams (ICOLD), *Tailings Dams Safety* (Draft);
- Canadian Dam Association (CDA), *Dam Safety Guidelines* (2013);
- South African National Standard (SANS) 10286:1998, *Mine Residue*; and
- South African Institution of Civil Engineering – Geotechnical Division, *Site Investigation Code of Practice* (2010).

Table 1.9: Design Criteria for the Kobada TSF

Item	Design Criteria	Value
1	Tailings Material	Gold (Oxides and Sulphides)
2	Design Life of Facility	15.1 years
3	Tailings Deposition Rate	3 million tonnes per annum (Mt/a)
4	Total Tailings Tonnage	45.15 Mt

The geochemistry analysis of the Kobada tailings was conducted on ore samples taken predominantly from the saprolite, with small portions from the laterite and transition zones to mimic the intended mining proportions. The geochemical testing and result analyses were done by Moss Laboratories to include the sulphide material, and the key points were summarised by ABS Africa (Pty) Ltd (ABS Africa), the environment lead, as follows:

> “The majority of the material tested exhibited limited acid generating potential and moderate acid neutralising potential. Therefore, the risk of acidic drainage is low.
>
> All thirty-five samples evaluated contained high to extremely high arsenic concentrations and mobilisation of the arsenic was significant in most cases, even used deionised water as a leach solution. There is a significant risk that leachate arising from the percolation of water through tailings, waste rock impoundments and ore stockpiles could accumulate arsenic to concentrations in excess of the Mali standards.”

SENET has, however, since incorporated measures to precipitate arsenic levels out in the process plant to achieve concentrations below 100 mg/L as per the World Health Organisation (WHO) guidelines.

Based on the above information and the fact that the process includes cyanide (with an associated cyanide treatment plant), the decision to HDPE line the TSF was accepted as a best practice approach.

The arsenic concentration of the sulphides material significantly exceeds the test results of the oxide material tested during the 2020 DFS. The sulphide waste material would therefore not be used for construction of the TSF embankment.

Within the mine lease area, four sites were identified as being potentially suited for the development of the TSF storing 16 Mt of tailings. The selected site was then revised to store 26 Mt for the DFS completed in 2020, documented in the Kobada TSF DFS Report (2020), and thereafter revised to 45 Mt, on which the current 2021 DFS is based.

The site selection has been reassessed to reaffirm that the selected site location (Option 8) is still favourable for the larger facility (storing 45 Mt of tailings). Although Option 8 has more resettlement aspects compared to the other two feasible sites, it is the only site which does not currently lay over potential future resources, making it the preferred candidate site. If further assessment in the future provides confirmation that Option 3 is not of interest from a potential resource perspective, it may be considered preferable to Option 8.

A geotechnical site investigation of the TSF footprint was undertaken, comprising the excavation and profiling of 40 test pits, from which representative samples of the soil horizons were retrieved for laboratory testing. The investigation, carried out during the 2020 DFS, did not include test pits near the critical section of the repositioned embankment wall and basin, and it is recommended that the site investigation be updated to include the entire footprint area in the early stages of the detail design phase. The typical soil profile comprises a 0.1 m to 1.0 m thick topsoil layer, below which a 1.0 m thick transported horizon occurs in the form of either loose and medium dense gravels in a clayey silt matrix or a stiff, shattered, clayey silt, which is often pin-holed and potentially collapsible. This is underlain by a 1.2 m thick layer of laterite nodules in a clayey silt matrix encountered in some of the test pits. Laterite was encountered in all the test pits at an average depth of 1.7 m below surface and in almost all cases extends to the bottom of the test pits. Refusal with the excavator occurred within this horizon in most of the test pits.

The TSF for Kobada comprises the following:

- An HDPE-lined, full containment valley Tailings Dam (TD);
- Associated TD infrastructure, which includes slurry distribution pipeline, catchment paddocks, toe drain system, underdrainage system, curtain drain system, blanket drain system, solution collection pipeline, collection sumps and manholes, seepage cut-off trench, storm water diversion trenches, emergency spillway, access roads and perimeter fence line; and
- A floating barge to decant supernatant tailings slurry water and storm water from the facility back to the plant.

The TSF shall be constructed in phases over the Life of Mine (LOM) as summarised in Table 1.10, with five downstream lifts following the construction of the initial starter embankment. The construction of Phase 1 has been split into Phase 1A in the first year of construction and Phase 1B in the second year of construction.

Table 1.10: Key Parameters Associated with the Kobada TSF

Tailings Dam Parameter	Phase 1A	Phase 1B	Phase 2	Phase 3	Phase 4	Phase 5
Total footprint area of the facility within the tailings surface and pond (ha)	160	200	260	320	350	370
Maximum design embankment wall elevation (mamsl)	388	390.5	396.5	401	403.5	406
Maximum design embankment wall height (m)	18	21	27	31	34	36
Outer side slope of embankment wall	1V:3H					
Inner side slope of embankment wall	1V:2H					
Embankment wall crest width (m)	8					

Tailings Dam Parameter	Phase 1A	Phase 1B	Phase 2	Phase 3	Phase 4	Phase 5
Embankment wall material	Waste material from mining operations (to be confirmed with geotechnical test work)					
Years of tailings deposition	1.25	1.25	3	3.2	3.2	3.2
Cumulative years of tailings deposition	1.25	2.5	5.5	8.7	11.9	15.1
Tonnes of dry tailings stored in TSF (Mt)	3.6	3.7	8.2	10.5	9.6	9.5
Cumulative tonnes of dry tailings stored in TSF (Mt)	3.6	7.3	15.5	26.0	35.6	45.2

The delineation of the zone of influence of the TSF was carried out based on a high-level volumetric dam breach assessment for the full release of tailings and compared with the method specified in SANS 10286, with the hazard classification in accordance with the CDA *Dam Safety Guidelines*. The classification was determined based on the agreed undertaking by AGG that the housing/private infrastructure within the zone of influence of the TSF shall be relocated. Based on the assessment criteria outlined in the CDA guidelines, the TSF is classified as a "Very High Hazard Facility".

Seepage and slope stability analyses were conducted on the TSF using material parameters from laboratory test work results conducted on the tailings and samples obtained from the geotechnical site investigation.

The waste material mined from the pit, which is deemed to have similar properties to the saprolite soil from the TSF basin, shall be used for the various TSF embankment phases. It is recommended that geotechnical laboratory test work be conducted on representative samples of the overburden/waste material from the mining operation to validate its suitability for the construction of the TSF embankment walls.

The slope stability factors meet, or are greater than, the prescribed values recommended by CDA under normal conditions, during construction, and seismic conditions, namely 1.5, 1.3 and 1.0, respectively. In the absence of a recommended factor of safety for particular upset conditions (non-operational drain, liner damage, etc), Epoch has assumed 1.3.

A TSF/mine-wide deterministic monthly water balance has been developed in Excel, based on the average normal year monthly, wettest year monthly and driest year monthly rainfall and evaporation figures to simulate the flow of water between the TSF and plant over the LOM. The outcomes of the balance indicate that:

- The mine wide water balance is positive, i.e., a surplus of water is generated within the mine wide water circuit, requiring discharge. Surplus water from the open-pit dewatering boreholes and open-pits themselves, due to storm water accumulation during the wet season, needs to be discharged into the downstream environment when not consumed by the process plant. This occurs intermittently from Year 1 to Year 3 and all year round from Year 3 to end of LOM.

- The process plant can accept 80% of the TSF slurry water volume sent to the TSF back, with the remaining 20% being sourced from clean water sources. As such, the TSF meet the 80% process plant water demand from May to October. For the remaining months additional water, above the 20% clean water sources, needs to be augmented to the TSF water from the open pits dewatering boreholes, etc.
- The TSF pool size reaches a maximum storage volume of 1,35 million m^3 in the month of August based on the wettest year rainfall water balance simulations.

The capital expenditure (CAPEX) associated with the TSF has been determined to a Class 2 Association for Advancement of Cost Engineering (AACE) accuracy of (+20%, -15%) based on quantities measured by Epoch and rates sourced from earthworks and liner tender enquiries undertaken & provided by SENET.

The estimated CAPEX has been determined for each phase of the development of the TSF, allowing for the costs to be allocated either to the initial CAPEX budget for the Project or the sustaining CAPEX/operating cost (OPEX) as deemed necessary.

The estimated costs associated with each phase of the TSF, which represent the initial and sustaining TSF CAPEX that occurs over the LOM, are given in Table 1.11. The Preliminary and General costs (P&Gs) are set to 26.7% of the measured works, as was done for the 2020 DFS. No contingencies have been allowed for as this is provided for in the overall project contingency.

Table 1.11: CAPEX for the Kobada Gold Mine TSF

Description	Amount (US$ Million)					
Schedule	**Phase 1A**	**Phase 1B**	**Phase 2**	**Phase 3**	**Phase 4**	**Phase 5**
Site Clearance	1.64	0.00	0.46	0.44	0.35	0.28
Earthworks and Excavations	4.81	1.36	4.62	6.94	4.94	2.71
Drainage	11.94	2.71	3.58	3.17	2.49	1.88
Concrete Structures	0.06	1.61	0.33	0.32	0.32	0.32
Pipe Work	1.59	0.00	2.08	1.41	0.09	0.00
Gabions	0.00	0.14	0.14	0.14	0.14	0.14
Catwalk	0.01	0.00	0.00	0.00	0.00	0.00
Warning Signage and Safety	0.30	0.00	0.00	0.00	0.00	0.00
Miscellaneous	0.12	0.15	0.37	0.52	0.65	0.72
Total Measured Works	20.47	5.98	11.58	12.95	8.98	6.06
P&G Costs	5.47	1.60	3.09	3.46	2.40	1.62
Total CAPEX per Phase	25.94	7.58	14.68	16.41	11.38	7.67
Total CAPEX of Final TSF						**83.65**

The OPEX associated with the TSF have been estimated at US$ 12.58 million over the LOM, based on typical full containment operational costs sourced from tailings dam management companies. The OPEX of US$ 0.74 to US$ 0.9 million per annum comprises:

- US$ 0.6 million per annum for operational management, comprising a team leader/manager, a supervisor, unskilled labour, and a spares workshop. This team shall be responsible for:
  - Day to day depositional management;
  - Maintaining the pool wall;
  - Maintenance and repairs to the slurry delivery pipeline and valves;
  - Monitoring and cleaning of the drains;
  - Seepage collection sump pump monitoring;
  - General maintenance (cleaning trenches); and
  - Monitoring various components (freeboard, drain flows, water returns, rainfall, tonnes deposited, etc.).
- US$ 0.08 to US$ 0.31 million per annum for pipeline and valve replacement costs and maintenance, depending on the pipeline length per phase of construction; and
- US$ 0.06 million per annum for quarterly inspections, monitoring and quarterly reports by the design engineer.

The total LOM cost associated with the TSF is estimated at a Present Value of US$114.66 million at a zero-discount rate. In addition to the costs described above, this includes

- Detailed design of the TSF and construction supervision during the construction of the various TSF phases (a total of US$3.93 million); and
- Rehabilitation and closure of the TSF and post-closure monitoring for a period of five years (a total of US$14.50 million) estimated to an accuracy of ± 35 %

The following recommendations are proposed for consideration and evaluation during the detailed design of the TSF:

- The geotechnical site investigation be completed to include test pits in the revised embankment wall area during the early stages of the detail design phase;
- Undertake a select number of geotechnical drillholes in the location of the TSF main embankment wall;
- Determine the geotechnical and geochemical nature, suitability, and selected sourcing of the open pit overburden material to be utilised in the construction of the TSF embankment walls;
- Undertake a more detailed flow slide assessment of the TSF to revalidate the zone of influence and path travelled by the pond water and liquefied tailings;
- Reconfirm/reassess the housing/private infrastructure to be relocated for the TSF zone of influence as part of the Resettlement Action Plan;
- Assess possible further optimisation of the TSF drainage system; and
- Appropriately size and position the TSF emergency spillway to pass the inflow design flow, considering flood routing through the dam.

1.9 HUMAN RESOURCE AND OPERATIONAL READINESS

1.9.1 Human Resources

The AGG human resource element is a very vital part of ensuring the operational success of the Kobada Gold Project. Significant consideration has been given in the recruitment strategy to ensure a seamless transition between commissioning and normal operation. AGG also recognises the necessity for the operation to employ a sustainable localisation plan within the Kobada mine area and nationally, and as such part of the policy is to recruit locally as far as practicable and implement a skills development plan for Mali nationals with focus on those local to the mine site.

In order to effectively manage the operations at Kobada, a labour schedule was drawn up to include labour for mining, process plant and administration duties, and to describe the labour complement that will be required for the Kobada Gold Project, inclusive of expatriates, West African national employees, and local Malian employees.

1.9.2 Overall Mine Management Structure

Figure 1.6 shows the overall management structure proposed for the Kobada Gold Project. The mine management will be structured in four main departments: the process plant, mining, finance and administration, and the health, safety, security, environmental and community (HSSEC) departments. All the respective departmental managers will report to a general manager, who will be responsible for the mine’s overall operation.

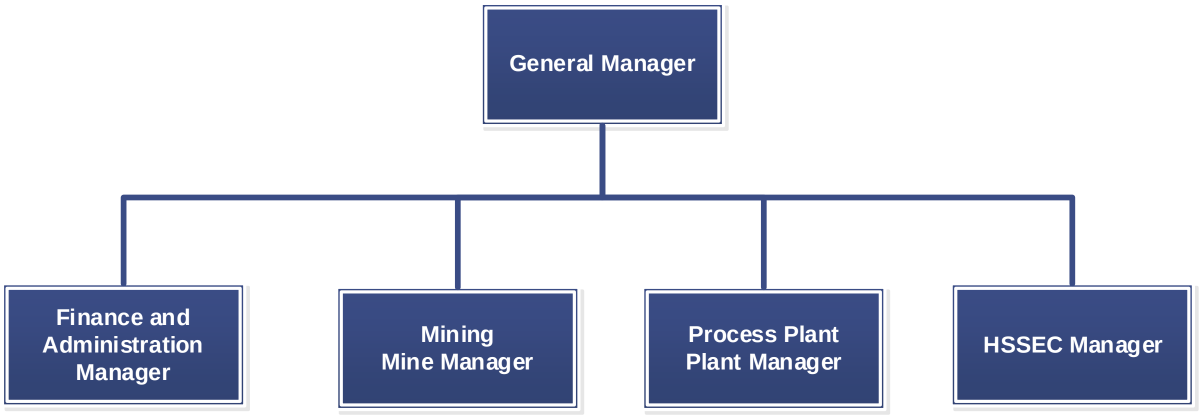


Figure 1.6: Overall Mine Management Structure

The process plant division will include a labour complement for the following:

- Maintenance and engineering
- Plant operations

The mining division will include the following departments:

- Mining operations
- Mining technical services

The finance and administration division will include the following departments:

- General management
- Human resources
- Finance
- General services
- Procurement and supply
- Legal, information technology (IT) and control

The HSSEC division will include the following departments:

- Health and Safety
- Security
- Environment
- Community Relations

Table 1.12 gives a summary of the total labour complement while Figure 1.7 shows the overall labour distribution as a percentage per division.

Table 1.12: Total Labour

Department	Number of Employees	% Distribution
Mining	39	20 %
Process Plant	101	50 %
Finance and Administration	49	24 %
HSSEC	12	6 %
Total	**201**	**100 %**

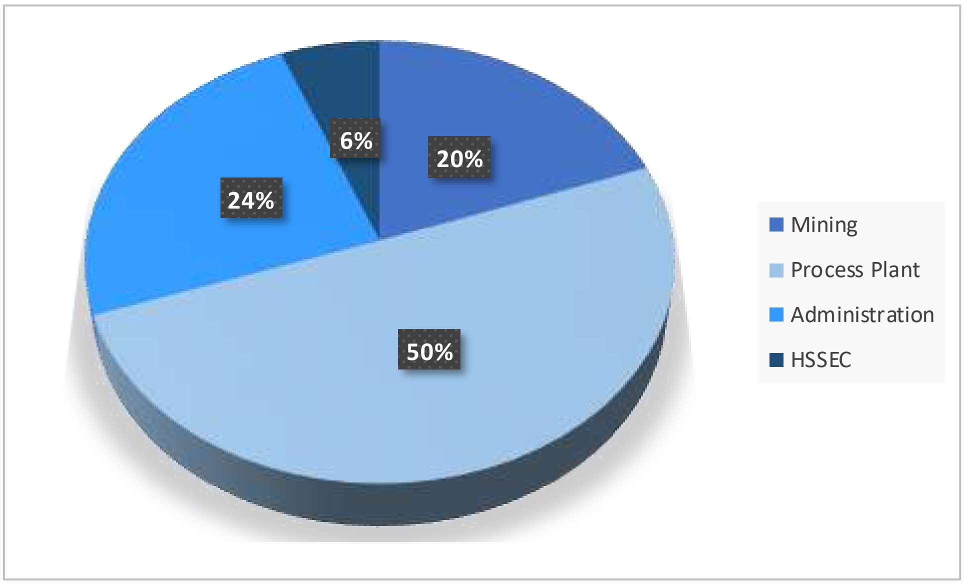


Figure 1.7: Total Labour Distribution

1.9.3 Operational Readiness

An Operational Readiness Plan (ORP) is recognised as a key factor in the successful delivery of capital projects. The ORP guides the organisation from the conceptual phase through design, construction, commissioning, ramp-up and steady-state operations by addressing all the aspects required to implement a successful operation. These include developing systems, programmes and materials required for start-up, maintenance systems and operational manuals. Training programmes linked to the personnel ramp-up, hands-on training, systems roll-out, commissioning planning and production ramp-up. The preparation of operational systems and procedures, such as safe work instructions to ensure that the plant can be operated in an efficient and safe manner, must be planned for.

The purpose of an implemented ORP is to mitigate risks during the transitional period from project to an operating asset. Risks are mitigated by early identification of operational alignment and asset management issues. The mitigation includes developing clear staffing dates to ensure sufficient time for operations, maintenance and support groups to be trained and ready to operate on functioning and implemented systems, processes and procedures.

The ORP spells out how the project will be phased into the operating business. It describes what will be done, how it will be done, in what sequence it will be done, and who will do it.

The OPR can be split into four main categories:

- Business Readiness
- Commissioning Readiness
- Maintenance Readiness
- Processing Readiness

1.10 PROJECT ON-SITE INFRASTRUCTURE

The Kobada Gold Project is a greenfield project, and as such minimal infrastructure has been established on the project site. The on-site infrastructure required will be related to the processing plant and the supporting facilities as follows:

- In-plant access roads
- Plant prefabricated buildings, i.e. plant office, change rooms, metallurgical laboratory, guardhouse, and weighbridge control room
- Plant steel and cladded structure buildings, i.e. process plant warehouse
- Plant reagents and consumables stores
- Process plant site-wide drainage system including pollution control dam
- Sewage reticulation and treatment
- Security buildings and perimeter security fencing
- Raw water dam and reticulation
- Process water dam and reticulation
- Potable water treatment and reticulation
- Communication systems and reticulation
- Power reticulation
- Localised process plant diesel storage and reticulation

1.11 PROJECT OFF-SITE INFRASTRUCTURE

The proposed project off-site infrastructure will support the mining and plant operations.

The main off-site infrastructure required for the development of the Project will be the following:

- Access roads and bridges, i.e. main access roads to the process plant and TSF, respectively, including nominal upgrades to main access routes
- Mining infrastructure and buildings, i.e. mining workshop, mining and process plant main administration building, and assay laboratory
- Camp and catering facilities
- Medical facilities, i.e. clinic and emergency room complete with ambulance facilities
- Power generation facility and distribution to all operating nodes, i.e. mining infrastructure and outer pit dewatering pumps, process plant, raw water dam pumping station and camp
- Fuel storage and dispensing systems, i.e. heavy fuel oil (HFO) and diesel storage for power generation and diesel fuel dispensing for the mining fleet
- Communication systems and reticulation
- Raw water supply and storage system, i.e. raw water abstraction from the Niger River including overland piping and raw water storage dam
- Sewage reticulation and treatment, i.e. for infrastructure buildings and camp

1.12 MINE CLOSURE AND SUSTAINABILITY

The Kobada Project is being undertaken with due consideration of biophysical, social and economic factors, as well as the relevant Malian legislative requirements, Equator Principles and International Finance Corporation (IFC) Performance Standards. The economic benefits of such a development are numerous; however, as in any mining project of this nature, there are also negative impacts, which will require planning, monitoring and mitigation during construction, operation, and decommissioning.

None of the negative impacts identified are considered fatal flaws, and high-significance impacts will become low to medium significance after the mitigation measures have been implemented.

An alternatives analysis was undertaken qualitatively, based on a comparison of the options against selected criteria, to determine the development footprint and impact.

A mitigation hierarchy has been applied throughout the Environmental and Social Impact Assessment (ESIA) and site selection process. The mitigation hierarchy is a systematic approach to mitigation planning and can be summarised into the following steps:

- Avoidance;
- Minimisation;
- Restoration; and
- Offsets.

AGG's objective for the rehabilitation and closure of the planned mine is to ensure that the site is left in a condition that is safe and stable, that long-term environmental impacts are minimised, and that any future liability to the community and future land use restrictions are minimised. The final post-closure land use will be determined in consultation with the Direction *Nationale de l'Assainissement et du Contrôle des Pollutions et des Nuisances* (DNACPN), stakeholders and local communities, and is likely to include the following:

- Areas for agriculture;
- Areas for livestock grazing;
- Wildlife habitat; and
- Water resources.

For reasons of health and safety, and in order to protect certain rehabilitation works from damage, portions of the Permit Area may be designated as exclusion zones. Natural soil covers and vegetation will, as far as possible, be re-established over these zones, but access by humans and/or livestock will be discouraged.

The conceptual closure plan developed for the Project has been based on the requirements of the Malian Mining Code (Law No. 2012-015 of 27 February 2012) and is in accordance with the international mining industry best practice.

1.13 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL IMPACT STUDIES

An ESIA was previously completed for the Kobada Project during 2015 which provided for the gravity treatment of ore to recover gold. This ESIA was approved, but the project was not developed. With the completion of the oxides DFS in 2020, which provided for a CIL plant and other key project changes, an updated ESIA was required to ensure compliance with the host country legislation as well as IFC PS and EP.

An ESIA process was undertaken between 2019 and 2021 with the project description based on the oxides DFS study completed in 2020. This assessment was based on the Carbon-In-Leach (CIL) process allowing for the processing of oxide ore and includes opencast pits, waste rock dumps, a TSF facility and supporting facilities and infrastructure. The oxides ESIA was submitted during August 2021 is currently with the *Direction Nationale de l'Assainissement et du Contrôle des Pollutions et des Nuisances* (DNACPN) awaiting approval. The approval is expected during Q4 2021.

The oxides DFS and associated ESIA provides for improved environmental and social management and mitigation measures, such as:

- Aligning the project with the requirements of the IFC PS and EP;
- Lining of the TSF and designing the facility according to industry best practice;
- The use of renewable energy;
- Installation of various pollution control measures/facilities; and
- Adjusting the layout taking into account the IFC PS 6 to minimise the impact on biodiversity.

An ESIA is currently under way for the 2021 sulphide DFS study with a view to aligning the oxides ESIA with the 2021 DFS study. The ESIA will be submitted to the DNACPN for approval once completed. The submission will be in the form of an addendum to the ESIA, which is a less onerous process.

The proposed mine is located 126 km southwest of Bamako, in the upper reaches of the Niger River catchment. The orebody is 9 km west of the Fié River and 7 km east of the Niger River.

The wind field in the Project Area is bi-modal with winds from the northeast during the dry season from November to May and winds from the southwest during the wet season from June to September.

The Concession Area has six watercourses and four of these are within the Project Footprint. Five of these flow eastwards to the Fié River, and one flows northwards to the Niger River. The proposed development is unlikely to impact measurably on the Fié or Niger Rivers, except for a possible new access road and bridge across the Fié River, east of the proposed mine.

Two aquifers were identified in the project area (upper saprolite material aquifer and underlying competent and fractured rock aquifer). The aquifers present in the area are classified as minor aquifers based on the expected relatively low yields, but they are of high importance to the surrounding communities and farmers as they are used for domestic, agricultural (stock watering) and artisanal mining purposes.

Groundwater samples were collected from 7 of the 23 hydrocensus points and submitted to an ISO/IEC 17025 accredited laboratory for chemical analysis in 2020. The water qualities were compared to the World Health Organisation (WHO) drinking water standards. In general, the groundwater quality is good, with only arsenic and nickel exceeding the WHO drinking water quality guideline at selected sampling locations.

Air quality sensitive receptors in the vicinity of the Kobada Project include the villages of Kobada to the west and Foroko to the north, as well as scattered residences and homesteads to the south and east of the operations. Current sources of fugitive dust and gaseous emissions include artisanal mining, vehicle activity on unpaved roads, subsistence farming, small-scale livestock farming, wildfires, and wind erosion from exposed areas. Household fuel burning and possibly refuse burning also constitute a significant local source of low-level emissions.

Noise receptors within a 3 km radius of the Kobada Project include the villages of Kobada to the west and Foroko to the north, as well as scattered residences and homesteads to the south and east of the operations.

Two threatened bird species, four threatened plant species, two near-threatened plant species and one data-deficient plant species were confirmed to occur. A mosaic of natural and modified habitat is present; the natural habitat is represented by seven floristically distinct vegetation associations while the modified habitat is represented by a degraded vegetation association (artisanal mines, villages, cultivated lands).

The ecological importance of aquatic ecosystems in the Project Area is Very Low, except for seasonal depressions, which have High importance because they support a moderate diversity of aquatic plant species and associated biota, including the barb *Enteromius pobeguini.* Aquatic ecosystems in the Project Area are classified as “Modified” in terms of the IFC Performance Standards. This classification is attributed to the elevated turbidity and sedimentation caused by artisanal mining that has resulted in a low diversity and low abundance of aquatic biota, including aquatic macroinvertebrates and fish.

The risk of waterborne diseases, such as cholera, malaria and bilharzia, within the Project Area is very high because of the abundance of artificial areas of inundation associated with artisanal mining. The period with the highest risk is likely to be towards the beginning of the dry season.

The Kobada Mining Permit affects four communes, namely Nouga, Kaniogo, Tagandougou and Séléfougou. These communes are located in the Kangaba Circle of the Koulikoro region as well as the Yanfolila Circle of the Sikasso region. There are eleven hamlets and sub-hamlets and six “mother” villages registered in the four communes, two circles and two regions, Koulikoro and Sikasso.

The main economic activities of the population include agriculture, livestock breeding, fishing, small-scale trade, and traditional gold mining.

Eleven cultural heritage sites have been identified in the villages of Banancoro, Foroko, Faraba Coungo and Séléfougou. In addition to these sites, cemeteries are also considered to be an integral part of the cultural heritage of the Project Area.

A summary of the predicted highly significant impacts prior to the implementation of mitigation measures is given in Table 1.13.

Table 1.13: Summary of Anticipated Significant Impacts

Aspect	Impact
Social	**Negative Impacts** • An influx of people seeking employment can be expected. This will place additional demand on services such as public safety, health care, water, sanitation, and housing • Minor, major and fatal injuries from potential mine health and safety incidents • Decommissioning and closure of the mine will have a negative impact on those previously employed by the mine and the families they support, and the businesses that provide services to the mine • Increase in malaria and other vector-borne diseases • Increase in HIV (Human Immunodeficiency Virus) infections and other sexually transmitted diseases • Physical displacement • Loss of artisanal mining sites
	Positive Impacts • The development will create local and regional employment opportunities in the construction and operational phases • Implementation of the community development plan • Procurement of local goods and services by the mine, employees and contractors will stimulate local business and create opportunities for entrepreneurship • The payment of royalties and taxes by mining companies to the government
Groundwater	Lowering and contamination of groundwater
Air Quality	• Elevated particulate matter (PM) PM_{10} and $PM_{2.5}$ concentrations over the LOM • Elevated dust fall levels over the LOM

Aspect	Impact
Terrestrial Ecology	• Loss of natural habitat of High or Moderate biodiversity value • Disturbance/loss of conservation-important plant and animal species • Loss of faunal habitat • Introduction/proliferation of alien invasive plant and animal species • Increased utilisation of plant and animal resources as a result of an influx of people into the Project Area
Surface Water	Contamination of surface water through hydrocarbon spills and metals of concern
Soils and Land Capability	Disturbance/loss of soil resources
Noise	Mining and construction of surface infrastructure will have a high impact on the Kobada community, and a 350 m buffer zone from the pit and associated infrastructure is considered adequate. For the purposes of this study a 500 m buffer zone was adopted.
Resettlement	• Partial resettlement of the Kobada community due to the placement of Project infrastructure and the implementation of an exclusion zone around the mine infrastructure. • Economic displacement and compensation of various farmers, landowners and caretakers in the area due to the establishment of mining infrastructure and associated activities on farms carrying subsistence and cash crops • Loss of agricultural resources
Geology and Sterilisation	• Sterilisation of gold and other resources • Landscape instability due to load placement of the waste rock dumps • Potential for pit wall failure
Traffic and Transport of Hazardous Materials	• Transport of fuel and reagents to the mine presents a risk of spill • Road dust from increased traffic, associated with the Project and general development in the area, presents a risk

It is anticipated that the negative environmental impacts will have an acceptable residual risk after mitigation measures were implemented.

1.14 CAPITAL COSTS

The capital cost (CAPEX) estimate includes engineering, procurement, construction, start-up and cold commissioning for the mining, process plant, TSF and infrastructure. Provision has also been made for the Owner’s costs.

The estimate is within the required accuracy level of +15 % −10 %. The estimate covers the direct field costs of executing the project; the indirect costs associated with the design, construction and commissioning of the new facilities; and the Owner’s support costs for items such as management teams, operational staff, environmental, permitting, insurance and utilities such as water supply, bulk power and construction power.

The total development/initial capital cost for the Kobada Gold Project was estimated to be **US$165 945 321,** which includes project execution, EPCM and contingency costs. These capital costs are summarised in Table 1.14.

Table 1.14: Initial/Development CAPEX Summary

Description	Capital Cost	Contingency	Total Capital Cost
	US$	US$	US$
Mining Preproduction & Establishment	27,094,882	352,000	27,446,882
Plant & Infrastructure	82,883,293	5,788,140	88,671,434
TSF Phase 1A	26,750,301	2,675,030	29,425,331
Pre-production Costs	14,198,934	1,419,893	15,618,827
Working Capital	4,348,043	434,804	4,782,847
Total Initial CAPEX	**155,275,453**	**10,669,868**	**165,945,321**

The total sustaining capital cost for the Kobada Gold Project was estimated to be **US$127,371,789,** which includes project execution, EPCM and contingency costs. These capital costs are summarised in Table 1.15.

Table 1.15: Sustaining CAPEX Summary

Description	Total Capital Cost
	US$
Mining	7,002,058
Process Plant	28,458,836
TSF Phases 1B to Phase 5	60,833,324
Mine Wide – Rehabilitation and Closure	25,435,654
Mine Wide – Post-Closure Costs	5,641,917
Total Sustaining Capital	**127,371,789**

1.15 OPERATING COSTS

The purpose of this operating cost (OPEX) estimate is to provide operating costs, and the associated general and administrative (G&A) costs, to an accuracy of +15 % −10 % that can be utilised for the economic analysis of the Kobada Gold Project.

The project's annual OPEX estimate for the LOM consists of the following:

- Mining operating costs estimated by DRA Mining
- Process plant and raw water supply operating costs estimated by SENET
- TSF operating costs estimated by Epoch
- Site G&A costs estimated by SENET
- Bullion transport, insurance and refining costs estimated by SENET

The overall LOM OPEX for the Kobada Gold Project is summarised in Table 1.16 with the cost distribution shown in Figure 1.8.

Table 1.16: Overall LOM Plant OPEX Summary

Description	LOM		
	US$/t processed	US$/oz	Distribution
Mining	11.25	421.24	47.81 %
Processing	8.55	320.39	36.36 %
G&A	2.23	83.34	9.46 %
Refining and Transport	0.10	3.70	0.42 %
Royalties	1.40	52.46	5.95 %
Total	**23.53**	**881.13**	100 %

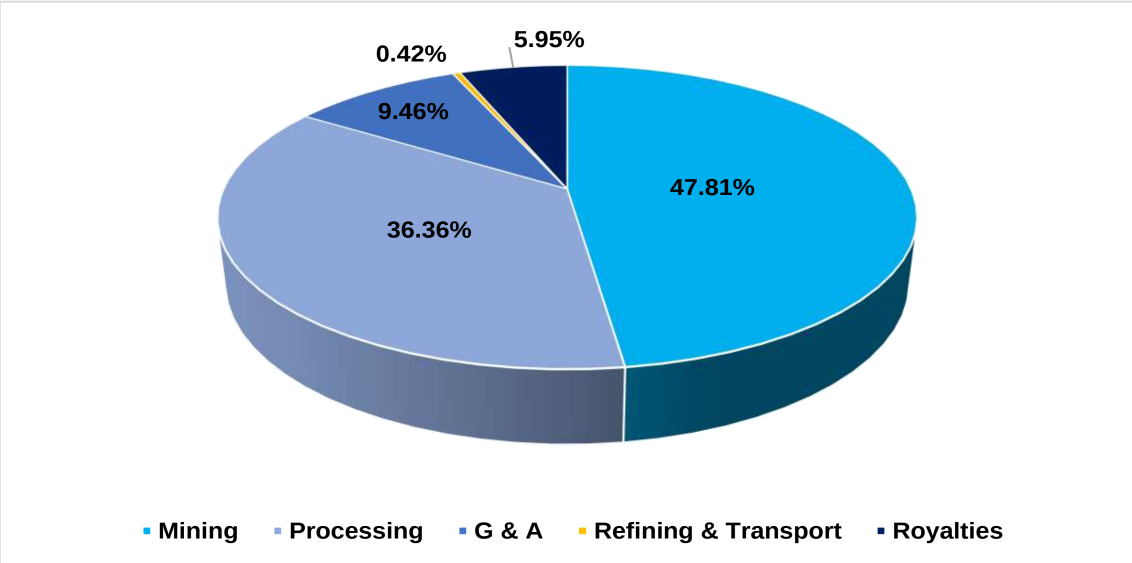


Figure 1.8: OPEX Distribution

The scheduled start of production is July 2023. The monthly plant operating cost for the first year of production ramp-up and the annual process plant operating costs for the LOM for the Project are summarised in Table 1.17 and Table 1.18, respectively.

Details of the operating costs are shown in the sections that follow.

Table 1.17: Year 1 (2023) Monthly OPEX Summary

Description	Unit	Year 1 Totals	Jul-23	Aug-23	Sep-23	Oct-23	Nov-23	Dec-23
			Month 1	Month 2	Month 3	Month 4	Month 5	Month 6
Mining	US$	23,386,714	4,162,592	3,767,348	3,799,819	3,819,034	3,841,266	3,996,655
Process plant, TSF and assay	US$	11,080,011	1,207,607	1,947,813	1,996,203	1,973,834	1,976,963	1,977,590
G&A	US$	3,313,495	552,249	552,249	552,249	552,249	552,249	552,249
Bullion transport, insurance and refining	US$	186,291	30,055	31,404	31,093	31,324	31,169	31,245
Government Royalties	US$	2,641,185	440,197	440,197	440,197	440,197	440,197	440,197
TOTAL	US$	38,846,906	6,392,702	6,739,011	6,379,364	6,376,442	6,401,647	6,557,739
Plant Throughput	Mt	1,377,348	125,871	245,969	253,820	250,190	250,698	250,800
Total Cost	**US$/t feed**	**28.20**	**50.79**	**27.40**	**25.13**	**25.49**	**25.54**	**26.15**

Table 1.18: LOM OPEX Summary

Description	Unit	LOM	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
			Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
Ore Processed																		
Throughput - Oxide (Saprolite)	t	28,290,947	1,041,199	2,704,187	2,842,214	2,783,174	2,659,879	2,742,496	2,338,542	2,054,526	515,617	3,885	1,868,248	2,873,879	2,727,521	1,135,275	304	0
Throughput - Oxide (Laterite)	t	1,624,004	336,145	292,341	145,900	31,360	58,988	27,647	1,482	2,405	101,029	0	32,780	109,602	188,316	280,777	307	14,923
Throughput - Transition	t	3,751,735	4	403	18,849	91,656	210,736	193,595	620,950	972,921	41,951	97,828	27,270	0	2,233	902,773	499,584	70,982
Throughput - Sulphide	t	11,362,145	0	0	0	0	0	0	0	0	2,337,024	2,957,126	1,028,781	14,032	14	712,062	2,540,700	1,772,407
Total Plant Throughput	t	45,028,831	1,377,348	2,996,931	3,006,964	2,906,190	2,929,603	2,963,738	2,960,975	3,029,853	2,995,620	3,058,838	2,957,078	2,997,514	2,918,084	3,030,887	3,040,896	1,858,313
Au Feed Grade																		
Au Grade - Saprolite	g/t	0.88	1.14	1.10	1.09	1.09	1.09	1.12	1.10	0.85	0.50	0.74	0.48	0.46	0.45	0.43	0.35	0.00
Au Grade - Laterite	g/t	0.85	1.29	1.14	1.12	0.99	0.95	1.08	0.74	0.66	0.46	0.00	0.48	0.46	0.45	0.45	0.48	0.45
Au Grade - Transition	g/t	0.87	,	0.45	0.89	1.26	1.17	1.16	1.20	1.07	1.12	1.16	1.04	0.56	0.55	0.47	0.44	0.41
Au Grade - Sulphide	g/t	0.84	,	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.33	1.04	0.97	0.80	0.80	0.49	0.46	0.44
Au Grade - Average	g/t	0.85	1.18	1.10	1.09	1.10	1.09	1.12	1.12	0.92	1.16	1.05	0.66	0.46	0.45	0.46	0.46	0.44
OPEX																		
Mining	US$	506,451,220	23,386,714	52,350,160	56,462,505	58,310,600	56,530,874	61,965,452	65,533,889	65,952,048	29,324,836	14,196,783	16,239,315	1,470,094	1,325,127	1,377,299	1,229,327	796,198
Process plant, TSF and assay	US$	385,195,669	11,080,011	23,048,746	23,119,153	22,776,405	22,976,062	23,178,485	23,028,120	23,616,250	30,360,154	32,605,509	26,189,933	23,397,730	22,888,467	26,127,596	31,459,855	19,343,193
G&A	US$	100,196,366	3,313,495	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	4,105,007
Bullion transport, insurance and refining	US$	4,448,421	186,289.66	379,085.73	394,816.47	366,378.39	366,938.42	381,066.50	382,158.12	319,007.78	391,594.59	360,207.10	217,852.70	156,626.43	148,640.30	154,427.15	153,633.78	89,697.81
Government Royalties	US$	63,068,991	2,641,184.56	5,374,616.00	5,597,643.86	5,194,453.47	5,202,393.43	5,402,699.06	5,418,175.91	4,522,840.60	5,551,964.61	5,106,957.86	3,088,680.27	2,220,624.11	2,107,398.09	2,189,443.05	2,178,194.85	1,271,720.98
TOTAL	**US$**	**1,059,360,667**	**40,607,695**	**87,779,599**	**92,201,108**	**93,274,827**	**91,703,257**	**97,554,692**	**100,989,333**	**101,037,137**	**72,255,540**	**58,896,447**	**52,362,772**	**33,872,065**	**33,096,623**	**36,475,755**	**41,648,001**	**25,605,817**
Plant Throughput	t	46,046,270	1,377,348	2,996,931	3,006,964	2,906,190	2,929,603	2,963,738	2,960,975	3,029,853	2,906,190	2,929,603	2,963,738	2,960,975	3,029,853	3,029,853	2,995,620	3,058,838
Total Cost	**US$/t feed**	**23.01**	**29.48**	**29.29**	**30.66**	**32.10**	**31.30**	**32.92**	**34.11**	**33.35**	**24.86**	**20.10**	**17.67**	**11.44**	**10.92**	**12.04**	**13.90**	**8.37**
Total Cost	**US$/oz**	**881**	**34**	**73**	**77**	**78**	**76**	**81**	**84**	**84**	**60**	**49**	**44**	**28**	**28**	**30**	**35**	**21**

1.16 MARKETING AND FINANCIAL ANALYSIS

The Kobada Gold Project financial analysis was prepared using the discounted cash flow model. In preparing this model, several assumptions and material factors were employed, as presented in Table 1.19.

Table 1.19: Financial Analysis Assumptions

Description	Unit	Assumption
Evaluation Start Date		01 January 2022
Revenue		
Gold Price	US$/oz	1,750
Refining Losses	%	0.08
Discount Rate	%	5.0
Fuel Prices		
Diesel Price	US$/L	0.797
HFO Price	US$/L	0.428
Fiscal		
Government Royalty	%	3
Tax Holiday	Years	3
Tax Rate (after tax holiday)	% of profits	30
Tax Rate (if there is a loss)	% of annual turnover	1
Dividend Tax	%	10
Depreciation	%	10 % over 10 years
Units of Measurement		
Kilograms to Ounces	kg/troy oz	32.1505
Diesel Specific Gravity (SG)	t/m^3	0.85
HFO SG	t/m^3	0.97
Other Charges		
Bullion Transport and Refining Costs	US$/oz	3.70
Exchange Rates	ZAR/US$	14.50
	£/US$	0.93
	A$/US$	1.57
	C$/US$	0.81
	CFA/€	655
	CFA/US$	561

The findings of the model are summarised in Table 1.20.

Table 1.20: Summary of Financial Findings

Description	Unit	Pre-Tax	After Tax
LOM Tonnage Ore Processed	kt	45,028	45,028
LOM Feed Grade Processed	g/t	0.87	0.87
Production Period	Years	15.6	15.6
LOM Gold Recovery	%	95.6	95.6
LOM Gold Production	koz	1,202	1,202
LOM Payable Gold After Refining Losses	koz	1,201.3	1,201.3
Gold Price	US$/oz	1,750	1,750
Revenue	US$ million	2,102	2,102
LOM Operating Costs	US$/oz	881	881
All in Sustaining Costs (AISC)	US$/oz	972	972
Net Present Value (NPV)	US$ million	506	355
Internal Rate of Return (IRR)	%	44.8	37.6
Discount Rate	%	5.0	5.0
Payback Period (from start of construction)	Years	3.91	3.91
Payback Period (from start of production)	Years	2.33	2.33
Project Net Cash	US$ million	773.1	549.9

The NPV sensitivity of the Project is shown in Figure 1.9, and Table 1.21 to Table 1.26 detail the NPV and IRR sensitivities of the Project to gold price, CAPEX, OPEX, recovery and feed grade.

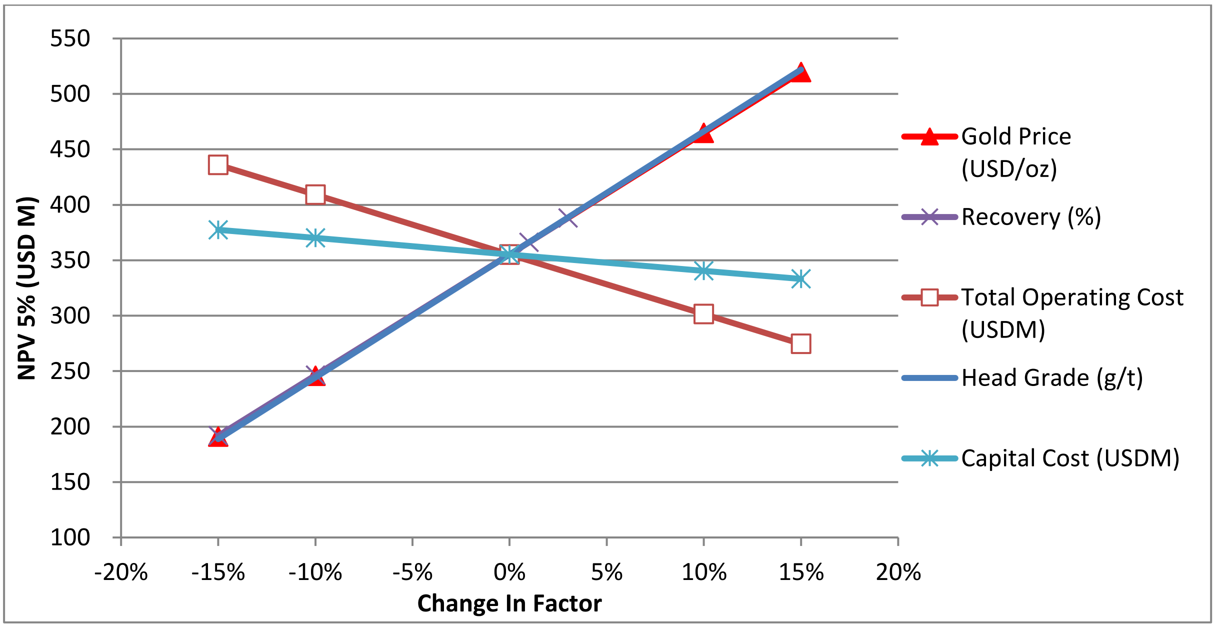


Figure 1.9: Sensitivity to NPV

Table 1.21: Key Project Metric Sensitivity to Gold Price

Description	Unit	Change in Gold Price (%)				
		−15	−10	0	+10	+15
		Average Gold Price (US$/oz)				
		1,488	1,575	1,750	1,925	2,013
NPV at 5 % – After Tax	US$ million	191	246	**355**	465	520
IRR	%	23.2	28.1	**37.6**	47.0	51.7
Cash Flow Payback Period	Years	3.87	3.22	**2.33**	1.76	1.55
Maximum Funding	US$ million	170.29	169.99	**169.37**	168.76	168.46
NOTE: The values in bold indicate the base case NPV.						

Table 1.22: Gold Price and Discount Rate Sensitivity Analysis

Gold Price	NPV		
US$/oz	0 % Discount Rate	5 % Discount Rate	10 % Discount Rate
1,488	324	191	109
1,575	399	246	150
1,750	550	**355**	234
1,925	700	465	317
2,013	776	520	359
NOTE: The value in bold indicates the base case NPV.			

Table 1.23: Gold Feed Grade and Gold Price Sensitivity

Head Grade	NPV at Varying Gold Prices (US$/oz)				
g/t	1,488	1,575	1,750	1,925	2,013
0.738	46	93	189	281	328
0.782	95	145	244	343	392
0.868	191	246	**355**	465	520
0.955	286	346	466	587	648
0.999	333	396	522	648	712
NOTE: The value in bold indicates the base case NPV.					

Table 1.24: OPEX and Gold Price Sensitivity

Change in OPEX	NPV at Varying Gold Prices (US$/oz)				
%	1,488	1,575	1,750	1,925	2,013
−15	272	327	436	546	601
−10	246	300	409	519	574
0	191	246	**355**	465	520
10	136	192	301	411	466
15	108	164	275	384	439
NOTE: The value in bold indicates the base case NPV.					

Table 1.25: CAPEX and Gold Price Sensitivity

Change in CAPEX	NPV at Varying Gold Prices (US$/oz)				
%	1,488	1,575	1,750	1,925	2,013
−15	214	268	377	487	542
−10	206	261	370	480	535
0	191	246	**355**	465	520
10	177	231	341	450	505
15	169	224	333	443	498
NOTE: The value in bold indicates the base case NPV.					

Table 1.26: Recovery and Gold Price Sensitivity

Recovery	NPV at Varying Gold Prices (US$/oz)				
%	1,488	1,575	1,750	1,925	2,013
80.6	49	96	192	285	332
85.6	96	147	246	345	394
95.6	191	246	**355**	465	520
96.6	201	256	366	477	532
98.6	210	265	377	489	545
NOTE: The value in bold indicates the base case NPV.					

1.17 IMPLEMENTATION

The project implementation schedule has been compiled to ensure that the engineering, procurement and construction management activities are aligned for successful project execution. AGG will appoint an EPCM Contractor to execute or manage and oversee the execution (by appointed EPCM/EPC/turnkey subcontractors, who will report to and be managed by the EPCM Contractor) of the following work packages:

- Mining operation
- Process plant
- TSF
- Mine infrastructure
- Plant infrastructure
- Off-site infrastructure and roads
- Raw water supply
- HFO and solar farm power plant
- HFO and diesel fuel depot

The project schedule assumes that there will be a seamless advancement between the various phases of the project evolution. It is also recognised that this is a moderately aggressive schedule and that it will require diligent progress monitoring and coordination of all the parties involved. The project milestones are given in Table 1.27.

Table 1.27: Project Execution Milestones

Project Milestone	Project Month
Commencement of detailed engineering	Month 1
Commencement of procurement and contracts administration	Month 3
Placement of orders for long-lead delivery items	Month 5
Mobilisation of earthworks and TSF construction contractor	Month 5
Mobilisation of Structural, Mechanical, Plate Work and Piping (SMPP) construction contractor	Month 11
Mill delivered to site	Month 14
ROM material ready to feed ROM bin	Month 17
TSF complete	Month 17
Solar plant available for start of production	Month 18
Construction complete	Month 17
Commissioning complete	Month 19
First gold from CIL circuit	Month 19
Ramp-up to 100 % of nameplate production	Month 21

Placing the purchase orders for the long-lead equipment is crucial not only to ensure that the equipment is on site in time to allow for a seamless construction sequence and a successful project execution but also to obtain the certified information from the supply vendors on their equipment to complete the detailed engineering phase of the Project.

Some of the long-lead equipment for the Project is as follows:

- Ball mill
- Primary crusher
- MV switchgear
- Solar farm thermal plant
- Apron feeder
- Gantry cranes
- Interstage screens
- Tower crane
- Pumps
- Sewage treatment plant

1.18 RISKS, OPPORTUNITIES, CONCLUSIONS AND RECOMMENDATIONS

1.18.1 Risks

The risks that are most likely to occur and would have a marked impact on the viability of the Project or operation and the processes or mechanisms in place to mitigate the potential impact of these risks are as follows:

- **Gold Price:** The Project has shown favourable economics. Sensitivity of gold price has been investigated in detail, and the Project has proven viable at a decreased gold price (−15 %).

- **Currency Fluctuations:** Specifically related to the strength of the euro and United States dollar, the currencies in which commodity prices are generally quoted. Risk of these currencies gaining or losing value, which will affect commodity prices, and Project CAPEX and OPEX. Possible purchase of forward cover and hedging could be applied to mitigate the impact on the Project.
- **Country/Political Risk:** This includes political unrest, economic policy changes, legislative and fiscal changes. AGG will ensure political insurance on all loans for the duration of the Project.
- **Logistics:** AGG has a project implementation plan that considers the potential logistics challenges such as the rainy season. A construction period of 19 months has been estimated and is considered adequate, with respect to logistics. AGG will appoint a reputable Malian transporter with experience in mine projects and operations, to ensure minimal logistics problems during construction and the operation of the Project. AGG will have a 1-month stockholding for all reagents and consumables, ensuring enough buffer to mitigate any logistics challenges.

1.18.2 Opportunities

An increase in resource can be realised in the following opportunities:

- **Mineral Resource:** With limited additional drilling there is an opportunity to convert further Inferred oxide Mineral Resource to an indicated Mineral Resource and increase the Mineral Reserve and LOM within the current pit limits.
- **Exploration Upside:** The current mineral resource is focused on a 4 km strike of the Main Shear Zone. A further 26 km of shear zones with similar structural geology have been identified on the concession through regional exploration techniques. Additional drilling within these identified shear zones may increase the size of the overall resource.
- **Construction Schedule Optimisation:** Ordering long-lead items as early as is practically possible, even prior to detailed engineering, will greatly improve the Project schedule.
- **Marginal Ore:** If it is decided to use a higher gold price for the COG calculation, there is a possibility of increasing the amount of marginal ore to be stockpiled and then processed in order to increase the LOM.
- **Underground Mine:** The Kobada deposit is currently interpreted open at depth and hence underground mining could be investigated for viability after the open pit reaches a certain depth.
- **Transition Ore:** A conservative assumption was made when considering the transition ore's characteristics and metallurgical performance. With further test work, there is a potential upside to optimise gold recovery and OPEX.

A further opportunity can be realised in the gold price, the Project's viability was evaluated at a gold price of US$1 750. The current gold price is > US$1 762 and increasing. The sensitivity shows a more than US$100 million increase in NPV based upon a gold price of US$1 925.

1.18.3 Conclusions and Recommendations

Since AGG has become involved in the Kobada Gold Project, considerable effort has been made and expenditure has been incurred to certify what is now believed to be a significant gold resource and reserve at Kobada. This Report attests to the extensive amount of exploration, tests and study work carried out on the Project. It is believed that the level of accuracy used herein is sufficient to consider this Report to be compliant with the NI 43-101 requirements with its demonstration of the technical feasibility to develop a gold mine at Kobada that will produce the current reserve of 1 202 276 oz over a 15.6-year LOM.

The Report has demonstrated that the Kobada ore deposits can be economically mined using the open-pit method and processed through conventional gravity/CIL technology at an annual rate of about 3 Mt/a.

The Report has been completed based on closely spaced drilling of only 4 km of the Main Shear Zone. Several other geologically similar shear zone structures have been identified on the concession, and these, with the exception of the Gosso target, are yet to be drilled.

2 INTRODUCTION

2.1 ISSUER RECEIVING THE REPORT

This updated NI 43-101 Technical Report was compiled by SENET for AGG with contributions from the QPs as set out in Table 2.1 to support AGG's press release dated 29 September 2021 and to summarise the results of the updated DFS of the Kobada Gold Project.

Minxcon (Pty) Ltd ("Minxcon") was commissioned by African Gold Group Inc. ("AGG" or "the Client") to update the Mineral Resource Technical Report on the Kobada Gold Project ("Kobada" or "Project") situated in Mali.

Table 2.1: Qualified Persons and their Contributions

Qualified Person	Company	Contribution
Uwe Engelmann	Minxcon Group (South Africa)	Geology and mineral resources
Ghislain Prévost	DRA Americas (Canada)	Mining, mineral reserves
Stephanus JP Coetzee	ABS Africa (South Africa)	Environmental and social assessment
Guy Wiid	Epoch Resources (South Africa)	Tailings facilities
Nicholas Dempers	SENET (South Africa)	Metallurgical test work interpretation Processing plant and project infrastructure Economic evaluation Coordination and compilation of Report

2.2 TERMS OF REFERENCE AND PURPOSE OF THE REPORT

SENET was commissioned by AGG to compile an updated NI 43-101 Technical Report on the Kobada Gold Project by coordinating the contributions from SENET's QP and the QPs from the other consultants. The Report is compiled in accordance with the National Instrument 43-101 and Form 43-101F1 (NI 43-101).

Minxcon was commissioned by AGG to update the Independent Technical Report (this "Report") on the Kobada Project. The Report is presented as a Mineral Resource Report and is compiled in accordance with the National Instrument 43-101 and Form 43-101 F1 ("NI 43-101").

The intention of this Report is to present the findings of a recent drilling campaign, as well as the results of a revised Mineral Resource estimation for the Project, and a updated DFS undertaken on the Kobada Gold Project.

The intention of this Report is to present the findings of the recent 2020 drilling campaign, as well as the results of the updated Mineral Resource estimation for the Project.

2.3 SOURCES OF INFORMATION AND DATA CONTAINED IN THE REPORT

The following sources of information were used to compile this Report:

- SENET (Pty) Ltd (2021). Kobada Gold Project Sulphides Update. Definitive Feasibility Study. September 2021.

- Minxcon (Pty) Ltd (2021). Updated NI 43-101 Technical Report on the Kobada Gold Project, Mali. Mineral Resource Report. Effective date 1 July 2021.
- Wolfe et al. (2016). Kobada Gold Project, Mali. Feasibility Study. Prepared for African Gold Group, Inc. Effective Date 3 February 2016.

For further details on references, please refer to Section 27.

This Report represents the independent opinions of the QPs based on the available source data, as supplied by AGG. The opinions are premised on historical data received from AGG as well as additional recent exploration drilling data. AGG has confirmed that, to the best of their knowledge, the information provided by them was true, accurate and complete, and not incorrect, misleading or irrelevant in any aspect. The QPs do not have any reason to believe that any material facts have been withheld. The historical data supplied by AGG was checked and verified to the extent possible.

2.4 QUALIFIED PERSONS' PERSONAL INSPECTION OF THE PROPERTY

A summary of the QP's qualifications and responsibilities is given below.

Mr Engelmann holds a BSc (Zoology & Botany) degree and BSc Honours (Geology) degree, is a registered Professional Natural Scientist with the South African Council for Natural Scientific Professionals (Pr.Sci.Nat. Reg. No. 400058/08) and is a member of the Geological Society of South Africa.

Mr Engelmann was involved in the planning and execution of the infill drilling campaign undertaken during 2018, 2019, 2020 and 2021. During this time, Mr Engelmann visited the project on numerous occasions, including site visits in June 2018, August 2019, and November 2019. During these site visits, Mr Engelmann reviewed and assisted in the implementation of the logging and sampling protocols, as well as the selection of the metallurgical samples for the feasibility study. In November 2019, Mr Uwe Engelmann and Dr Andreas Rompel (Vice President Exploration, AGG) visited the SGS Laboratory in Bamako to review their processes.

Mr Stephanus Coetzee, BSc (Hons) Environmental Management, Pr Sci.Nat (40044/04), is the Qualified Person responsible for the environmental and social assessment and compilation of the corresponding sections of this technical update report. By virtue of his education, membership of a recognised professional association, and relevant work experience, Mr Coetzee is an independent Qualified Person as defined by the NI 43-101 guidelines.

Dr George Papageorgiou, BSc Eng., MSc Eng. and PhD Eng., a Director and Senior Tailings Dam Engineer at Epoch Resources, is responsible for the design of the tailings storage facility.

Mr Guy John Wiid, BSc Eng., MSc Eng. and P. Eng., a Director and Senior Tailings Dam Engineer at Epoch Resources, is responsible for the review of the design of the tailings storage facility. Mr Wiid is a registered professional engineer with the Engineering Council of South Africa (#940269). By virtue of his education, membership of a recognised professional association, and relevant work experience, Mr Wiid is an independent Qualified Person as defined by the NI 43-101 guidelines.

Mr Nicholas Dempers, MSc Eng (Chem), BSc Eng (Chem), BCom (Man), Pr.Eng (RSA), Reg.No 20150196, FSAIMM (RSA), a Principal Process Engineer at SENET, is the Qualified Person for the mineral processing, metal recoveries and metallurgical testing sections and oversaw the compilation of the project infrastructure and the capital and operating costs as per the SENET quality management system. Mr Dempers did not visit the Project Area. By virtue of his education, as well as relevant work experience and membership of recognised professional associations, he is a Qualified Person as defined by the NI 43-101 guidelines.

Mr. Ghislain Prévost, M.Sc.A in Mining Engineering, OIQ membership No. 119054, a Principal Mining Engineer with DRA Americas Inc, is the qualified person for the mineral reserve and mining methods sections. By virtue of his education, membership of a recognised professional association, and relevant work experience, Mr Prevost is an independent Qualified Person as defined by the NI 43-101 guidelines.

3 RELIANCE ON OTHER EXPERTS

The information, conclusions, opinions, and estimates contained in this Report are based on the following:

- Information available at the time of preparation of this Report
- Assumptions, conditions, and qualifications as set forth in this Report
- Data, reports, and other information as supplied by AGG and other third-party sources

For this Report, the authors have relied on ownership information provided by AGG. In the consideration of all the legal aspects relating to the Kobada project, the authors have relied on AGG and assumed that the information relating to the legal aspects, and the status of surface and mineral rights, is accurate.

Property information in this Report has been sourced from previous reports supplied by AGG. The authors are not responsible for the accuracy of any property data, and do not make any claim or state any opinion as to the validity of the property disposition described herein.

For the preparation of this Report, the authors relied on maps, documents, and electronic files generated by the current and past exploration crews on behalf of AGG. To the extent possible under the mandate of an NI 43-101 compliant report, the data has been verified relating to the material facts.

Except for the purposes legislated under provincial securities laws, any use of this Report by any third party is at that party's sole risk.

According to AGG, there are no known litigations potentially affecting the Kobada Gold Project.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 AREA OF THE PROPERTY

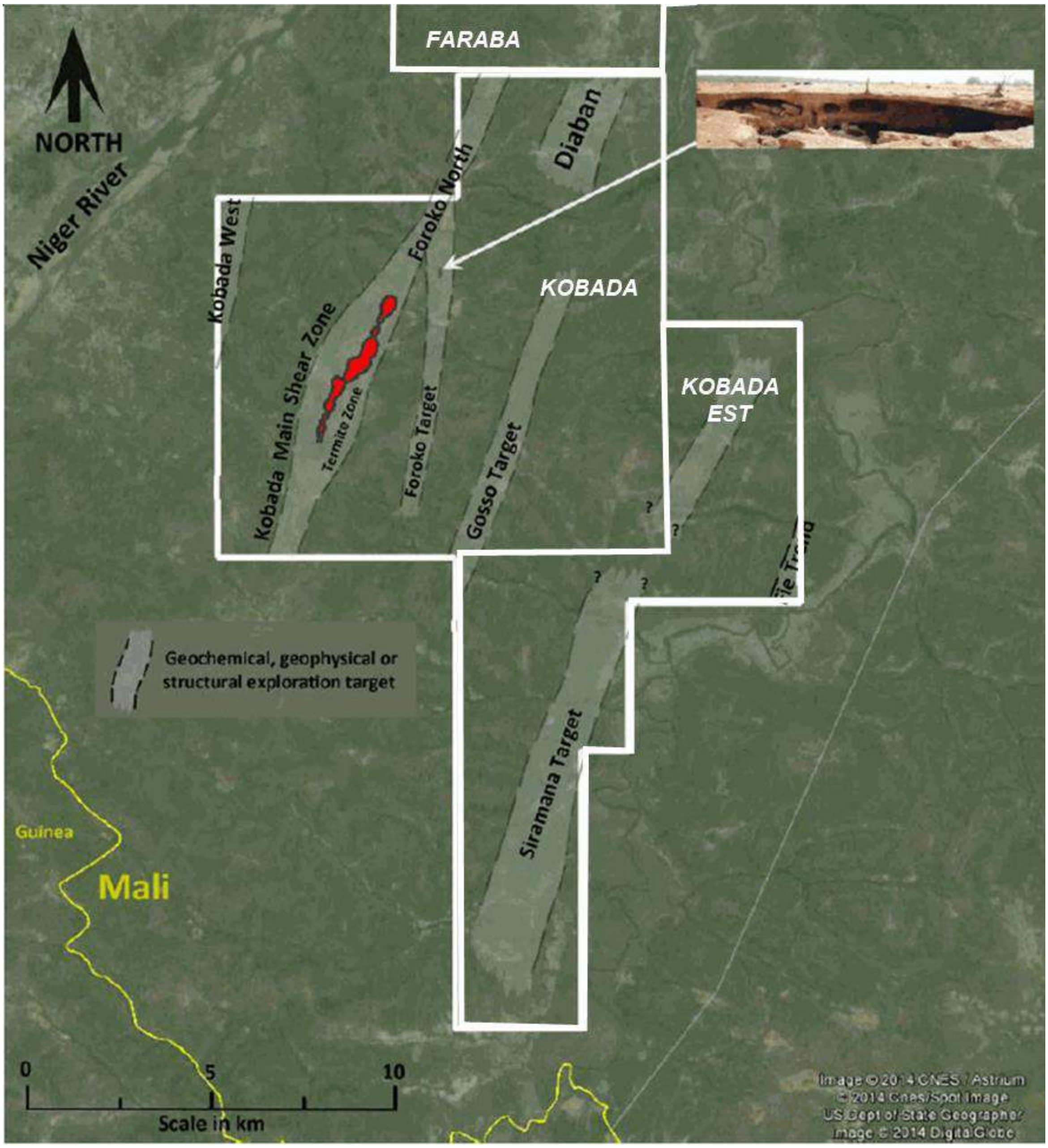
Kobada is an advanced exploration/development project in Mali, targeting primary lode gold mineralisation and secondary mineralisation in laterite. Secondary placer deposits also occur and have been locally exploited by artisanal miners.

The Project is at an advanced exploration stage, with a number of drilling campaigns having been completed over the years. A feasibility study was completed by Wolfe et al. in February 2016 that presented Mineral Resource and Mineral Reserve estimates in compliance with NI 43-101, mining methods, processing and recovery methods, required mining infrastructure, capital and operating cost estimates and an economic valuation. An additional infill drilling campaign was completed between September 2019 and February 2020 to confirm and upgrade the previous Mineral Resources. A revised feasibility study was completed by SENET for AGG in 2020. An updated DFS was completed in 2021 incorporating sulphides.

The Kobada Project comprises the following targets, as shown in Figure 4.1:

- Kobada Main Shear Zone
- Kobada West
- Foroko
- Foroko North
- Diaban
- Gosso
- Siramana

The geological model and Mineral Resources are currently limited to the Kobada Main Shear Zone, with limited drilling having been completed on the Gosso Shear Zone. Geophysical surveys and soil geochemical sampling programmes have been conducted on the remaining shear zones in varying degrees.



Source: AGG

Figure 4.1: Exploration Targets Identified Using Geochemical, Geophysical and Structural Interpretation

4.2 LOCATION OF THE PROPERTY

The Project is located in the Kangaba Cercle, Koulikoro Region of southern Mali, approximately 126 km due southwest of Bamako. The Niger River runs immediately north and northwest of the Project Area, with the Mali-Guinea international border occurring approximately 7 km to the west. The location of the Project within Mali is shown in Figure 13.2:

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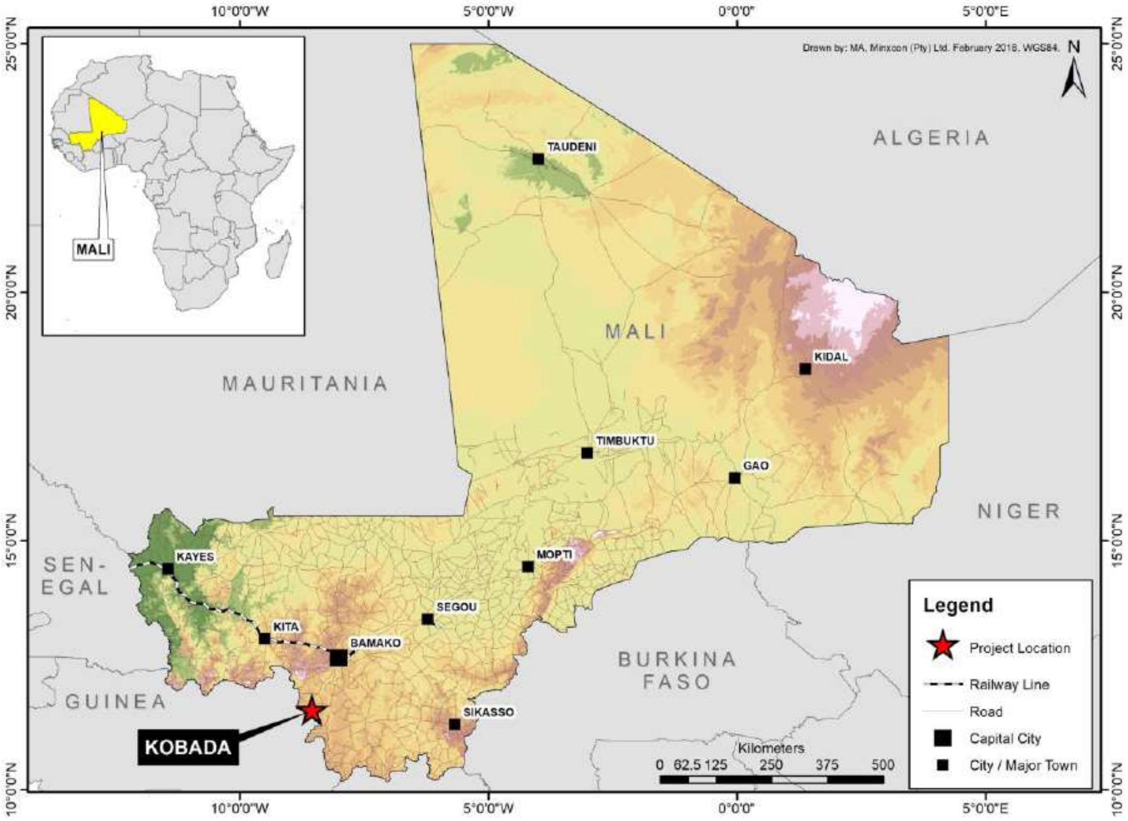


Figure 4.2: Location

4.3 MINERAL DEPOSIT TENURE

4.3.1 Country Mineral Legislation

The Republic of Mali is a member of the West African Economic and Monetary Union (WAEMU). The principal law regulating the mining sector is Ordonnance No.2019-022/P-RM dated 27 September 2019, which supersedes Law No. 2012-015 of 27 February 2012 (2012 Mining Code) as described below. The mineral titles relating to the Project are issued in terms of the 2012 Mining Code, thus the following regulations are relevant:

- 2012 Mining Code, which supersedes the 1999 Mining Code.
- Decree No. 2012-311/P-RM of 21 June 2012, implementing the 2012 Mining Code.
- Regulation No. 18/2003/CM/WAEMU of 22 December 2003, enacting the WAEMU Mining Code.

The industry is administered by the Ministry of Mines, Energy and Water, the Mines and Geology National Department, and the Mines Authority.

Mineral rights are awarded for reconnaissance, exploration, and mining, as described by Cuvex-Micholin (2019):

- **Prospecting Licence**: allows for reconnaissance or prospecting activities.
- **Exploration Permit**: granted for a duration of no more than three years, renewable twice for periods that may not exceed two years.

- Mining authorisation:
  - **Mining Permit**:
    - Granted for a period of thirty years, renewable for periods of ten years until available reserves within the permit are depleted.
    - May only be granted to the holder of an exploration permit or of a prospecting licence, may only cover the area included within the exploration permit or within the prospecting licence, and can only concern the minerals for which these titles were granted.
    - Must be held by a company incorporated under Malian law, with a free participation of the Malian State of 10 % free of any encumbrances. Moreover, the State reserves the right to exercise an option for additional participation of up to a further 10 % in the operating company.
    - Application for a mining permit shall be accompanied by an environmental impact study.
  - **Small-Scale Mining Permit**
  - Authorisation for **Artisanal Mining**

The 2012 Mining Code provides that indigenous persons or entities can acquire at least 5 % of the shares of a mining company under the same conditions as for other private shareholders. In addition, the State has a 10 % free participation in the mining company and may negotiate for itself additional participation in the capital of the mining company of up to 10% more (Cuvex-Micholin, 2019).

The new mining code sets out the following main changes, as described by Maiga and Schwartz (2019):

- Reduction of the cut-rate corporate tax period from 15 years to three years
- Removal of VAT (value-added tax) exemptions during mining production
- Introduction of a new windfall tax

The new mining code also aims to reduce the stability period, which exempts mining companies from any mining code changes that may take place after they have committed to investment in the country, to 10 years from the 30 years allowed under the previous code (McKay, 2019).

4.3.2 Kobada Project

The Kobada Project is held under a contiguous mining permit (Kobada) and two exploration permits (Kobada-Est and Faraba), issued in terms of the 2012 Mining Code:

- A mining permit (Permis d'Exploitation), No. PE 15/22 (Décret No. 2015-0528/PM-RM), for Group 2 minerals[1] was issued on 31 July 2015 to African Gold Group Mali

[1] Gold, silver, platinoids, copper, lead, molybdenum, zinc, titanium, vanadium, zirconium, niobium, tantalum, tungsten, rare earth elements, lithium, tin, cobalt, nickel.

SARL (AGG Mali SARL) and is valid for 30 years expiring on 30 July 2045. The permit must be renewed every ten years and is currently active. It encompasses an area of 135.7 km².

- The Kobada-Est exploration permit (Permis de Recherche), No. PR 18/957 (Arrêté No. 2018-3016/MMP-SG), for gold and Group 2 minerals, was issued on 16 August 2018 to AGG Mali SARL. The permit encompasses an area of 77 km² and was valid for three years, having expired on 15 August 2021. A renewal application was submitted to the Ministry of Mines, Energy and Water on 3 June 2021.
- The Faraba exploration permit, No. PR 17/921 (Arrêté No. 2018-0992/MMP-SG), for gold and Group 2 minerals, was issued on 6 April 2018 to AGG Mali SARL. The permit encompasses an area of 45 km² and was valid for three years, having expired on 5 April 2021. A renewal application was submitted after the expiry date (on 26 April 2021), triggering a late licence renewal application penalty, which has been settled.

The mineralised target areas under consideration are restricted to the boundaries of the mining permit.

The mineral rights are summarised in Table 4.1 and shown in Figure 4.3.

Table 4.1: Summary of Mining and Exploration Permits

Name	Permit Type	Number	Holder	Area km²	Validity	Minerals
Kobada	Mining	PE 15/22	AGG Mali SARL	135.7	31 July 2015 to 30 July 2045	Group 2 Minerals
Kobada-Est	Exploration	PR 18/957	AGG Mali SARL	77	16 August 2018 to 15 August 2021 *Renewal application submitted on 3 June 2021*	Gold and Group 2 Minerals
Faraba	Exploration	PR 17/921	AGG Mali SARL	45	6 April 2018 to 5 April 2021 *Renewal application submitted on 26 April 2021*	Gold and Group 2 Minerals
NOTE: Group 2 minerals are gold, silver, platinoids, copper, lead, molybdenum, zinc, titanium, vanadium, zirconium, niobium, tantalum, tungsten, rare earth elements, lithium, tin, cobalt, nickel.						

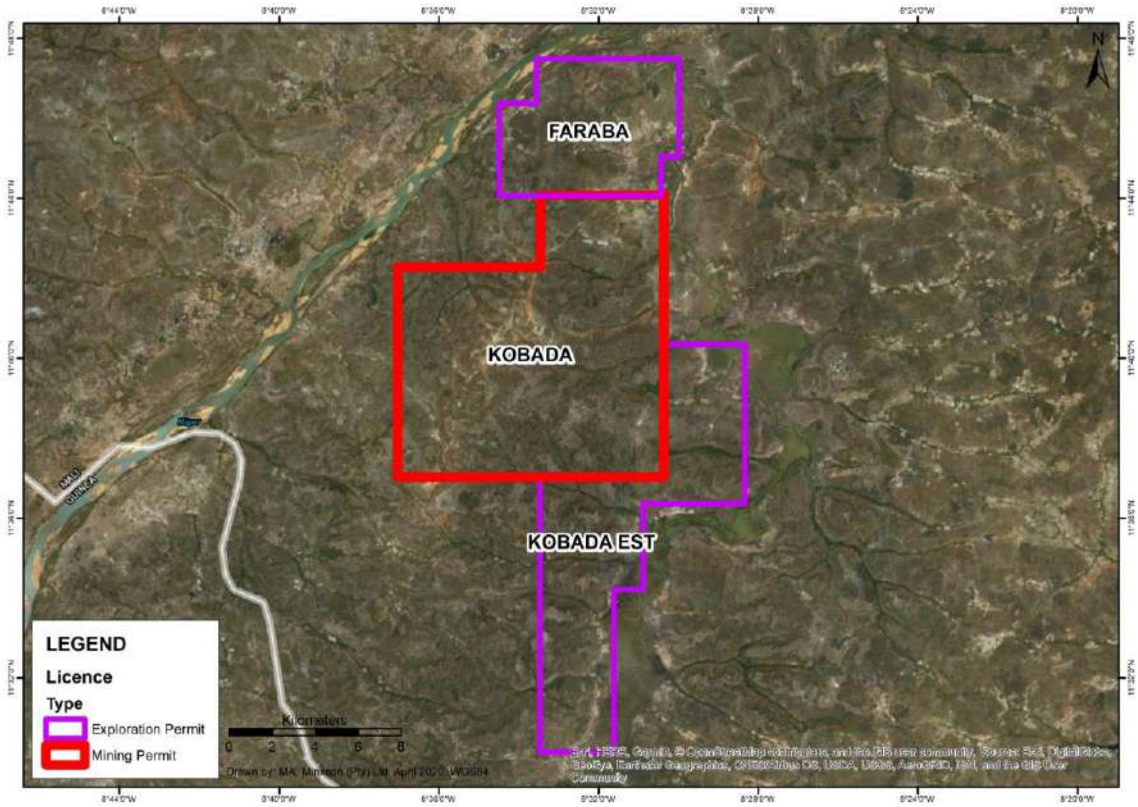


Figure 4.3: Mining and Exploration Permit Areas

4.4 ISSUER'S TITLE TO/INTEREST IN THE PROPERTY

The mineral rights that encompass the Project Area include a mining permit and two exploration permits, all issued to AGG Mali SARL. AGG Mali SARL is a wholly-owned subsidiary of AGG through AGG (Barbados) Limited.

The Government of Mali has the right to a 10 % free carried interest in the Malian Company that will operate the project (Kobada Operating Company SARL). The mining permit will ultimately be held in this subsidiary. The Government also has an option to purchase a further 10 % of the operation.

4.5 ROYALTIES AND PAYMENTS

Under the 2012 Mining Code, as per Cuvex-Micholin (2019), holders of an exploration permit are exempt from a number of taxes, including VAT, contribution or any direct or indirect taxes. However, they are subject, inter alia, to the following taxes:

- Registration fees
- Tax on salaries and emoluments due by the employees
- Annual surface royalty
- Charges and social contributions due for the employees

The holders of a mining permit have to pay, inter alia:

- Annual surface royalty
- Flat-rate contribution
- Charges and social contributions due for the employees
- Capital yields taxes
- Statistical royalty

The holder of a mining permit is exempted from VAT for a period that finishes at the end of the third year following the date of commencement of production.

4.6 ENVIRONMENTAL LIABILITIES

As indicated by AGG, no environmental liabilities are currently associated with the Project.

4.7 PERMITS TO CONDUCT WORK

An environmental permit (No. 2015-0032 MEADD-SG) relating to the oxides project only was held for the mining permit area and expired in 2018. An application for a new environmental permit, including an updated environmental and social impact assessment (ESIA) and community development plan, was submitted on 24 June 2021 to the National Directorate of Sanitation and Pollution and Nuisance Control. The application is currently under review.

An ESIA amendment may be required at a later stage for the development and mining of the sulphides portion of the Project.

Environmental permits are not required for exploration permits.

Prior authorisation must be obtained before commencing mining operations that would impact the flow and quality of water.

In Mali, mining title holders are obliged to provide compensation to private surface title owners of the affected land portions. However, AGG has indicated that the surface rights of the land on which the Project occurs are held by the Government of Mali, thus AGG has unencumbered rights of access.

Minxcon is not aware of further permits that are currently required for the Project.

4.8 OTHER SIGNIFICANT FACTORS AND RISKS

Artisanal mining is a common occurrence across Mali and most of West Africa. In the past, there has been significant artisanal mining activity across the concessions, targeting the nuggety gold in the upper lateritic layer. The numbers of artisanal miners fluctuate depending upon the availability of other higher-grade concessions in the region and the availability of the nuggety gold in the laterite layer. Minimal disruption occurs to the predominantly gold-bearing saprolite due to the soft and friable nature of the material, and the fineness of the entrained gold.

There have been no incidents of disruption by the artisanal miners throughout the past two years and through all of the drilling campaigns. AGG has adopted a strategy to keep lines of communication open with the artisanal miners, and this has prevented any conflict to date, with a large number of miners moving off once drilling started. AGG believes that as the mine moves closer to construction, the frequency of artisanal miner intrusions will subside as they will move on to other unoccupied tenements.

AGG believes that the risk pertaining to artisanal miner interruptions is therefore minimal.

Minxcon is not aware of any factors or risks that may impact the Project. Renewal applications have been submitted for Kobada-Est and Faraba, and the QP does not foresee these not being granted and is satisfied with the security of tenure.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 TOPOGRAPHY, ELEVATION AND VEGETATION

The topography of the Project Area is a generally flat erosional plain with a few low hills and lateritic plateaus rising steeply 50 m above the surrounding plain. The average altitude is 350 m above sea level (Wolfe et al., 2016).

Drainage is predominantly to the east towards the sinuous north-flowing Fié River, a tributary of the Niger River, which occurs approximately 4 km west of the Project boundary and 6 km west of the current exploration camp. Intense and fast-receding flooding may occur in the area. During the dry season, smaller streams may cease to flow (ESDCO, 2014).

ESDCO (2014) identified five main types of vegetation at Kobada:

1. Gallery Forests: 30 m to 70 m strips of richly diverse vegetation, located along permanent or temporary waterways and low-lying areas. Trees are typically > 13 m high and are mostly found in the southwest of the Project Area.
2. Wooded Savannah: fairly diverse 8 m to 13 m high trees, similar to fringe riparian species. These occur in the southwest of the Project Area.
3. Arboreous Savannah: less diverse 6 m to 8 m high trees and shrubs, scattered among 1 m high grass cover. They are found towards the southern portion of the Project Area.
4. Shrubby Savannah: diverse < 7 m high woody shrubs and bushes, scattered among tall grass cover.
5. Bowes: bare land with continuous or discontinuous grass cover and with some timber. The Bowes generally have a silty clay surface and/or laterite cover.

During the wet season, the countryside is lush with tall grass. During the dry season, villagers burn the grass (Wolfe et al., 2016).

Wildlife in the region is limited and generally occurs as small mammals, antelope, birds, and a few snakes (Wolfe et al., 2016; ESDCO, 2014).

5.2 ACCESS TO THE PROPERTY

The main access to the site is approximately a 2.5 h journey via a network of relatively well-maintained sealed roads from Bamako to Banankoro and from there via a short gravel road to the Niger River. Crossing is via a short ferry trip to a small village approximately 10 km west of the Project camp. AGG has negotiated a deal with the local community for the use of a 40 t capacity barge to transfer goods, equipment and personnel across the river. For larger equipment, the military make available a 100 t pontoon should it be required.

The ferry crossing (Banankoro) in relation to the AGG exploration camps is shown in Figure 5.1.

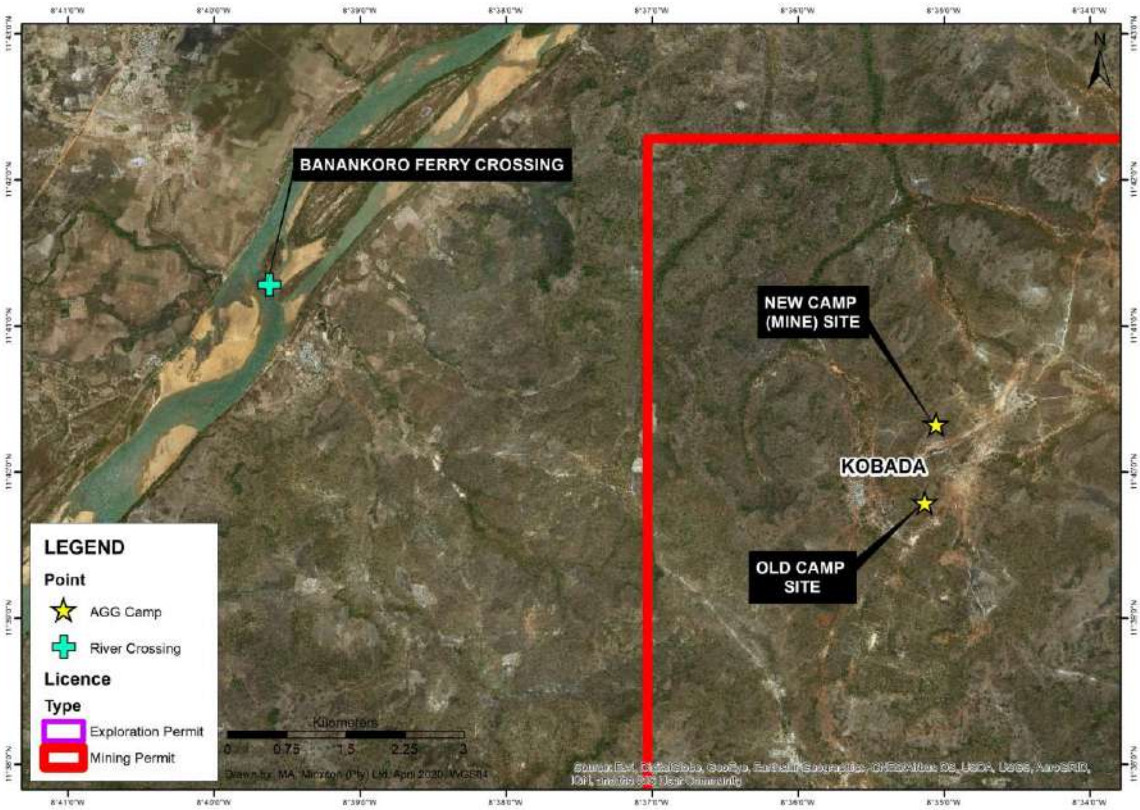


Figure 5.1: Location of Ferry Crossing in Relation to AGG Camp

Photographs of the military and commercial ferries utilised are provided in Figure 5.2 and Figure 5.3, respectively.

Figure 5.2: Commercial Ferry

Figure 5.3: Military Ferry

Alternatively, an approximately 6 h route from Bamako via car provides access to the Project Area from the eastern side. The journey is approximately 86 km on the RN7, and then 53 km from RN7 to the Sélingué Dam, and finally 48 km to the Project Area (Wolfe et al., 2016).

A network of gravel roads crosses the property.

5.3 PROXIMITY TO POPULATION CENTRES AND NATURE OF TRANSPORT

The Project Area is located in a rural and undeveloped region.

A number of villages occur in the area, with populations dedicated to farming, fishing, trading or artisanal mining. Kobada lies approximately 0.7 km to the west of the Mineral Resource area and Foroko 1.5 km away. The nearest sizeable population centre is the town of Sélingué, which lies 48 km east of Kobada (Wolfe et al., 2016).

Transport to the site is via a road network. A bridge constructed from sea containers provides access across the Fié River and is used during the dry season when the Niger River is too low for ferry boats. During the wet season, access is restricted to commercial ferry boats across the Niger River.

Whereas the Modibo Keita International Airport is located in Bamako, no aerodromes or railway lines are evident in the local area.

5.4 CLIMATE AND LENGTH OF OPERATING SEASON

According to ESDCO (2014), the Project Area lies within the pre-Guinean, or Sudan–Guinea, bioclimatic zone. This zone is characterised by average rainfall greater than 1,150 mm per year.

Typically, two seasons are distinguishable: a rainy season from May to October, and a dry season that is subdivided into a cold season from November to February and a hot season from March to April.

Over 65 % of the rainfall occurs over the period from July to September, with August hosting heavy rain and being the wettest month with an average rainfall of 265 mm.

The average annual temperature in the region is 29 °C. Peak midday temperatures in April and May often exceed 40 °C. Temperature differences between day and night are typically around 14 °C. Rivers and waterways create local, cooler microclimates.

The climate is unlikely to adversely affect operations for an extended period of time. Low rainfall, however, influences the water levels of the Niger River, which may hamper ferry crossing.

5.5 INFRASTRUCTURE

Infrastructure at the site is currently limited, but cell phone reception is good. Informal artisanal mining camps also occur in the area (see Figure 5.4).

Figure 5.4: Artisanal Mining Camp and Remnants in the Project Area

There are no power lines to the site, and currently power is supplied via generators. An old exploration camp includes basic buildings, which are currently being upgraded. Concurrently, a new exploration camp has been established approximately 1 km to the north-northeast with the intention of serving as the main camp, temporary office and other required buildings for the future proposed mine site. The sites relative to one another are shown in Figure 5.1. Figure 5.5 shows some of the accommodation at the new camp.

Roads that traverse the Project Area are gravel and not in all-weather condition.

Figure 5.5: Accommodation at the New Camp

6 HISTORY

6.1 PRIOR OWNERSHIP AND OWNERSHIP CHANGES

The Geological Survey of France, the Bureau de Recherches Géologiques et Minières (BRGM), conducted work in the Project Area in the 1980s, but it is unclear if the Project was owned by them. Additional work was completed in the 1990s by La Source (a joint venture between Normandy Mining NL and BRGM).

The property was acquired in 2000 by Compagnie Minière Or (Cominor), then a subsidiary of BRGM.

The Project Area was later sold to AGG in 2005.

6.2 HISTORICAL EXPLORATION AND DEVELOPMENT

Historically, the area has been worked by artisanal miners, with the first reported work occurring between 1935 and 1937 by one E. Julien (Wolfe et al., 2016). The first recorded work at Kobada occurred in the 1980s, when BRGM identified the Kobada Shear Zone by using geochemistry surveys in 1982. BRGM drilled 7 diamond drillholes over 913 m in total depth in 1988. In 1996, La Source completed a reverse circulation (RC) drilling programme that comprised 50 drillholes over 4,825 m. In 2002 and 2004, Cominor drilled a total of 132 drillholes, including 8,377 m of RC and 1,736 m of air core drilling. In 2009, IAMGold drilled two diamond (DD) drillholes over 200 m and 10 RC drillholes over 1,136 m.

Between 2005 and 2012, AGG completed 904 drillholes, including 26,901 m DD and 81,985 m RC. In a 2015 drilling campaign, a further 13 DD drillholes were completed over 1,398 m.

In 2019 and 2020, AGG commenced with a drilling programme to test and upgrade the Kobada orebody and Mineral Resource. The drilling programme consisted of a number of phases of drilling from the south to the north. The Gosso Target was also drilled to test the orebody between the historical drilling locations.

A feasibility study investigating proposed open-pit mining at Kobada was completed in February 2016 by International Resource Solutions (Pty) Ltd, Obsidian Geological Limited, Gekko Systems (Pty) Ltd and John Dunlop and Associates (Pty) Ltd, i.e. Wolfe et al. (2016). The report sets out the Mineral Resources, Mineral Reserves, and production schedules, and details the capital and operating expenditures over an eight-year life of project. The report culminates in a full economic analysis of the Project's value, with a Net Present Value (NPV) of US$86 million (5 % discount rate, US$1,200/oz gold price) and an Internal Rate of Return (IRR) of 43 %.

SENET completed a new feasibility study in 2020 which focused on the laterites, oxides and transitional zones but excluded the sulphides. An update of this feasibility study is currently in progress and is expected to be completed in August 2021 and focuses on the inclusion of the sulphides.

6.3 HISTORICAL MINERAL RESOURCE ESTIMATES

No Mineral Resources were stated by the previous owners of the Project.

The Mineral Resources were estimated in 2020 by Minxcon and reported in a compliant NI 43-101 Mineral Resource Report. The Mineral Resources were estimated utilising Ordinary Kriging, while the geological model was determined using Leapfrog Geo. These historical Mineral Resources are provided in Table 6.1. The applied lower cut-off grade was 0.35 g/t Au.

Table 6.1: Kobada Project Mineral Resources as at 1 June 2020

Mineral Resource Classification	Tonnes	Tonnes Less Geological Loss	Au Grade	Au Content	
	Mt	Mt	g/t	kg	koz
Measured	24.63	23.25	0.79	18,379	591
Indicated	22.02	19.70	0.95	18,673	600
Measured and Indicated Total	**46.66**	**42.95**	**0.86**	**37,053**	**1,191**
Inferred Total	**31.54**	**26.71**	**1.33**	**35,421**	**1,139**
Source: Minxcon					

6.4 HISTORICAL MINERAL RESERVE ESTIMATES

No Mineral Reserves were stated by the previous owners of the Project. The open-pit Proven and Probable Mineral Reserves were reported as part of the 2020 feasibility study, and updated as part of the 2021 SENET feasibility study (see Table 6.2).

Table 6.2: Kobada Project Mineral Reserves as at 24 September 2021

Category	Material	Ore (kt)	Grade (g/t)	In Situ Ounces (koz)	Recovered Ounces (koz)
Proven	Laterite	336	0.87	8.3	8.0
	Saprolite	12,276	0.90	331.8	320.1
	Transition	1,776	0.90	47.4	42.9
	Sulphide	5,870	0.94	140.4	133.9
Subtotal		**20,259**	**0.91**	**527.8**	**504.9**
Probable	Laterite	1,288	0.85	36.1	34.8
	Saprolite	16,015	0.88	465.6	449.3
	Transition	1,975	0.87	57.2	51.8
	Sulphide	5,492	0.84	165.9	158.2
Subtotal		**24,770**	**0.87**	**724.7**	**694.1**
Total Proven and Probable	Laterite	1,624	0.85	44.4	42.8
	Saprolite	28,291	0.88	797.3	769.4
	Transition	3,752	0.87	104.6	94.6
	Sulphide	11,362	0.84	306.3	292.2
Total		**45,029**	**0.87**	**1,252.5**	**1,199.0**

Notes:
1. The effective date of the Mineral Reserve Estimate is 24 September 2021.
2. Mineral Reserves are reported in accordance with the CIM guidelines.
3. A marginal COG of 0.35 g/t Au for all material is applied.
4. Mineral Reserves were estimated at a gold price of US$1,610/oz and include modifying factors related to mining costs, dilution and recovery, process recoveries and costs, G&A costs, and royalties.
5. Figures have been rounded to an appropriate level of precision for the reporting of Mineral Reserves.
6. Due to rounding, some columns or rows might not add up exactly as shown.
7. The Mineral Reserves are stated as dry tonnes processed at the crusher. All figures are in metric tonnes.
8. The in situ and recovered ounces are in troy ounces.

6.5 HISTORICAL PRODUCTION

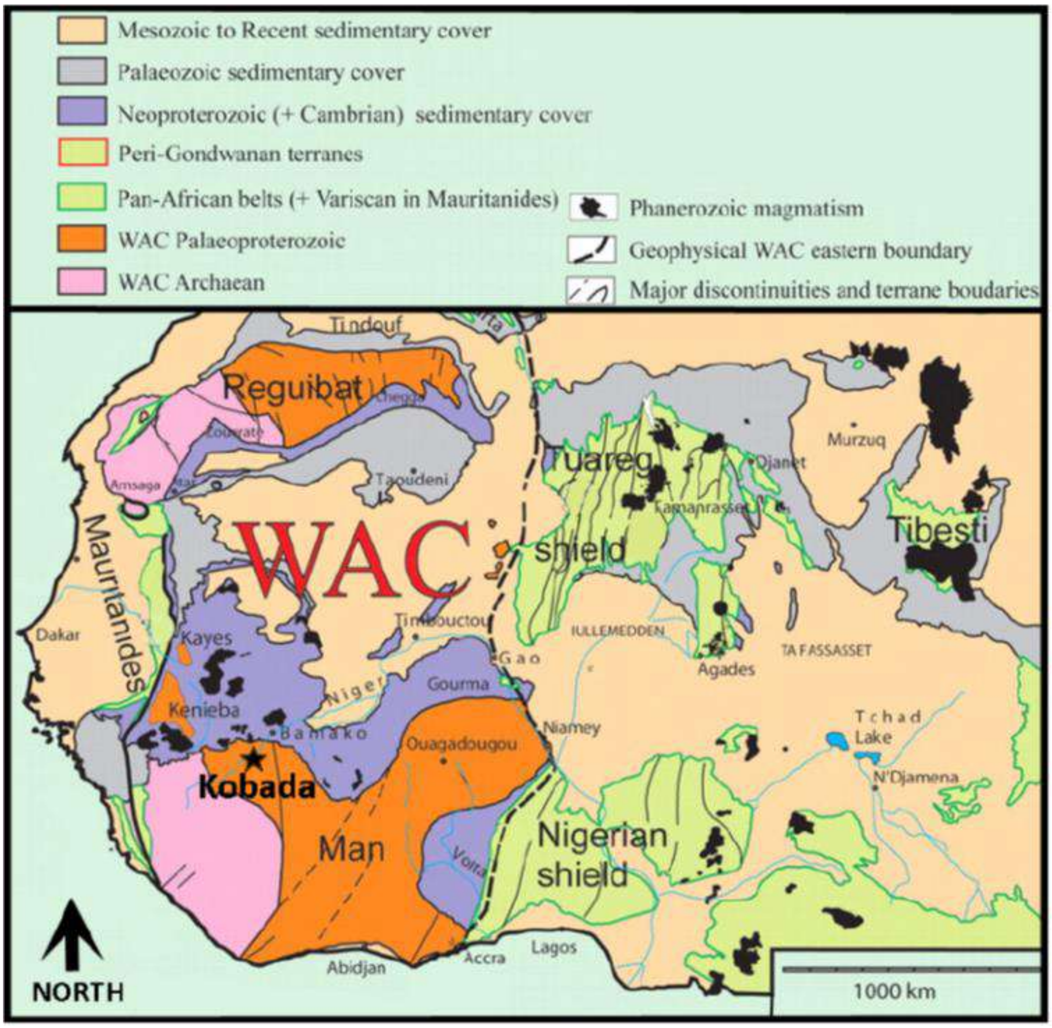
Artisanal mining (*orpaillage*) occurs in the area, but no formal production from the Kobada properties has historically been recorded.

7 GEOLOGICAL SETTING AND MINERALISATION

The following sections have largely been summarised from Wolfe et al. (2016).

7.1 REGIONAL GEOLOGY

The Project is located in the Bagoe Formation in the north-central border of the Birimian (2.2 Ga to 1.8 Ga) rock units that form part of the Leo, or Man, Rise in the southern part of the West African Craton (see Figure 7.1).



Source: Wolfe et al., 2016

Figure 7.1: West African Craton (WAC)

Northwest African geology is characterised by the Precambrian West African craton (WAC) that stabilised about 1,800 Ma. It comprises the south Leo (Man) and the north Reguibat Shields (Rises) and the significantly smaller geological windows of Kéniéba and Kayes. These basement exposures are bounded and separated by northerly trending Pan-African fold belts to the west and by an extensive cover of Late Proterozoic to Phanerozoic sedimentary basins, namely the central Taoudeni Basin.

The Leo Shield is composed of the Keneman-Man Domain in the southwest, and the Proterozoic Baoule-Moussi Domain. The Baoule-Moussi contains relics of Archaean rocks and the Paleoproterozoic Birimian formations. The Birimian consists of narrow elongated belts of mainly epi-metamorphosed volcano-sedimentary formations deformed by a number of regional events. These belts host the majority of the gold deposits of West Africa. Regional metamorphism reaches greenstone facies with amphibolite facies restricted to the intrusive granitoids contact.

Two volcano-sedimentary series occur in the Massigui Yanfolila region, separated by a north-orientated granitic intrusive unit (Massigui Batholith) and the NNE trending Banifing Shear system (Sassandra). To the west, the Bougouni-Kekoro Formation is a volcano-sedimentary unit composed mainly of orthoquartzite. To the east, the northern portion of the Bagoe Formation is composed of intermediate felsic volcanics with a few rare interlayers of basalt and metasediments.

The Bagoe Formation is divided into three distinct lithological units with gradual transition between these members:

- The east member, a flyschoide unit composed of sandstone to argillite with graphitic and conglomeratic bands
- The centre member, quartz litharenite
- The west member, mostly felsic volcaniclastite comporting chert and manganese units

7.2 LOCAL AND PROPERTY GEOLOGY

The following sections have been summarised from Wolfe et al. (2016).

The Project is situated on the western flank of the Bougouni Basin, composed primarily of sedimentary rocks with minor tholeiitic volcano-sedimentary intercalations. The sediments were deposited in a broad trough during the early Birimian period.

The Bougouni Batholith, a large felsic intrusion, occupies the central part of the basin and is known to appear approximately 25 km northeast and southwest of the Project Area. Small intermediate intrusions, mainly diorite to granodiorite, also occur within the basin.

The terrane is intensely lateritised, and with the exceptions of the granitic rocks, the protolith is rarely identifiable in outcrops. Large laterite plateaus cover most of the area. The underlying saprolite is exposed below the plateau boundaries and is generally of a yellowish ochre colour, whitened in numerous places by intense kaolinisation, likely of hydrothermal origin. Rare outcrops occur where resistive quartz veins protected the host rocks, and the schistose nature of the sediments can then be observed. Drilling intersected interbedded greywackes, siltstones, and mudstones.

The saprolite, whilst variable in colour from purple to brown, orange, cream and white, shows only very slight variations from what is now a clay (mudstone precursor) to a fine silty clay (fine siltstone precursor). There are no marker horizons, and no sedimentary features are preserved. The deformation intensity of these metasediments is moderate. Regional foliation is moderate and often not recognised in the saprolite, and while shear zones occur at Kobada, these tend to be discrete structures 5 cm to 50 cm wide. These discrete shears often contain limonite rinds parallel to the foliation, and the mottle zone supergene alteration extends down these structures, probably indicating increased groundwater movement through these natural pathways.

The laterite horizon is typically 3 m to 4 m thick and generally presents a stark contrast to the saprolite.

The quartz veins at the Project Area strike and dip at various orientations and angles, and three broad populations have been identified:

- **20°NE population parallel to the regional foliation**: These veins are consistent between 10°NE and 35°NE, dipping between 60°E and 90°E, and ranging from 5 mm to 1 m in width. They are often sheared (crack and seal), strongly brecciated, cemented with iron and manganese oxides, and mylonatised in places. They are associated with low-grade mineralisation (0 to 1 ppm Au).
- **E-W population**: These include strikes 45°NE to 135°NE and are concentrated between 80°NE and 110°NE, mainly dipping at 60°N to 90°N, with some dipping steeply south. They are 1 mm to 50 cm in width, pinch and swell, are relatively discontinuous, and can be sigmoidal. Stockwork zones up to 3 m in width can be formed. These veins display a fracture cleavage, which is commonly stained with red iron oxides. They are often surrounded by 5 mm to 10 cm wide limonitic alteration zones that are often wider than the veins themselves. Locally, they are folded in open folds. The veins crosscut the foliation-parallel veins and are not as intensely deformed as the latter. They are well mineralised (1 ppm to 17 ppm Au within the mineralised envelope). These veins may have formed as extensional fractures and/or Riedel shears in the Kobada Shear Zone, with a possible dextral (right lateral) shear sense.
- **Sub-horizontal population**: these veins display dips ranging from 0° to 30°, with varying strike directions. The veins vary from 1 mm to 10 cm in thickness and often occur as stockworks and ladder vein systems. They can form long continuous cross-cutting features and are moderately mineralised (1 ppm to 2 ppm Au within the mineralised envelope) with barren stockwork zones.

7.3 MINERALISATION

Gold at Kobada is present in the laterite, saprolite, unaltered rock as sulphides, and in the quartz veins. There are also placer-style deposits in the region, although these have largely been exploited by artisanal miners. Gold mineralisation was coeval with the hydrothermal events that introduced the regionally common quartz veins. The 20°NE structures are the only regional structures that have been identified on the property, the E-W and low-angle features seem to be confined to the mineralised zone in between the discrete shear zones.

Mineralisation at Kobada extends for a minimum strike of 4 km and is associated with narrow, irregular, high-angle quartz veins and with disseminated sulphides in the wall rock and vein selvages. Mineralisation occurs as free gold, whereas in sulphides mineralisation includes the occurrence of arsenopyrite, pyrite and rarely chalcopyrite. Visible gold is not common. Arsenopyrite (up to 5 mm) is localised near vein selvages and as fine-grained disseminated patches within the host rock. Pyrite occurs in finely disseminated patches within the host rocks and as euhedral crystals in the black shale, generally as traces up to 3 % by volume with up to 10 % locally in the wall rock at centimetre-scale intervals adjacent to the quartz veins.

Veins have a milky white colour and are generally discordant, with a thickness ranging from millimetric to sub metric. Mineralised veins are narrow, high-angle quartz veins, either cross-cutting another vein or the main fabric. This indicates that more than one generation of quartz veining is present, with a later phase resulting from remobilisation of gold mineralisation from an earlier hydrothermal event.

8 DEPOSIT TYPES

8.1 MINERAL DEPOSITS BEING INVESTIGATED

Three types of gold occurrences may be expected at Kobada:

- Primary lode gold mineralisation, associated with quartz veins and fault zones related to one or several Eburnean deformation phases (orogenic gold, shear zone gold or mesothermal gold deposits). Deposits of this type occur in the surrounding area.
- Lateritic deposits, resulting from the climatic weathering and alteration of primary deposits with diffusion of the gold into mushroom-shaped red oxidation zones (saprolite).
- Placer deposits, where the gold is associated with large angular fragments of quartz and intensely silicified pyritic rocks in alluvial sand and gravel horizons usually several metres in thickness. These fragments are derived from proximal lode deposits.

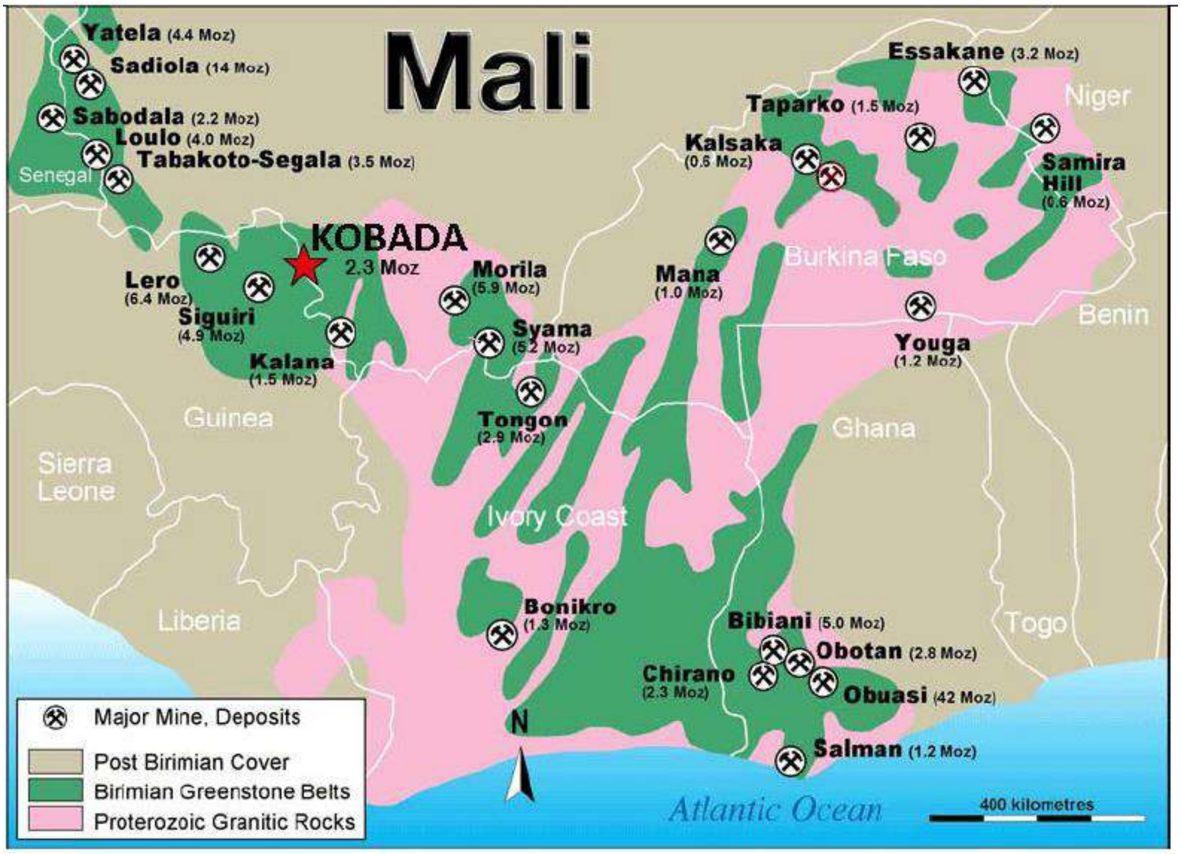
Placers are present on the Project and have been worked by artisanal miners for a long time.

The Kobada gold deposit is a quartz-carbonate veined mesothermal orogenic gold deposit hosted within a greenstone belt. It is located in arenites affected by a geological structure that is oriented northeast along the border of an intermediate intrusive that has basic components. Mesothermal veins are formed at moderate temperatures and pressures, in and along fissures or fractures in the rocks. They are known for their large size and continuation to depth and, therefore, are a major source of the world's gold production. Veins are usually less than 2 m wide and often occur in parallel sets. Typical mineralisation includes the sulphides chalcopyrite, sphalerite, galena, tetrahedrite, bornite and chalcocite. Gangue includes quartz, carbonates, and pyrite.

Greenstone-hosted orogenic deposits often represent > 10 Moz deposits. The quartz-carbonate veins in these deposits typically combine laminated veins in moderately to steeply dipping reverse shear zones with arrays of shallow-dipping extensional veins in adjacent competent and lower strain rocks. The reverse character of the shear-zone-hosted veins and shallow dips of the extensional veins attest to their formation during crustal shortening.

In greenstone belts, the significant vein deposits are typically distributed along specific regional compressional to transpressional structures. Owing to their association with regional structures, they are also located at the boundaries between contrasted lithologic or age domains within the belts. Along these structures, the deposits commonly cluster at localised bends or major splay intersections, and where deposits typically occur in associated higher-order structures. The larger deposits are commonly spatially associated with late conglomeratic sequences.

At local scale, favourable settings for these deposits represent a combination of structural and lithological factors. Favourable structural settings are linked mainly to the rheologic heterogeneities in the host sequences. Shear zones and faults are developed along lithological contacts between units of contrasting competencies and along thin incompetent lithological units. Along these contacts and along incompetent rocks, deposits will preferentially develop at bends and structural intersections. Competent rock units enclosed in less competent rock units favour fracturing and veining. Common lithological associations include Fe-rich rocks such as tholeiitic basalts, differentiated dolerite sills and Banded Iron Formations (BIFs), and competent porphyry stocks of intermediate to felsic composition, whether they intrude mafic, ultramafic volcanic or clastic sedimentary rocks. Figure 8.1 shows the regional occurrence of greenstone belts and associated gold mines.



Source: Wolfe et al., 2016

Figure 8.1: Regional Greenstone Belts and Gold Mines

8.2 GEOLOGICAL MODEL

Four factors were considered during the process of geological modelling:

- Structural model
- Grade model
- Lithological model
- Weathering model

8.2.1 Structural Model

Structural information was available from regional geophysics, and previous interpretations provide detail for the construction of the geological model. Historical orientation data was available for trenches and relogged drillholes.

The major structures affecting the deposit were modelled from the data and regional interpretations. The presence of major structures was deduced from the displacement of the grade model (see Section 8.2.2) as shown in Figure 8.2. These major structures were used in the geological model (see Figure 8.3). Only two major structures affect the characteristics or cause displacement of the deposit (see Figure 8.2) and were modelled.

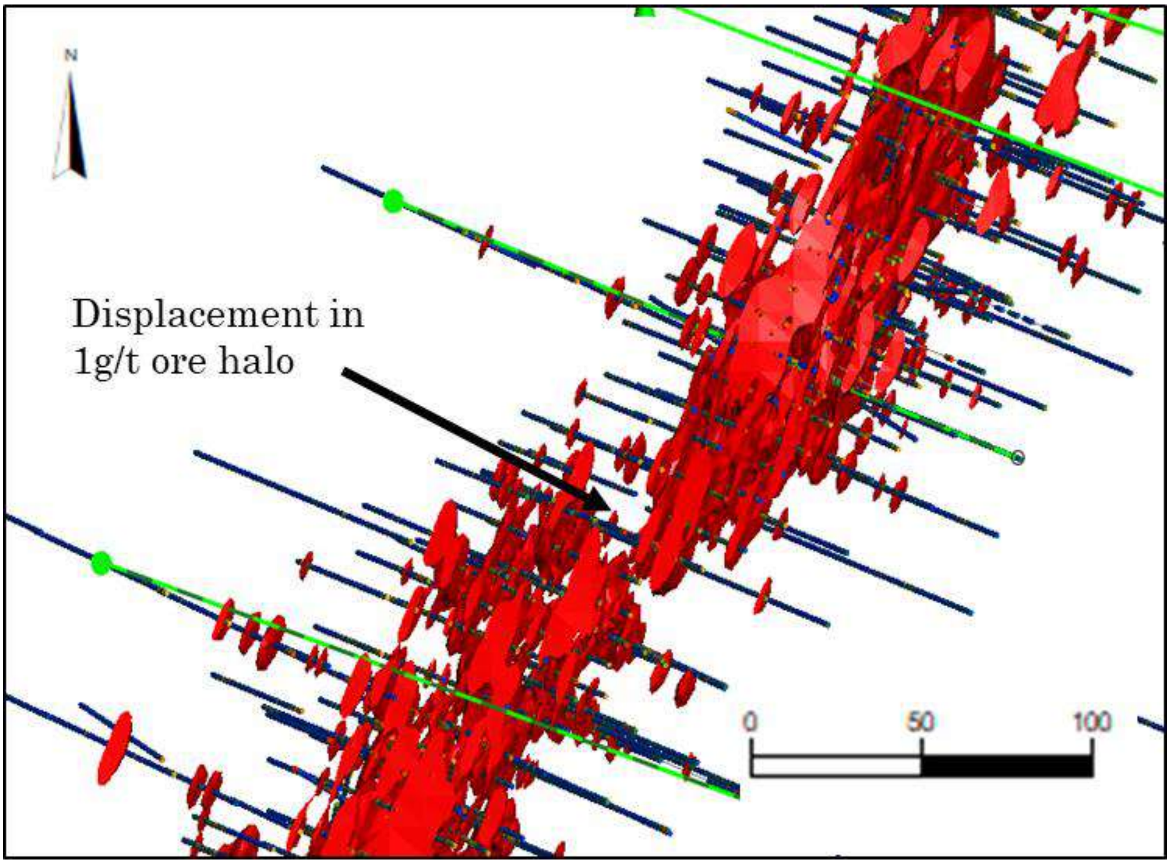


Figure 8.2: Translation in Ore Shell

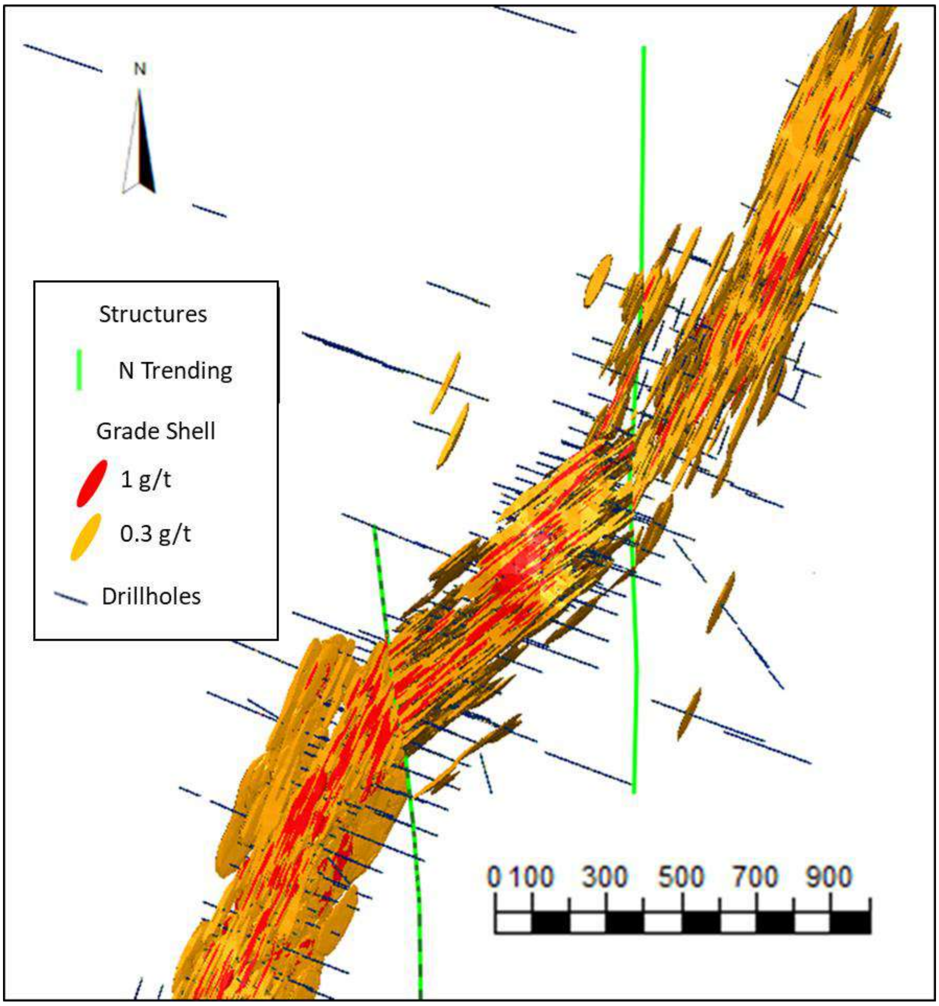
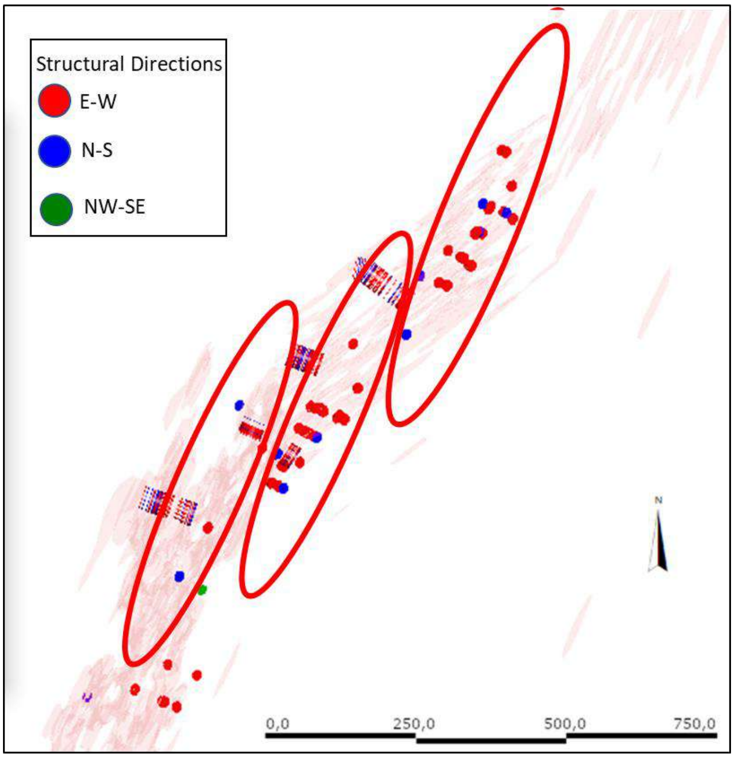


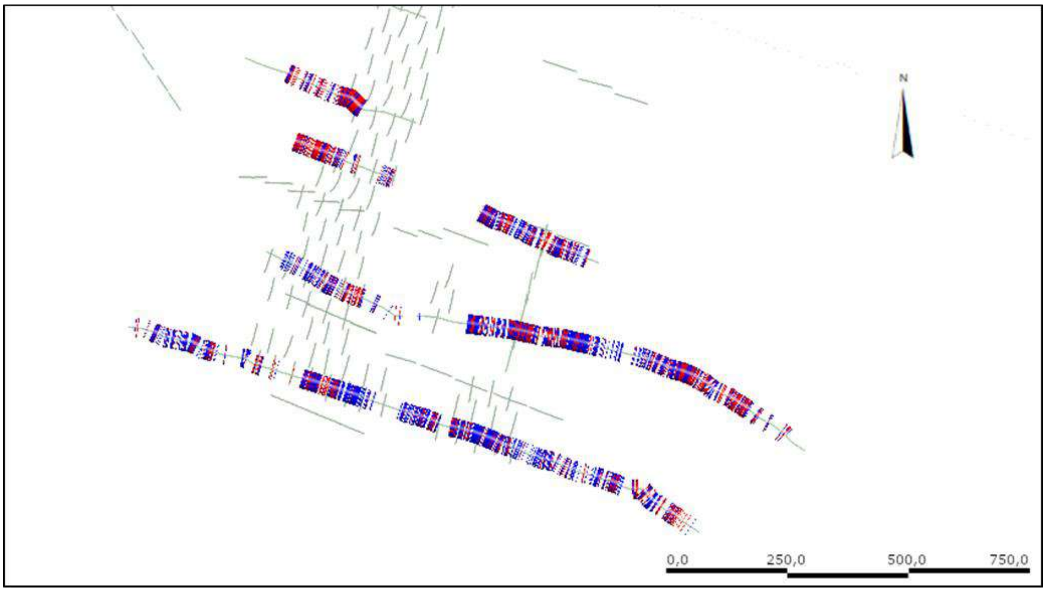
Figure 8.3: Major Structures Defined

The core bedding angle (CBA) information (352 observations with descriptions) was measured during the relogging of 180 drillholes. These observations cover the central area of the deposit. Most observations were for N-S or W-E structures. The N-S trending structures typically had a steeper CBA, while the W-E structures were flatter. All the descriptions of structural directions were derived from the historical database and trenches. Observations from recent drilling show similar orientations to those in the trench data.

The CBA data from the trench data shows a mixture of W-E and N-S dipping directions. The plan view in Figure 8.4 shows a close association of N-S (blue) and W-E (red) dipping directions. The pattern of the CBA data was compared to that of the major structures to aid correlation.



Trenches and Drillholes



Trenches only

Figure 8.4: Trench and Drillhole Orientations over the Southern Area of Kobada

The CBA orientations from the drillholes and trenches are plotted in Figure 8.5 to Figure 8.7. For the trenches, the N-S orientation is dominant. This reflects what is seen in the structural interpretations shown in Figure 8.3. The dips of the quartz veins are dominated by 45° to 60° in the trenches. The N-S CBA directions are dominant for the drillholes, in agreement with the trench data (see Figure 8.7); however, this is shown for a limited data set.

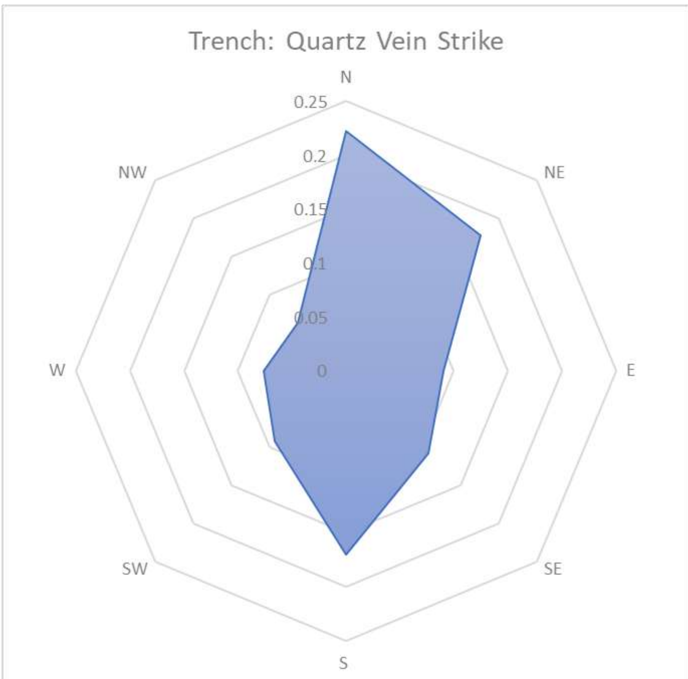


Figure 8.5: Orientations from Trenches

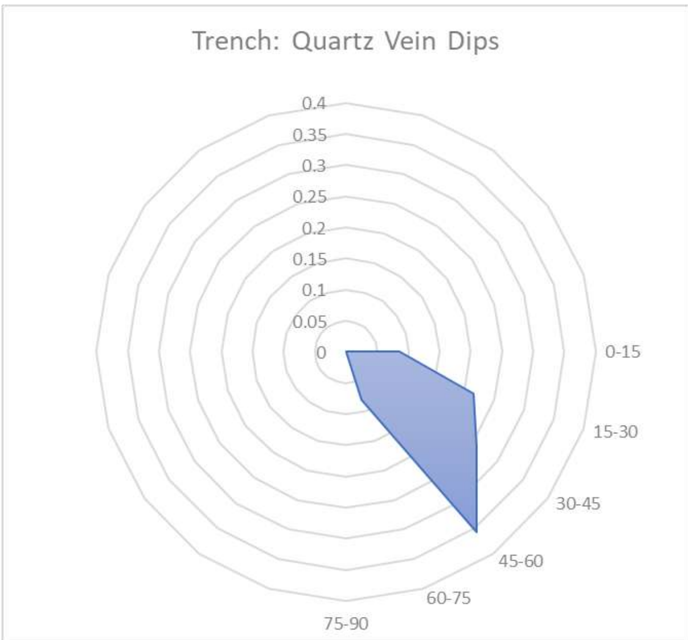


Figure 8.6: Quartz Vein Dips from Trenches

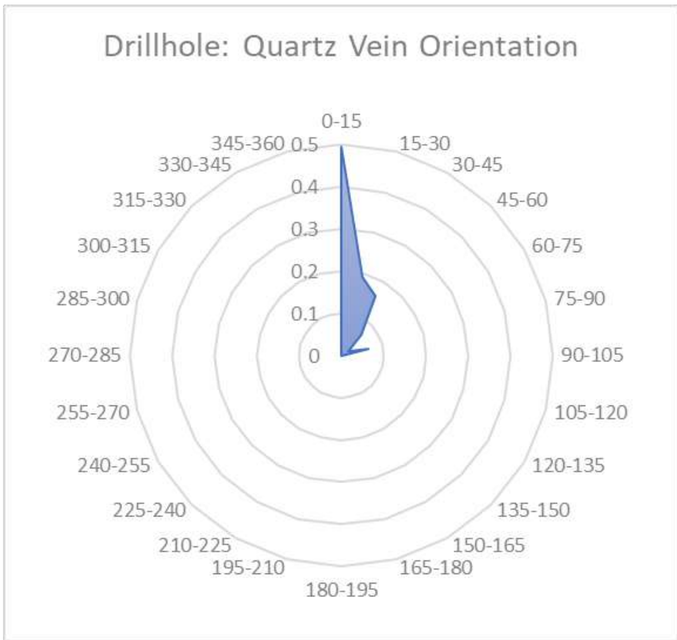


Figure 8.7: Orientations from Drillhole Logs

As part of the data analysis of the existing data set, the available grade that was associated with noted directions was correlated to see if any trends existed. This is a guide to aid modelling, as well as to check the directions inferred in the grade shells. The actual continuity of the grade shells and search ranges to be used in the estimation are informed by variograms. The grades were compared to the different orientations to establish if any orientation was associated with grade trends. For logging, a slightly higher grade was seen N-S; however, it is noted that these were very few samples (see Figure 8.8 and Figure 8.9). For the trenches, there were a lot of samples, and the mean grades were comparable, as was the distribution of data (see Figure 8.10 and Figure 8.11).

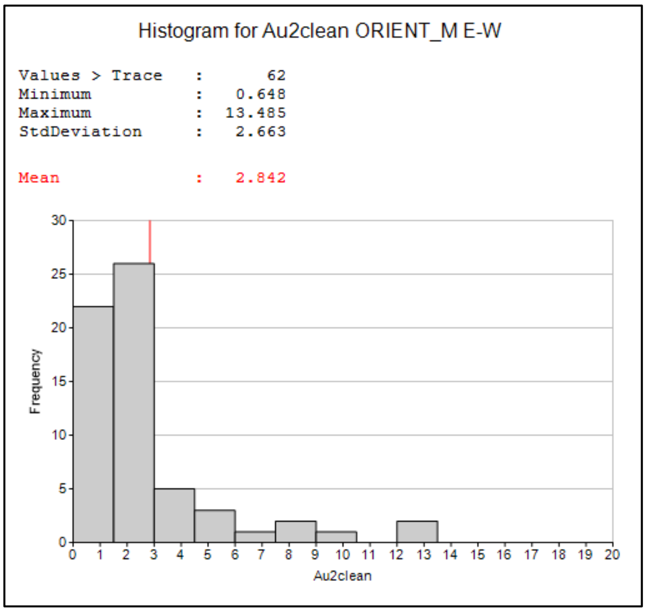


Figure 8.8: Histogram of W-E Orientated Drillhole Samples

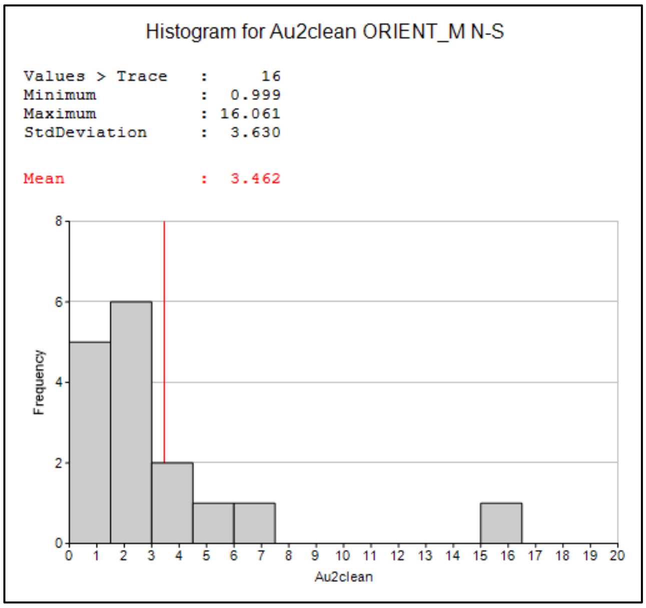


Figure 8.9: Histogram of N-S Orientated Drillhole Samples

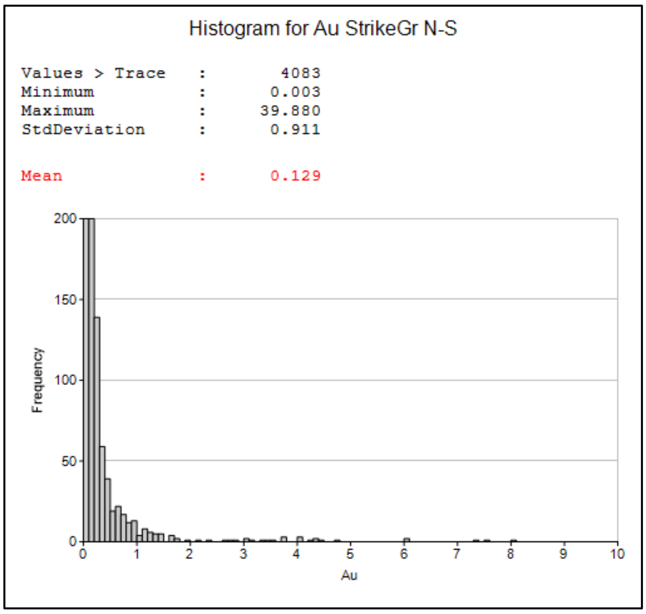


Figure 8.10: Histogram for Trench Values Trending N-S

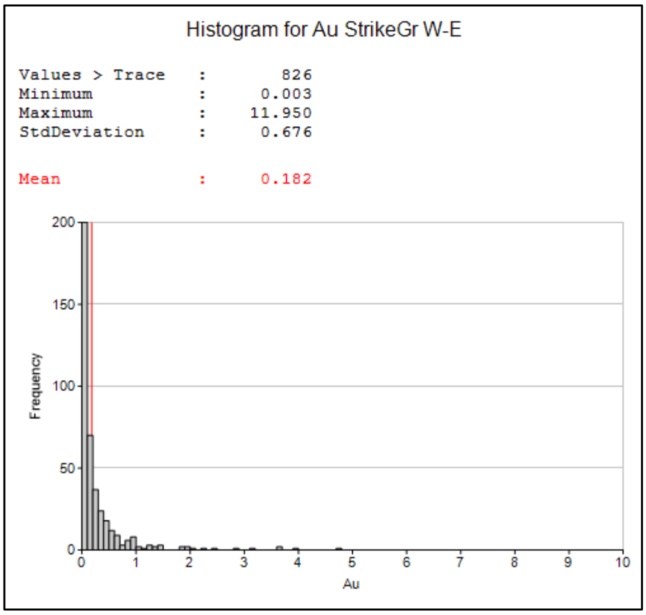
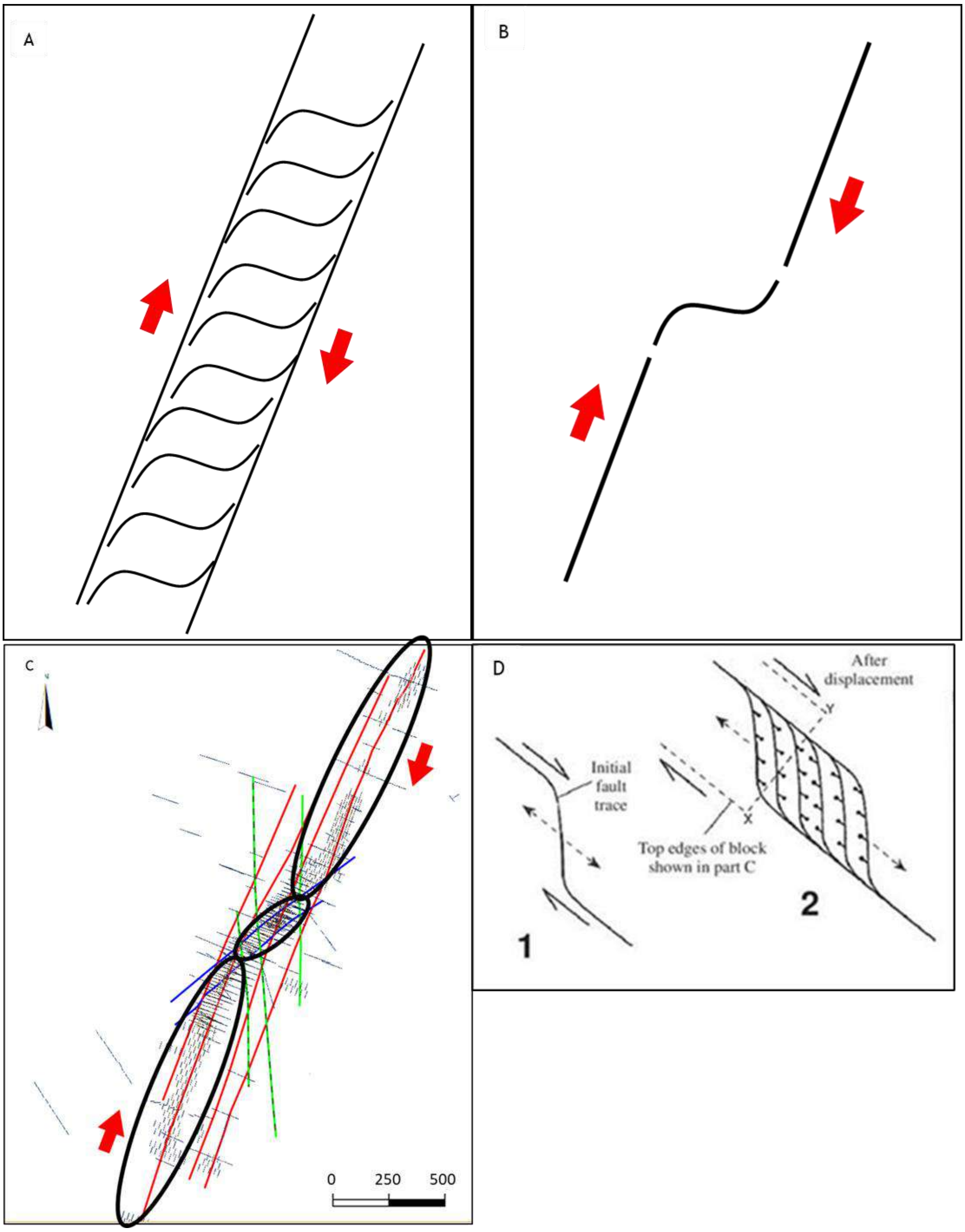


Figure 8.11: Histogram for Trench Values Trending W-E

In conclusion, the observations indicate a larger zone within the more structurally disturbed areas. Orientations agree with regional trends and the major structures already defined. This suggests that there is a presence of NNE-SSW orientated structures (regional structural direction), with closely associated E-W shears. These structures typically form part of a larger zone, which is characterised by mineralisation. Regional structures can thus be used as a guide to orebody orientation and to aid in the correlation between drillhole intersections.

Figure 8.12A shows the main N-S orientated shear zone and associated W-E *en-échelon* structures. There is also the potential that the central domain is one of these larger *en-échelon* structures (see Figure 8.12B) as opposed to an actual displacement of the existing main shear. Both these interpretations would fit in well with what is observed over Kobada (see Figure 8.12C), and further work will assist in defining the actual structural make-up of this area. Figure 8.12D explains the structures seen at Kobada resulting from dextral shear, opening up fractures for vein and mineralisation emplacement. This zone of shear fractures has also been described as an extensional or contractional duplex by Woodcock and Fischer (1986).



Source: Twiss and Moores (1992)

NOTE: A, B, and D show the principles of right lateral displacement. C displays the situation at the Kobada Main Shear.

Figure 8.12: Structural Interpretation at Kobada

The structural model utilised in the parameters for the Leapfrog Geo modelling process is therefore a dextral shear system, common for the Birimian Greenstone Belt in West Africa, with the principal shear orientations trending NNE, roughly E-W striking Riedel shears and extensional veins, and approximately north-striking conjugate sets. The first two shears and veins are the most prospective as these open up for gold-bearing fluid ingression during the deformation phase. The central domain of the structural model (see Figure 8.12C) could be interpreted as a dilational jog (releasing bend).

8.2.2 Grade Model

Grade shells were generated in Leapfrog Geo software to define the extent and limits of the mineralisation of interest, by defining the boundaries of what is potentially mineralised, as well as defining grade continuity.

The expected orientation of the grade continuity was known; however, the data set was considered afresh, with no preconceptions. Omnidirectional grade shells at 0.3 g/t and 1 g/t were generated to allow the data to initially develop the directions of inherent continuity without manual intervention. Different domains were identified with differing dips and strikes from the grade shell pattern. The dominant orientations are very apparent and typically continuous within each domain (see Figure 8.13).

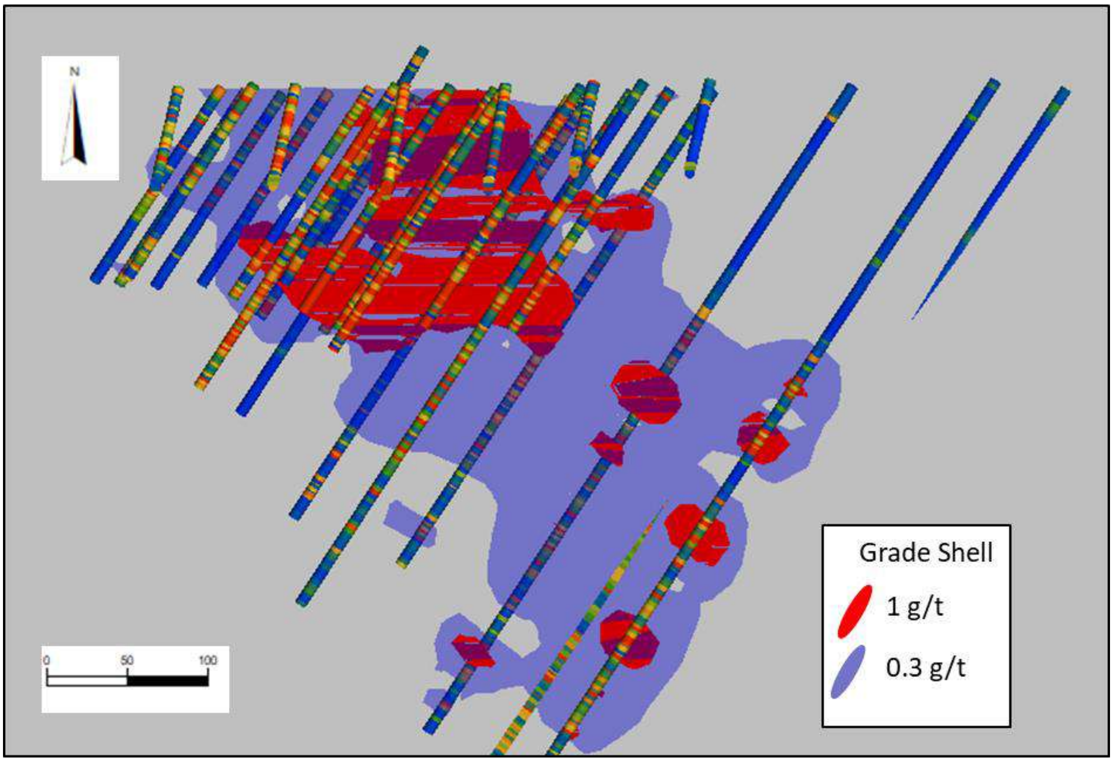


Figure 8.13: Omnidirectional Grade Shells

After interpreting the directions from the omnidirectional grade shells, the structural pattern was used to define the strike/main direction of the trend. The dips were defined by iteratively applying differently orientated anisotropies to optimally identify the directions providing the best continuity and connectivity between intersections. The best-fit planes for the north and central domains dip at 78° to the ESE and SSE, respectively. The best-fit plane for the south domain dips at 50°E.

Various test orientations were applied, based on structural observations from the trenches and drillholes. One of these included a dominant W-E orientation that was observed in the trenches, but this revealed little grade continuity.

Grade shells were created to the maximum variogram range with no compositing; therefore, all the data was honoured implicitly. All the parameters used in the creation of the grade shells are shown in Table 8.1. The range used in the interpolant was twice the variogram range that was used in previous estimates of the Project Area. All the outliers producing volumes < 25,000 m^3 were excluded. Future Inferred Mineral Resources or exploration targets may expand the ranges and include these outliers. Various iterations of grade shells were tested to determine the best set-up, the search parameters, and how well the result honoured the data. The final model produced has the best combination of connectivity of data, trend, and continuity, as well as fit to data. The model shown in Figure 8.14 to Figure 8.17 represents the high confidence model, honouring the data very closely.

Table 8.1: Indicator Function Settings used in Leapfrog Geo

Description	Parameter	Domain1	Domain2	Domain3	Domain4	Domain5
Variogram	Range	2X variogram range	2X variogram range	2X variogram range	2X variogram range	2X variogram range
Cut-off grade	Grade (g/t)	0.3	0.3	0.3	0.3	0.3
Interpolant settings	Interpolant Range (m)	130	130	130	130	130
	Type	Spheroidal	Spheroidal	Spheroidal	Spheroidal	Spheroidal
	Drift	None	None	None	None	None
Orientation	Dip (°)	50	78	78	78	78
	Dip-azimuth (°)	114.8	142	114.8	114.8	98.95
	Pitch (°)	0	0	0	0	0
Ellipsoid ratios	Maximum length (m)	8.5	8.5	8.5	8.5	8.5
	Intermediate length (m)	5	5	5	5	5
	Minimum length (m)	0.8	0.8	0.8	0.8	0.8
Settings	Exact Clipping	yes	yes	yes	yes	yes
	Resolution (m)	10	10	10	10	10
	IsoValue	0.1	0.1	0.1	0.1	0.1

The northern domain is generated at a dip of 78°ESE along the NNE trending mineralised structures (see Figure 8.14).

The central domain dips at 78°SW orientated along the NE trending mineralised structures (see Figure 8.15).

The southern domain (see Figure 8.16) dips 50°E along the NNE trending mineralised structures.

The three domains combined are shown in Figure 8.17.

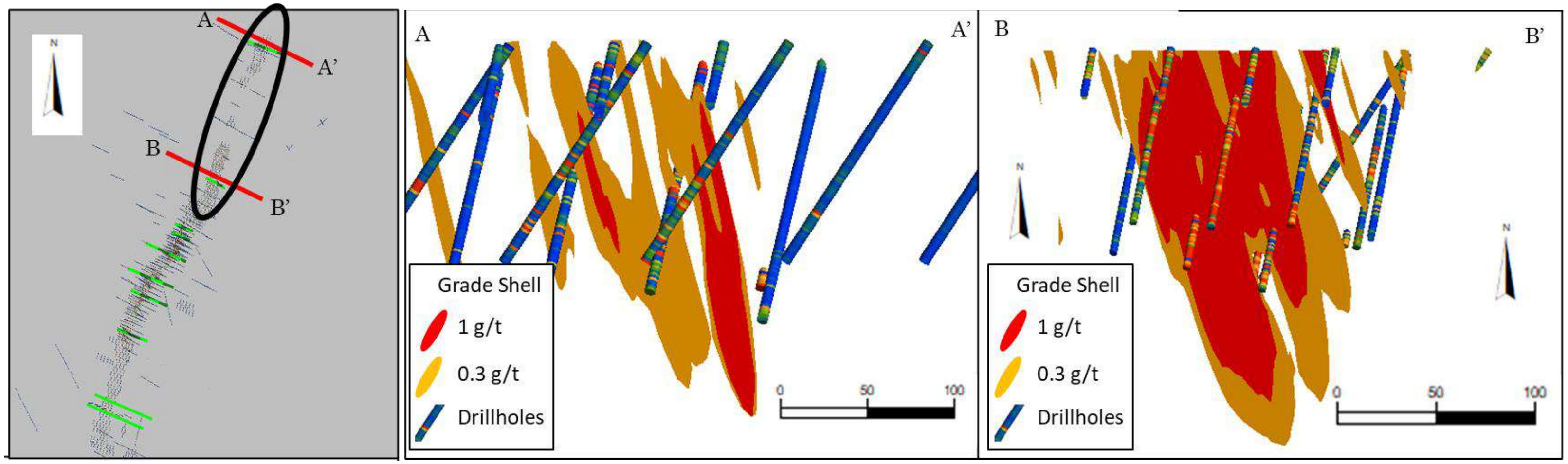


Figure 8.14: Northern Domain Sections

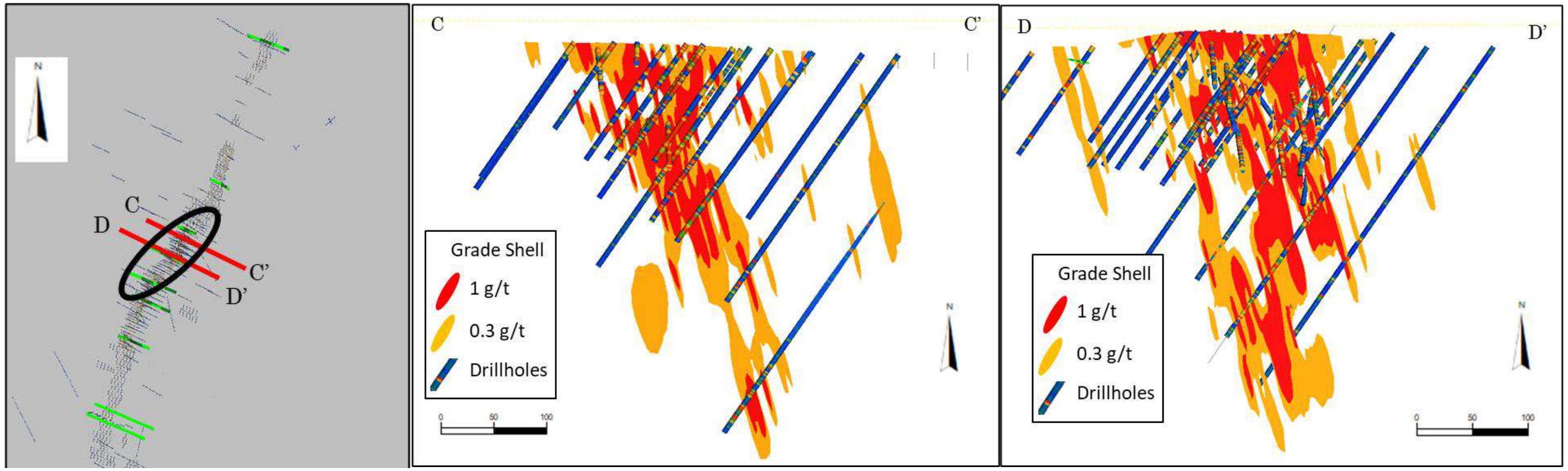


Figure 8.15: Central Domain Sections

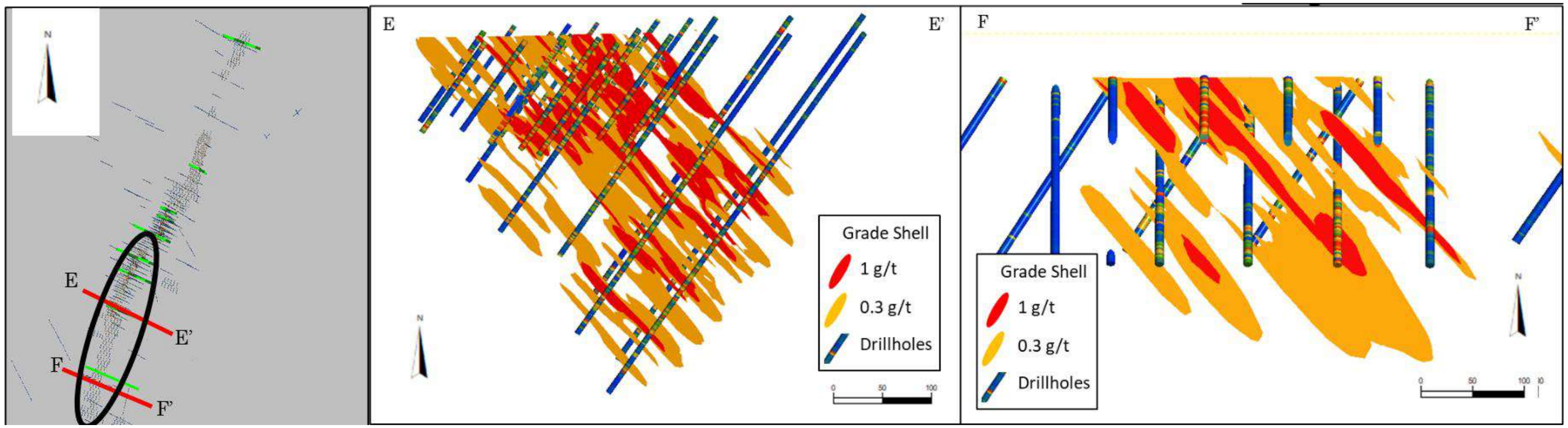


Figure 8.16: South Domain Sections

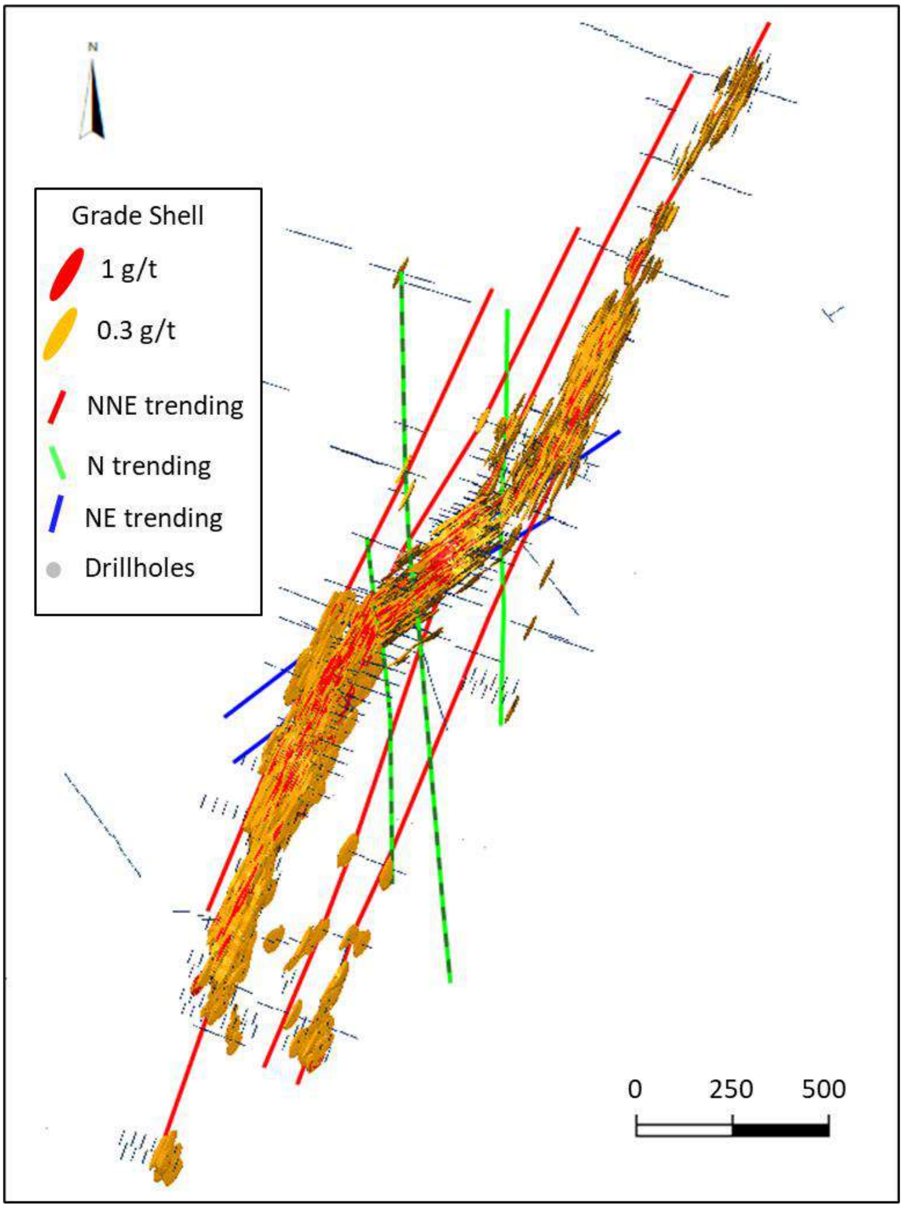


Figure 8.17: Combined Grade Shell Model

Representative sections across the grade shells are shown in Figure 8.18.

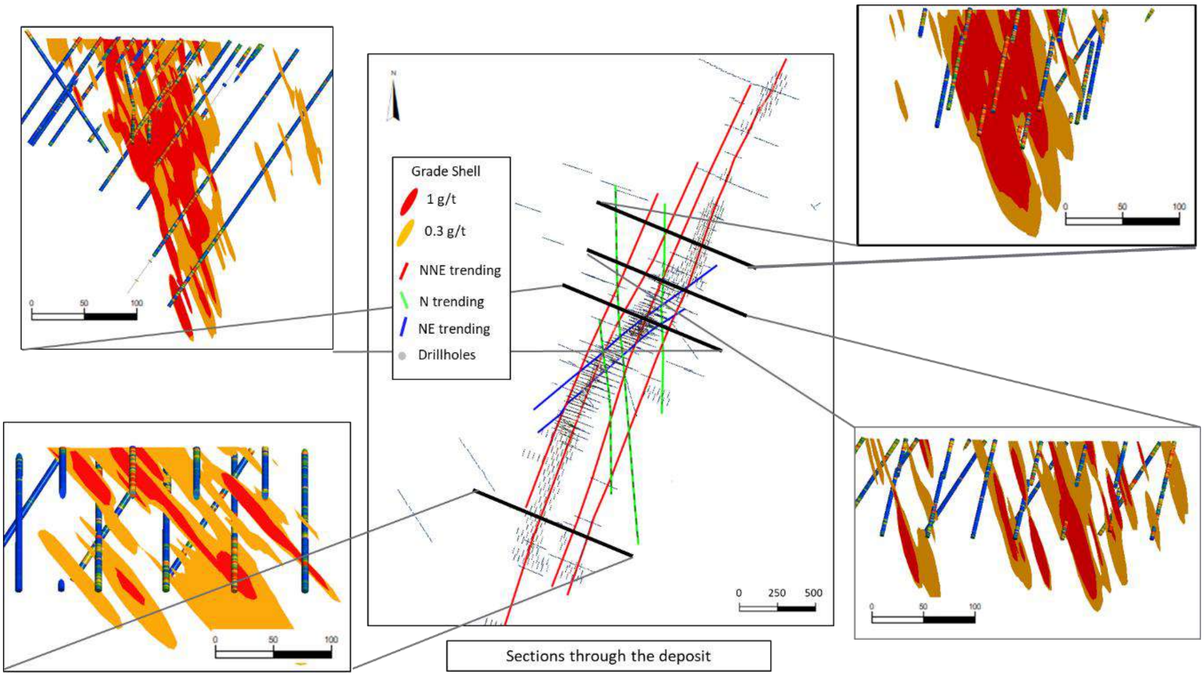


Figure 8.18: Sections through the Kobada Deposit

Various refinements of the base-case grade shell model were tested to develop a grade estimation strategy. The final set-up utilised to generate the numeric function in Leapfrog Geo is shown in Table 8.1. It is important not to artificially restrict the search volumes and risk underestimating the continuity of the mineralisation. These refinements used similar parameters as those for the original grade shells, while honouring the data and conforming to the observed trends. The data search was expanded to two and three times the variogram range, and compositing and other options were adjusted. The 2X variogram range was used as the final model, as it had limited changes to the width of the deposit, and instead elongated along the strike of the orebody (N-S) as well as down dip. This also resulted in better connectivity between sparser data points, especially in the north where there were some gaps in the models (between data). The extrapolation of the orebody was limited by the set-up criteria in the modelling. This was also accounted for in the Mineral Resource classification where the number of data points required to estimate a block, as well as the distance to samples, was considered in the classification. Thus, a block very far from the data would be downgraded.

A final check on the quality of the grade model and its correlation with the original data was to query the samples inside and outside the domain. It is expected that the bulk of the grade will be seen within the domain, with a decrease in the grades outside the domain indicating that most samples below the cut-off grade are excluded from the domain. Figure 8.19 shows the total orebody, with the sample average grade below 0.25 g/t outside the domain. The same process was repeated for the samples outside the domain that showed spikes in grade occurring only once within the mineralised volumes (see Figure 8.20). As the drilling results were updated during the Phase 1 and Phase 2 drilling (2019) and later the 2020 drilling, the model was adjusted with the additional data. These additional results confirmed the results in areas that were already well informed and validated the continuity that was inferred from the historical data set.

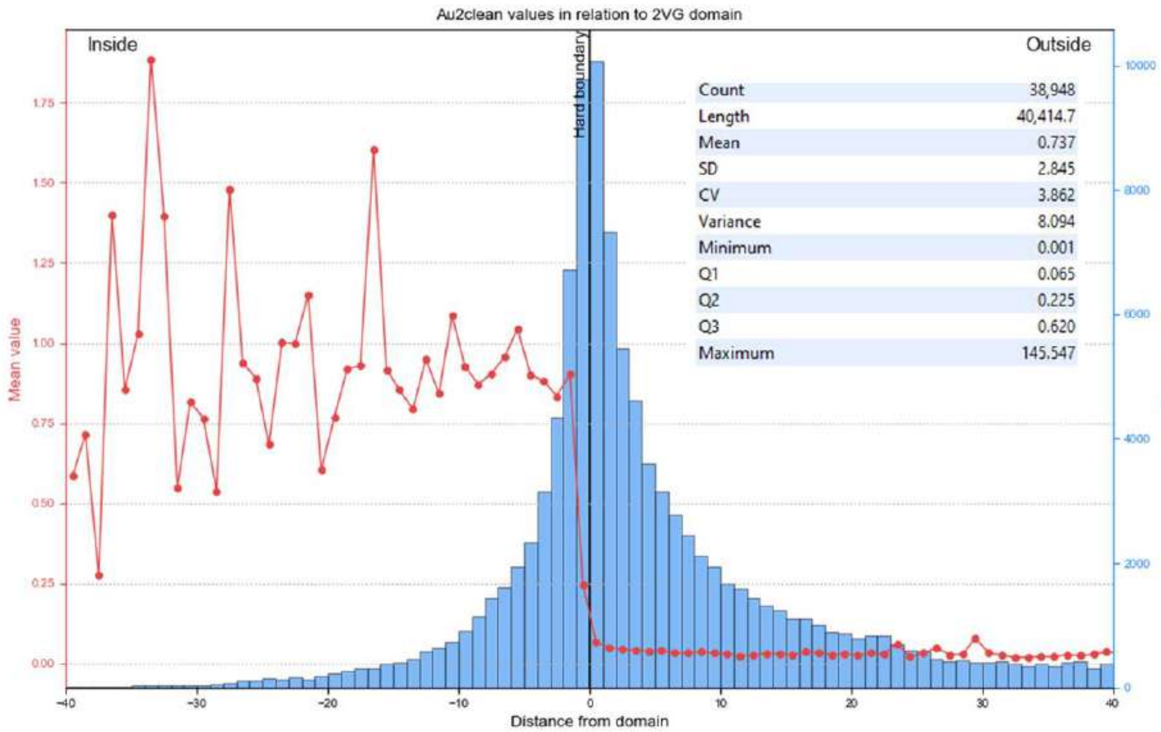


Figure 8.19: Au Values within the Total Combined 0.3 g/t Orebody

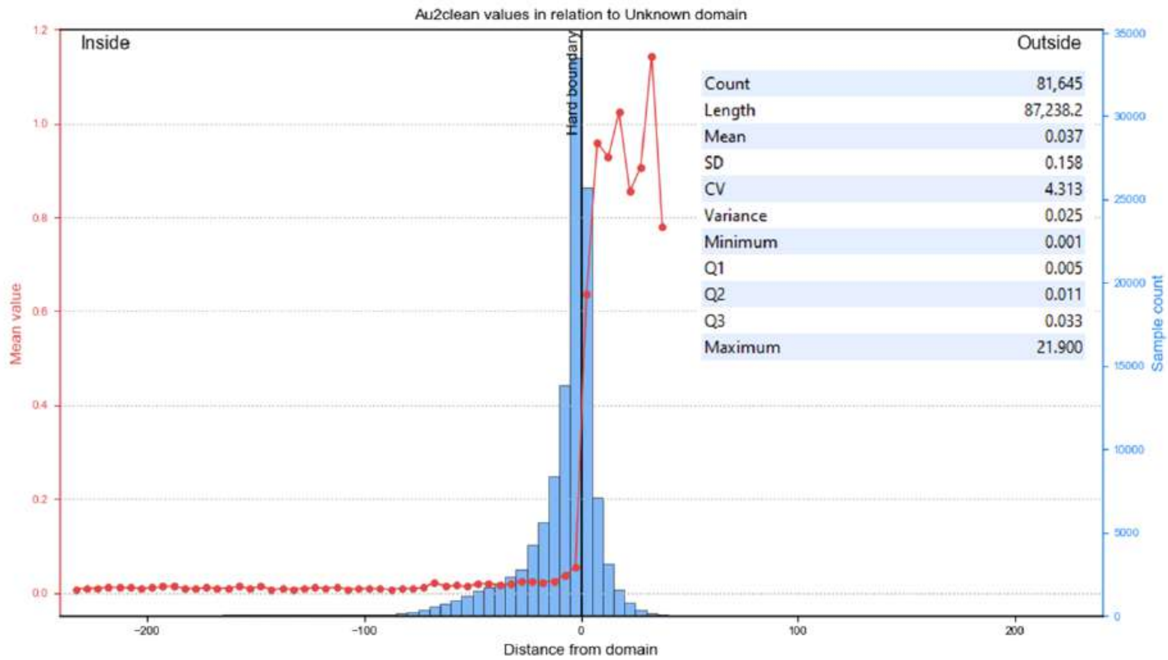


Figure 8.20: Au Values Occurring Outside the 0.3 g/t Ore Shell

8.2.3 Lithological Model

All the lithological information available in the historical database (pre-2018) was reviewed to define a separate lithological model in addition to the grade shell model.

The samples available per lithotype, as well as the grade associated with each lithotype, are shown in Table 8.2. The regolith, vein, and weathered lithologies appear to host the bulk of the grade.

Table 8.2: Lithologies identified at Kobada

Lithology	Total Samples	Samples with Grade	Average Au
			g/t
Intrusive	117	117	0.21
Weathered	2,818	2,090	0.43
Regolith	6,617	5,929	0.37
Saprolite	13,415	12,967	0.27
Schist	8,703	8,702	0.14
Sediment	79,557	79,527	0.28
Vein	389	385	0.69
Unknown	8,267	8,267	0.14
Total	**119,883**	**117,984**	

The weathered lithologies are considered to be a separate model and are discussed in Section 8.2.4. Only the primary lithologies are considered in the lithological model.

Within these main lithologies, minor rock types are identified as discussed in Section 7.2.

The main rock types encountered are as follows:

- Ferricrete: a very irregular and minor rock type.
- Laterite: typically of uniform thickness and very consistent, occurs close to the surface and within most of the drillholes throughout the deposit.
- Clay: localised and does not occur throughout the deposit. The mixed saprolite-clay zone is interspersed with saprolite drillholes, showing that the descriptions of saprolite and clay are often mixed and vary from drillhole to drillhole; therefore, the clay and saprolite are better considered together.
- Saprolite: typically below clay in the historical logs, has a varying thickness and occurs within most of the drillholes throughout the deposit. More recent relogging has merged the clay and saprolite into only saprolite, so it is one combined description in the current logs.
- Metasediments: occur below the saprolite. Shale occurs irregularly, argillite occurs in a few drillholes within the siltstone while the sandstone is more irregular; however, the siltstone lies consistently over the sandstone throughout the deposit.

- Intrusives: constitute a very minor portion of the deposit; however, their location or occurrence may indicate an association with major structures or shears, depending on age relationships.
- Quartz veins; mineralisation-hosting lithology that is recognised within most drillhole logs to varying degrees, but the thickness and continuity of which differ greatly between logs.

An initial model was created based only on the primary lithologies, laterite, saprolite, siltstone and sandstone.

The historical logs were inconsistent, and it is thus difficult to create a meaningful model directly from the data. Consequently, additional work, including relogging of the historical core, was carried out. During this work, it became apparent that it would be more beneficial to model the weathering profile/weathering intensity than the primary lithologies. The description of weathering intensity was also typically available for historical as well as recent logs, while primary lithologies were not consistently recorded.

8.2.4 Weathering Model

A total of 137 drillholes were relogged in order to confirm the weathering profile encountered in the drillholes. In these logs, laterite, saprolite and fresh lithologies were identified in addition to quartz veins and their orientations. The laterite was taken as the uppermost oxide horizon (modelled as laterite), and the saprolite was taken as the oxidised horizon. A transition zone was also defined, where a mixture of oxidised and fresh lithologies was seen. Fresh rock was defined by the absence of any weathered lithologies (see Figure 8.21).

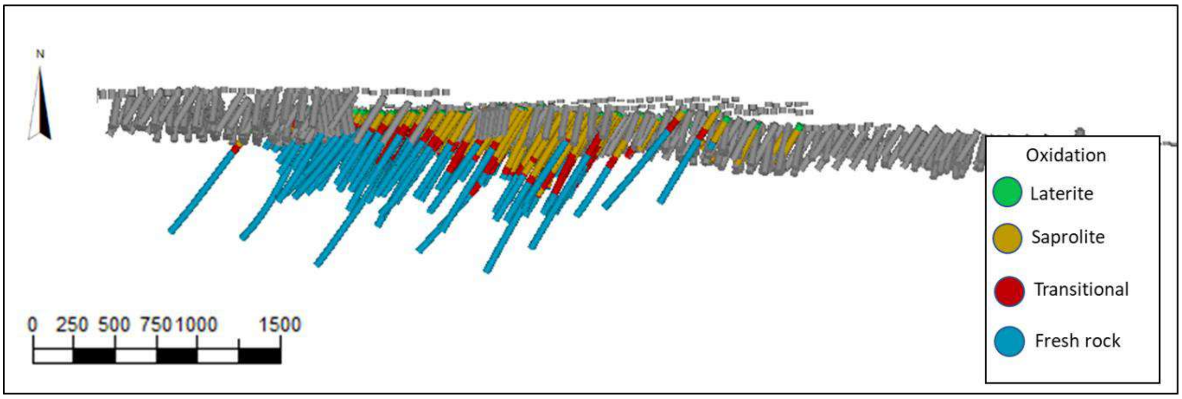


Figure 8.21: Weathering Profile Identified in Relogged Drillholes

The new logging was focused only on the central area of the deposit where the bulk of the diamond drillholes that were available were located. All the data from the relogging correlated very well, and the data was honoured explicitly (see Figure 8.22) in the modelling process. To the north and south of the centre of the orebody, where no new data was available, historical logs were used to define the weathering surfaces. These descriptions were inconsistent, so the intersections were manually selected. As new drilling results became available during 2019 and 2020, the new data was used to inform the model preferentially over less reliable and inconsistent historical data.

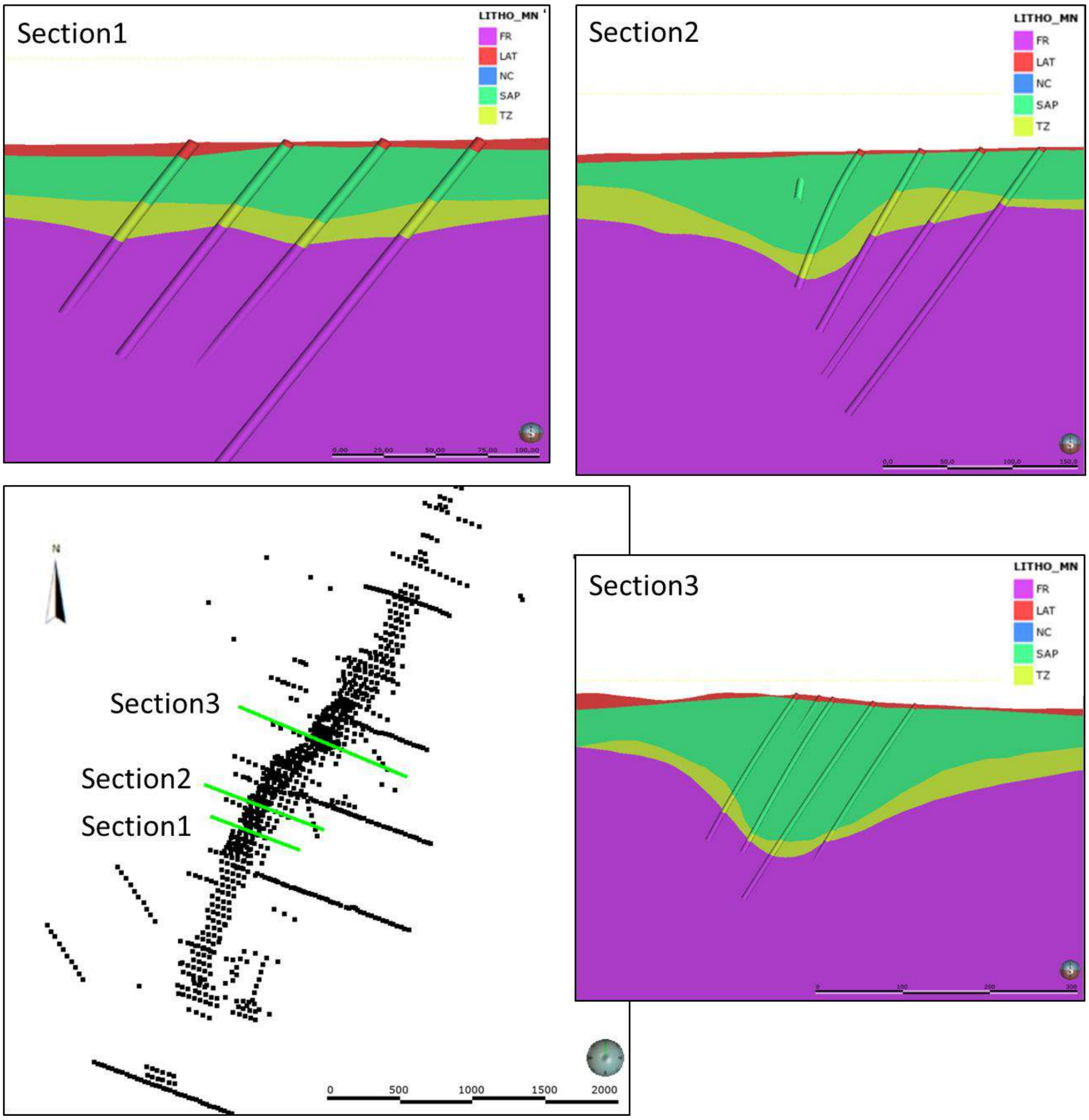


Figure 8.22: Sections through the Orebody Showing Volumes Defined from Drillholes

The weathering profile was combined with the grade shells to define the estimation domains. The 4 weathering levels were combined with the 3 structural domains, resulting in 12 mineral resource estimation domains (see Figure 8.23 and Section 13).

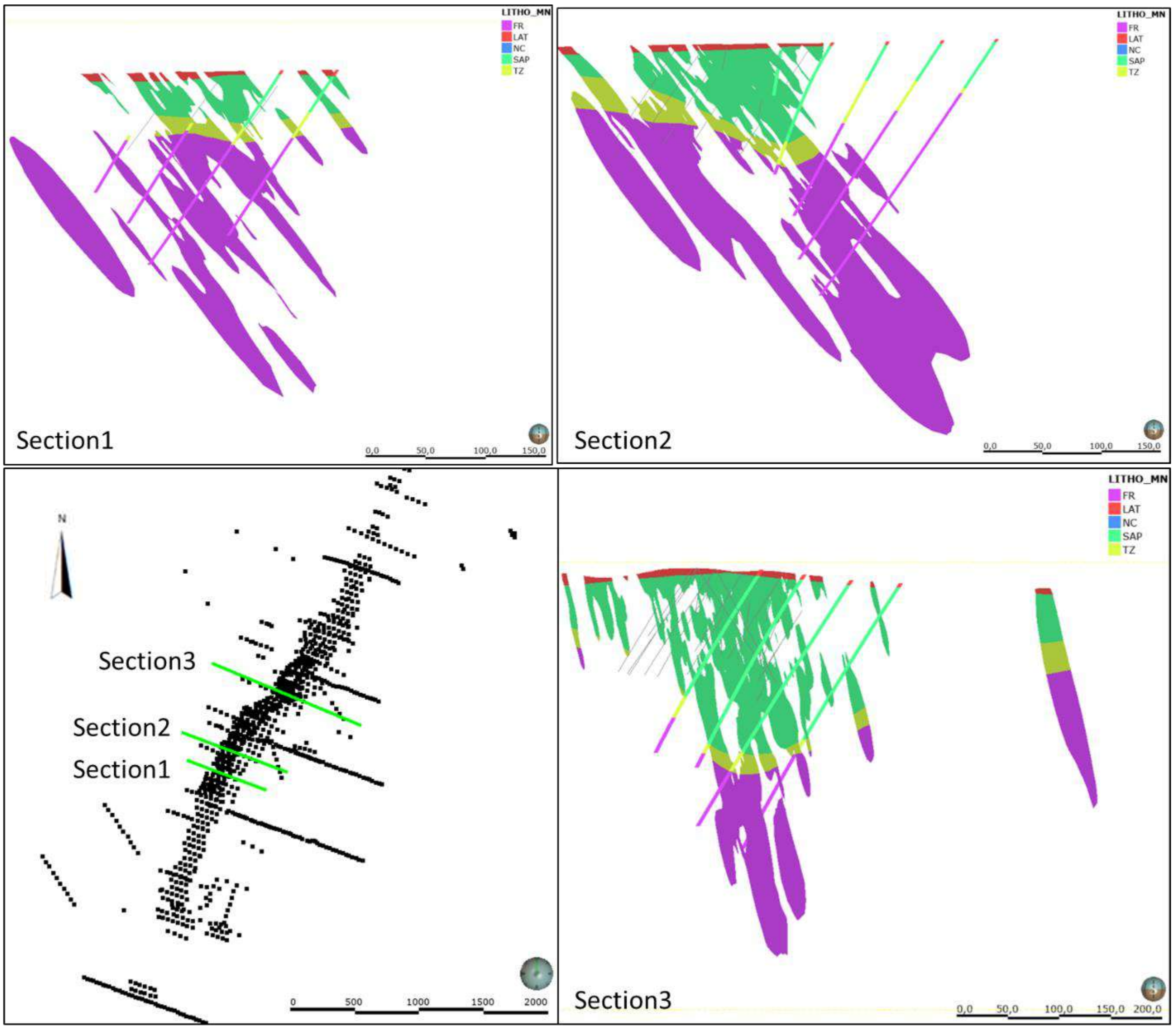


Figure 8.23: Sections through the Orebody Showing Combined Weathering Profile and Grade Shells

The model was updated as new data became available. The new drilling results during 2019 and 2020 confirmed the orientations of the grade shells. The original models did not require adjustment as the new results confirmed what had already been observed and modelled (see Figure 8.24). The sections in Figure 8.24 show the modelled weathering profile, as well as the grade shells that have been updated with the new drilling information.

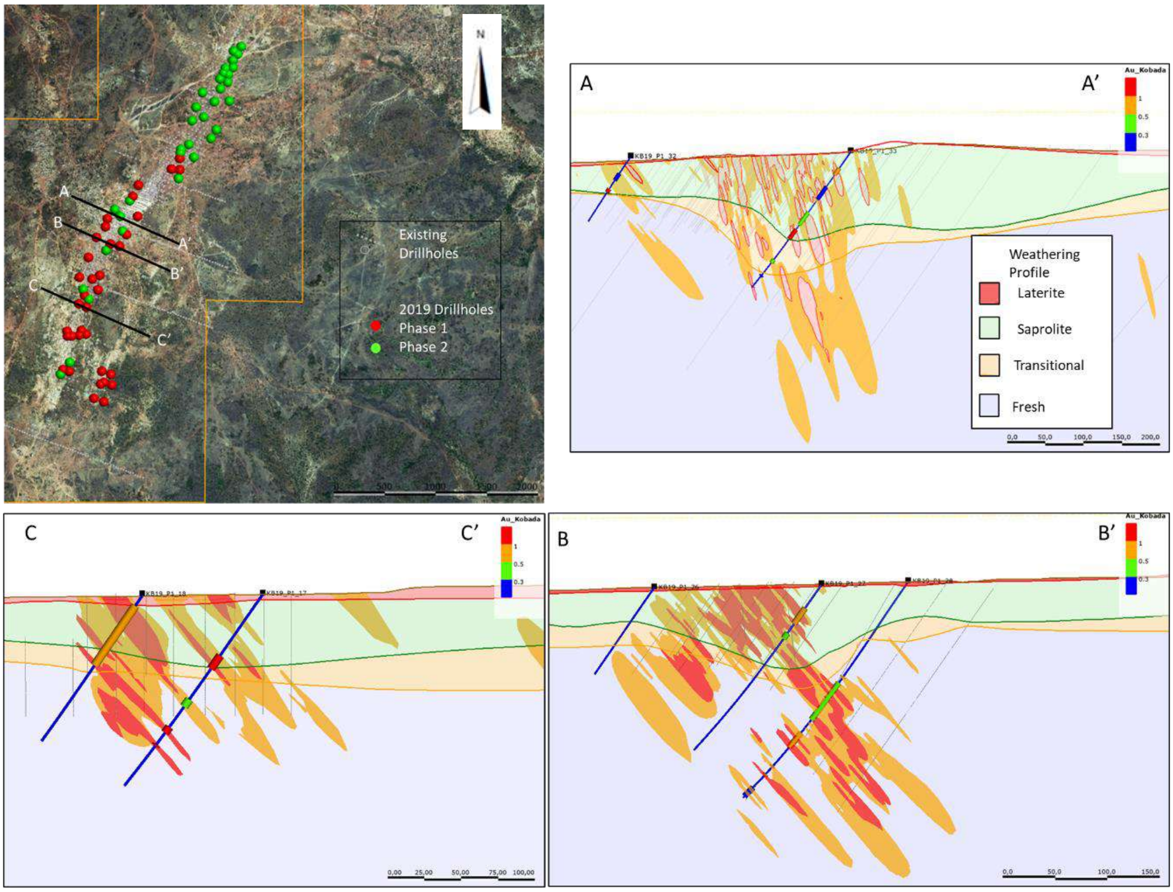


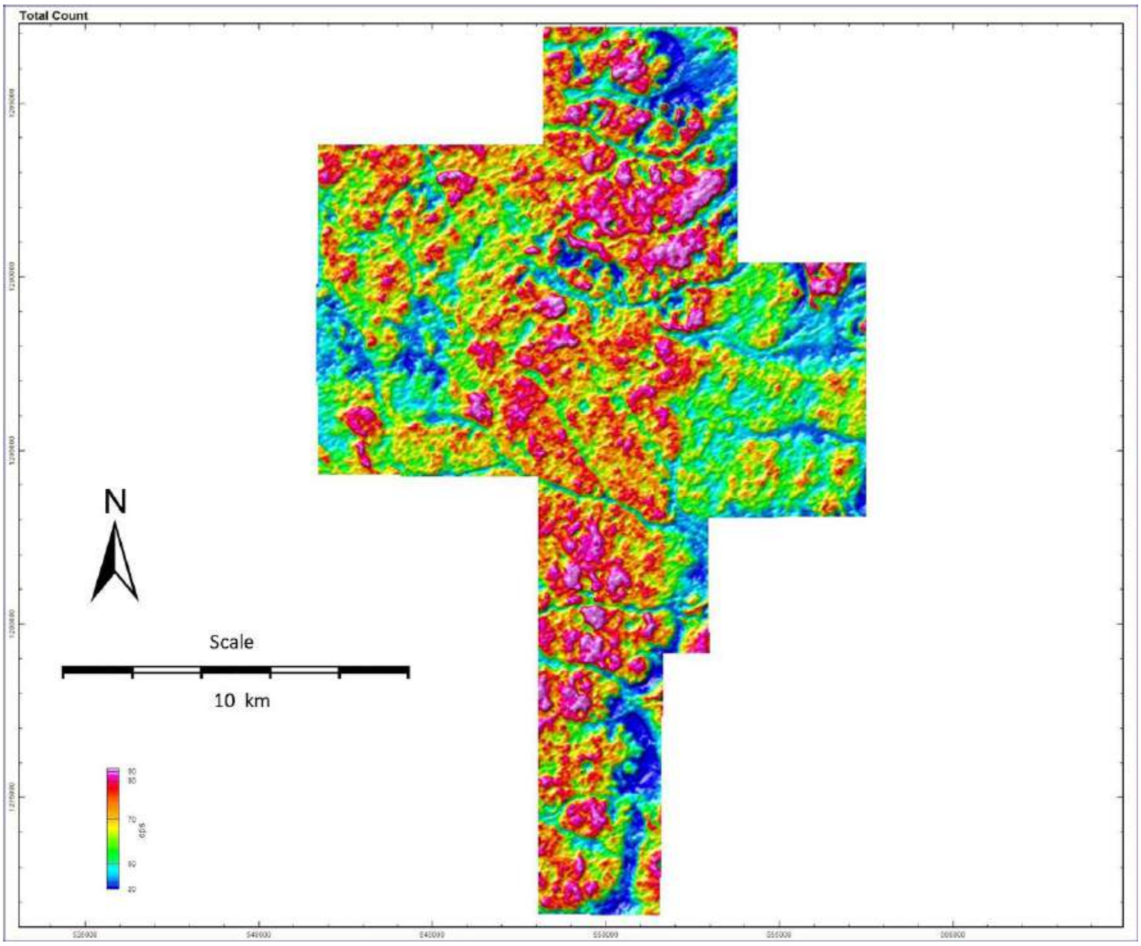
Figure 8.24: Sections through the Orebody Showing Weathering Profile and Grade Shells (Updated to Correspond to New Drilling Information)

9 EXPLORATION

9.1 SURVEY PROCEDURES AND PARAMETERS

In 2006, AGG conducted a 126 line-km induced polarised/resistivity survey to establish Induced Polarisation (IP) anomalies with known mineralised zones and to delineate discrete target potential associated with gold mineralisation.

In 2010, a high-resolution aeromagnetic survey was conducted by Xcalibur for AGG (see Figure 9.1). The airborne survey over the entire Kobada Project footprint (215 km^2) was conducted by a fixed wing aircraft, which flew a total of 4,700 line-km on E-W lines spaced 50 m apart with N-S tie lines spaced 500 m apart and with a ground clearance of 30 m. The airborne data was interpreted by Paterson, Grant & Watson Limited of Toronto, Canada, who indicated that the data was of high quality, and the interpretation resulted in the selection of seven new prospective targets.



Source: Xcalibur Airborne Geophysics report (2010)

Figure 9.1: Total Count Aeromagnetic Montage

A trenching programme (eight trenches for a total of 3,980 m) was completed in the Kobada South area. The trenches varied in depth from 0.5 m to 4 m. Two trenches totalling 96 m were dug by hand and two trenches totalling 247 m were dug using a Cat D7 dozer. All these trenches were mapped and sampled at 1 m intervals.

A reconnaissance termite mound geochemical survey was undertaken in 2010 covering 218.5 km^2 on a grid of 400 m N-S × 80 m E-W. Assay results indicated a more anomalous area in the west, and an infill sampling programme was completed to a grid dimension of 200 m N-S × 40 m E-W.

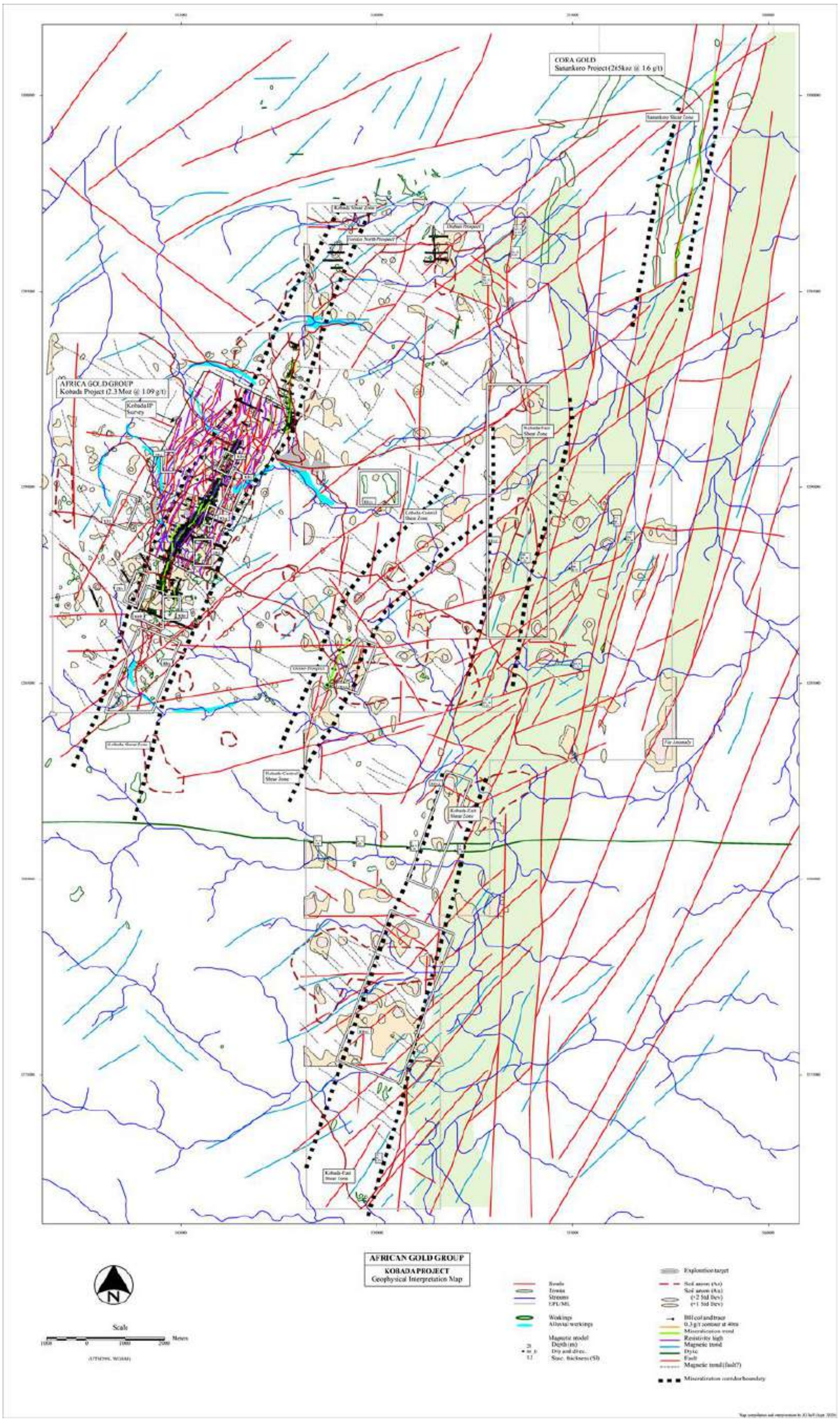
In 2011, a mobile metal ion (MMI) soil orientation survey was carried out over two lines (1200S in the Main Zone Northern Extension and 2300N in Foroko North). Sampling was conducted in 20 cm increments to a depth of 1 m. A total of 495 samples were collected and analysed for gold and 42 other elements. Ten additional mechanically dug and hand-dug trenches (4,049 m) were dug within the main zone and at Diaban, mainly for mapping purposes. A soil geochemical survey (1,377 sites) was conducted on the Diaban and Gosso targets.

In 2012, AGG conducted an additional MMI geochemical survey that covered 12 km^2 with 1,777 samples analysed for Au, As, Ba, Cu, K, Mg, Mn and Sn, and Fugro NPA Ltd of the United Kingdom processed the satellite stereographic imagery. The satellite imagery analysis provided a sub-meter topographic elevation map (digital elevation model). The survey was ground-controlled with a high-precision differential Global Positioning System (GPS).

In January 2020, a high-definition stereo satellite survey was conducted over the main orebody by PhotoSat Information Limited using their geophysical satellite processing system, to assist with the updated topography for the geological modelling and to improve the accuracy of the depletions caused by the artisanal miners, who have been targeting the laterites and some of the saprolite material. The survey accuracy is within 6 cm RMSE (root mean square error) and LE90 (90 % linear error) 11 cm.

In June 2020, the historical geophysical data was reinterpreted by John Bell to assist in identifying the exploration target areas. The exploration target areas were, however, similar to the original target areas.

These exploration target areas are shown in Figure 9.2.



Source: JG Bell (2020)

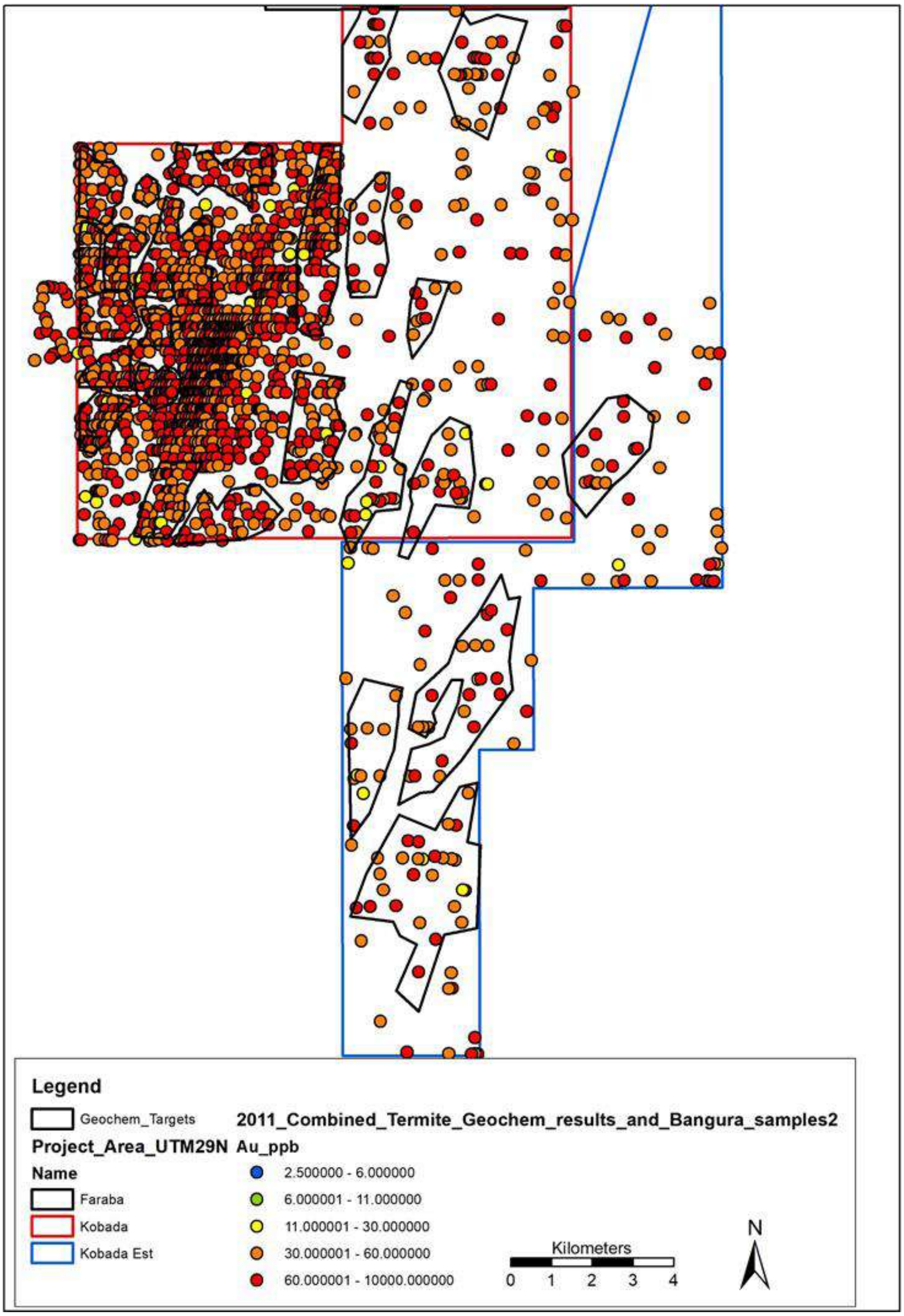
Figure 9.2: New Geophysical Exploration Target Interpretation

9.2 SAMPLING METHODS AND SAMPLE QUALITY

A total of 6,685 samples were collected for the geochemical survey, and an additional 418 soil samples were collected in lieu of termite sampling (see Figure 9.3). If *orpaillage*, quartz veins or other anomalous features were encountered during this programme, a grab sample was taken of the anomalous material; 240 grab samples were thus collected. The samples were analysed for gold at the ALS Laboratory in Bamako by 50 g fire assay. The sample quality was tracked with blanks or certified reference material every 20th sample.

A total of 4,323 trench samples were collected at 1 m intervals along the trench.

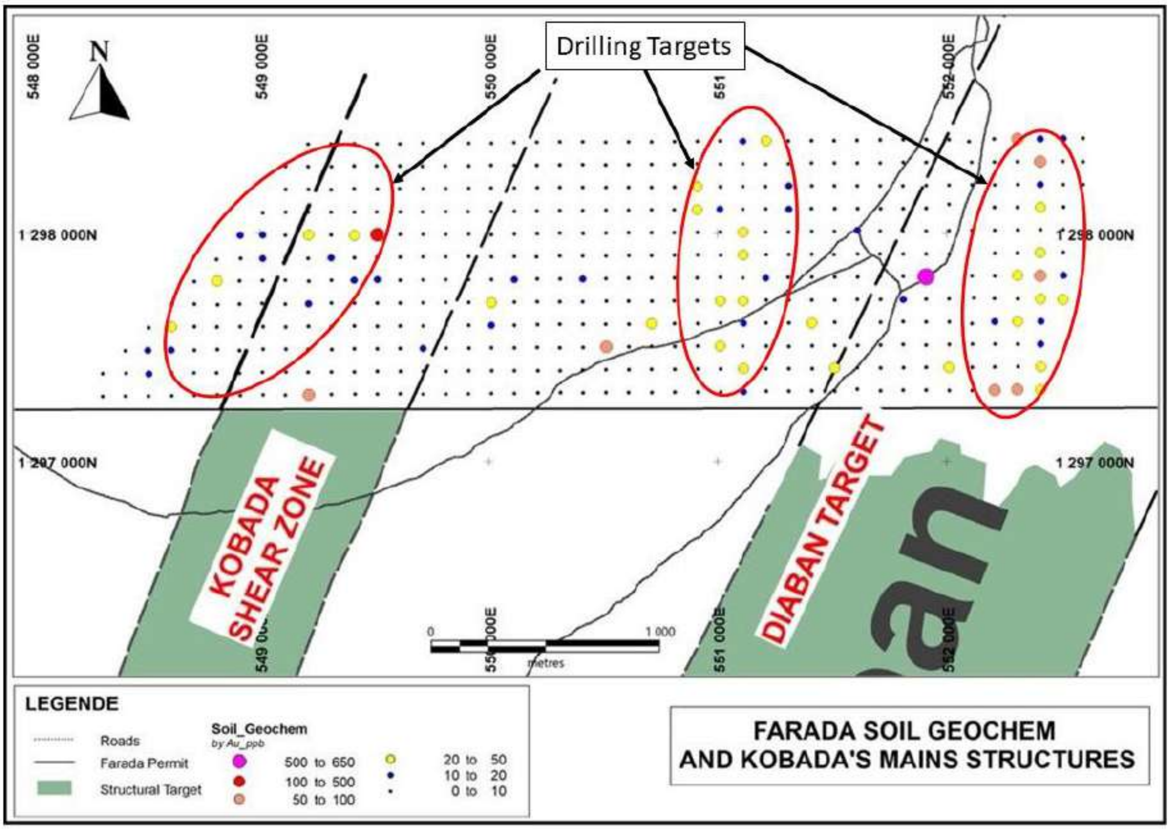
None of the soil samples or trench samples were used in the current Mineral Resource estimate as they only give an indication of the mineralised zones.



Source: AGG

Figure 9.3: Termite Geochemical Au Sampling

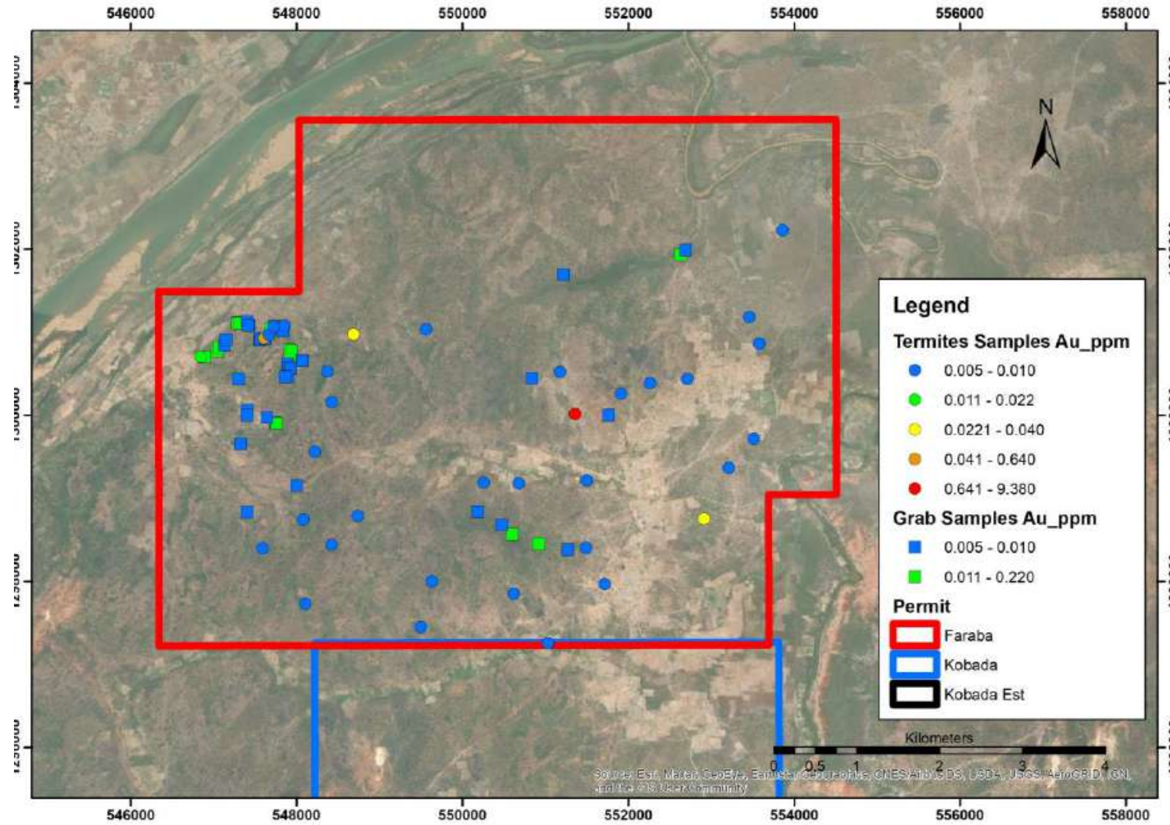
In January 2021, AGG commenced with a soil sampling programme at the Faraba permit area, and it was completed in February 2021. A total of 459 soil samples were collected on a grid spacing of 100 m × 100 m to test for additional drilling targets (see Figure 9.4). The samples were analysed for gold by fire assay at the SGS Laboratory in Bamako. As part of the quality assurance and quality control (QAQC) protocols, blanks, duplicates, and certified reference material (CRM) were inserted in the sampling to monitor the accuracy and precision at the laboratory. The batch commenced and ended with a blank sample, and every 10th sample was either a blank, duplicate or CRM.



Source: AGG

Figure 9.4: Faraba Soil Sampling Programme

An additional sampling programme was undertaken at the Faraba permit area as part of the requirements for the retention of the permit area. A total of 87 samples were collected, of which 52 samples were grab samples and 35 samples were termite mound samples (see Figure 9.5). The samples were again analysed for gold at the SGS Laboratory in Bamako with similar QAQC protocols.



Source: AGG

Figure 9.5: Grab and Termite Mound Samples at Faraba Permit Area

Also in 2021, 7 termite mound samples and 19 grab samples were collected at Kobada Est to test for additional drilling targets associated with the geophysical targets. The grab samples were collected in artisanal pits: 11 of the 19 grab samples were collected in Artisanal Pit 1 with samples averaging 3.00 g/t. Figure 9.6 shows the termite mound and grab samples at the Kobada Est permit area.

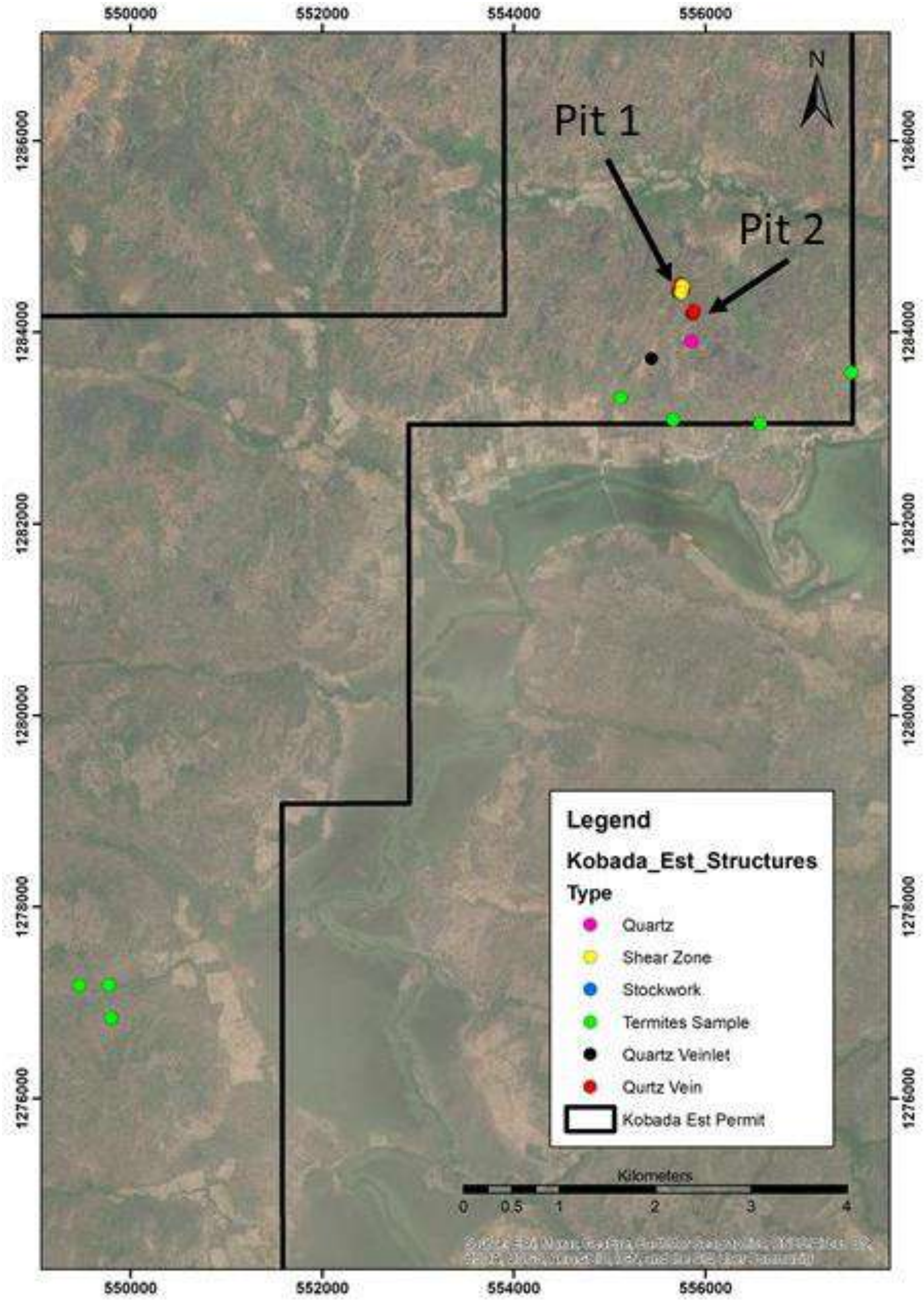


Figure 9.6: Kobada Est Grab and Termite Mound Sample Location

Table 9.1 shows significant grab samples greater than 0.3 g/t at Kobada Est in Pit 1.

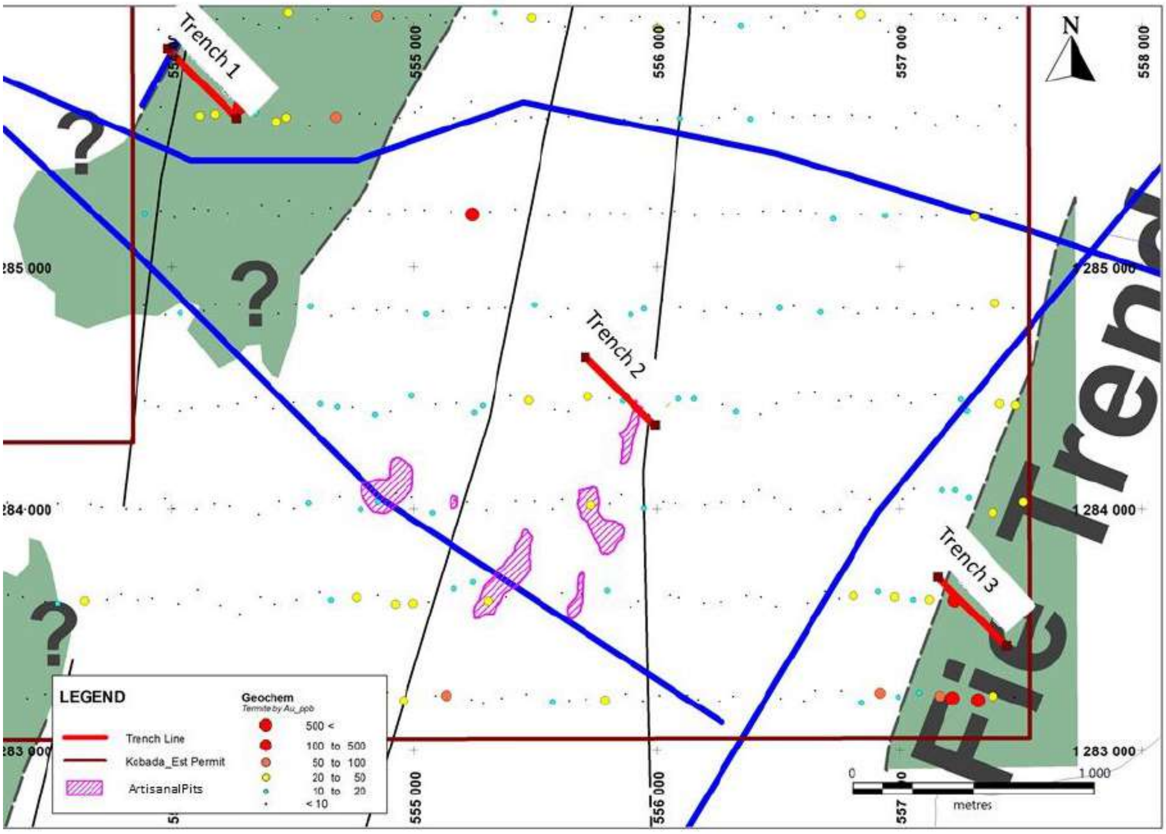
Table 9.1: Significant Grab Samples in Pit 1

Structure	Strike	Dip Angle	Dip Direction	Au
	o	o	o	g/t
Quartz Vein	110	75	200	0.38
Quartz Vein	110	80	200	29.70
Shear Zone	30	70	120	1.70
Shear Zone	20	60	110	0.36

Figure 9.7 shows a quartz vein at Kobada Est Pit 1. The vein is approximately 15 cm wide and strongly brecciated with iron oxide staining. The vein strike is 110° and dips approximately 75° to the southwest.

Figure 9.7 Quartz Vein

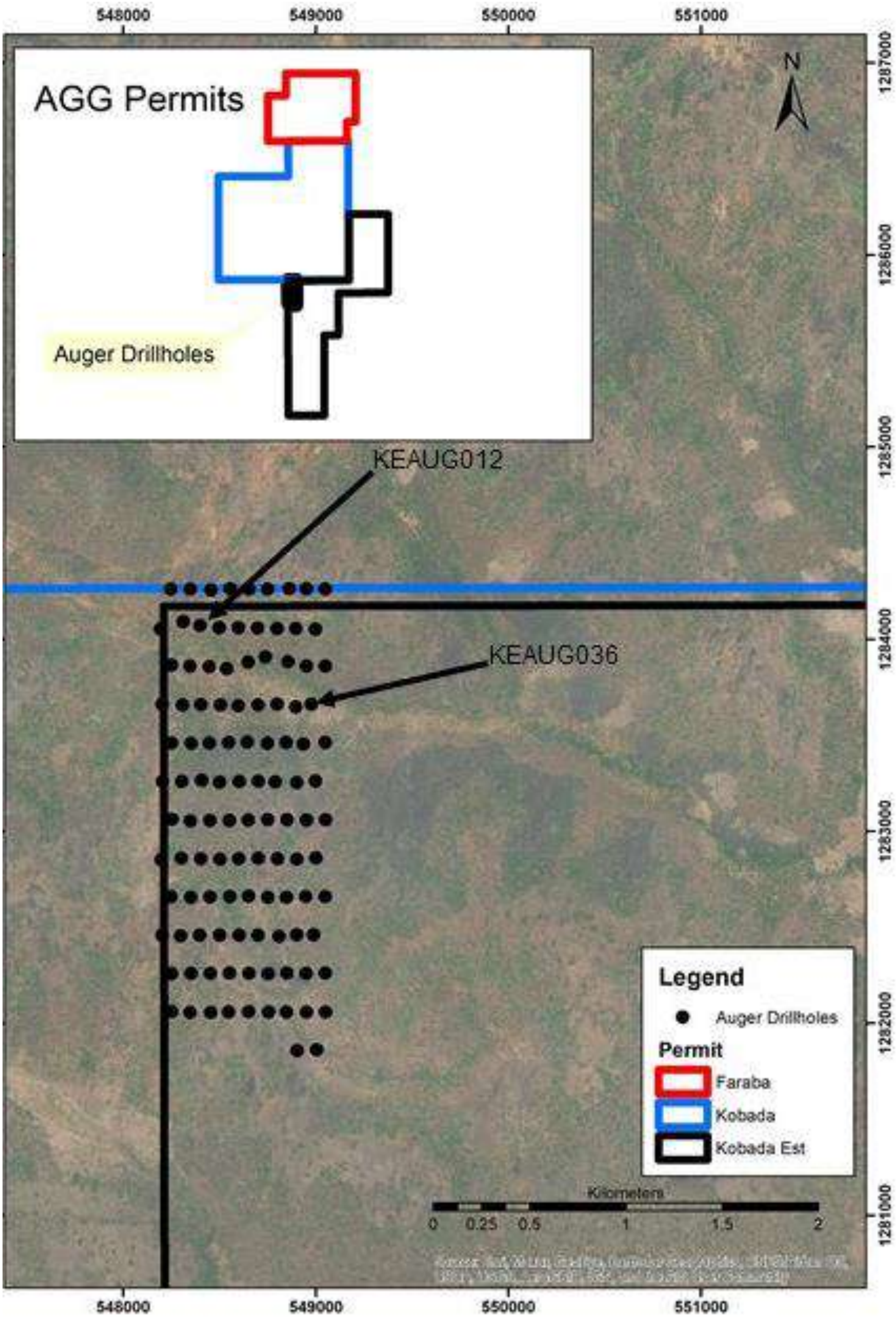
Trenching at Kobada Est is currently in progress. Trenches are dug to an approximate depth of 3.5 m and sampled at 1 m intervals. A total of three trenches are planned to be completed at Kobada Est. Trench 2 is currently being sampled, and the first batch of samples (84 samples including QAQC samples) have already been dispatched to the laboratory. Figure 9.8 shows the Kobada Est trenching programme.



Source: AGG

Figure 9.8: Kobada Est Trenching Programme

In June 2020, an auger drilling programme was undertaken at Kobada Est, and a total of 110 auger drillholes totalling 1,546 m were drilled. Drillholes were planned at a grid spacing of 200 m × 100 m (see Figure 9.9).



Source: AGG

Figure 9.9: Auger Drilling Programme

The aim of the auger drilling programme was to test for additional diamond drilling targets. Table 9.2 presents the auger drillhole intercepts greater than 0.40 g/t.

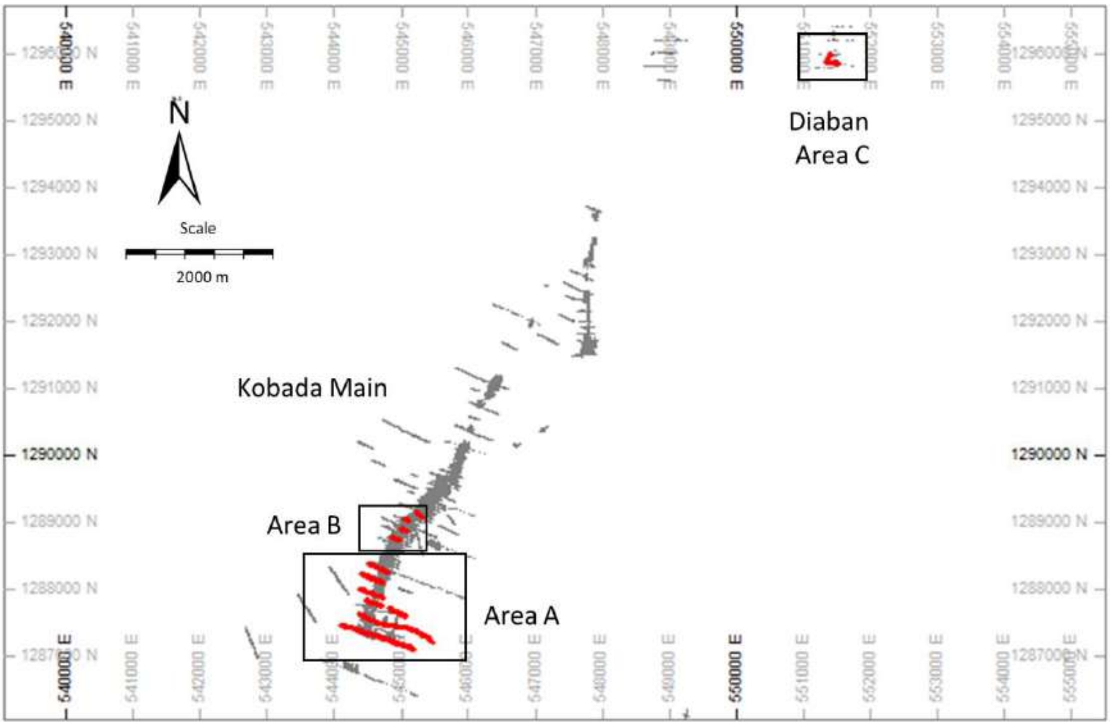
Table 9.2: Significant Drillhole Intercepts Greater than 0.4 g/t

Drillhole Name	From	To	Width	Au
	m	m	m	g/t
KEAUG036	15.00	17.00	2.00	3.66
KEAUG012	7.00	9.00	2.00	0.87

9.3 SAMPLE DATA

The termite mound geochemical survey was conducted at the termite mounds, and failing the occurrence of any termite activity within the 80 m × 40 m area around the idealised location, a soil sample was taken at that location. The results of this sampling are shown in Figure 9.3.

The 4,323 trench samples from the 8 trenches in 2010 were taken at 1 m spacing, depicted as Area A in Figure 9.10. The 10 additional trenches excavated in 2011 were used mainly for mapping purposes and are in Areas B and C.



Source: AGG

Figure 9.10: Trench Location in Kobada Main and Diaban Target Areas

9.4 RESULTS AND INTERPRETATION OF EXPLORATION INFORMATION

The 2010 geophysical interpretation identified future targets. Figure 4.1 shows the interpreted targets of the Kobada Main Shear Zone, Foroko Target, Foroko North Target, Diaban Target, Gosso Target and Siramana Target. The Kobada Main Shear Zone was targeted as the most promising and has been the focus of the exploration drilling over the years.

The results of the exploration culminated in a Mineral Resource model and estimation of the main Kobada orebody. The model shown in Figure 9.11 was refined and estimated using the drillhole information collected over the different exploration periods.

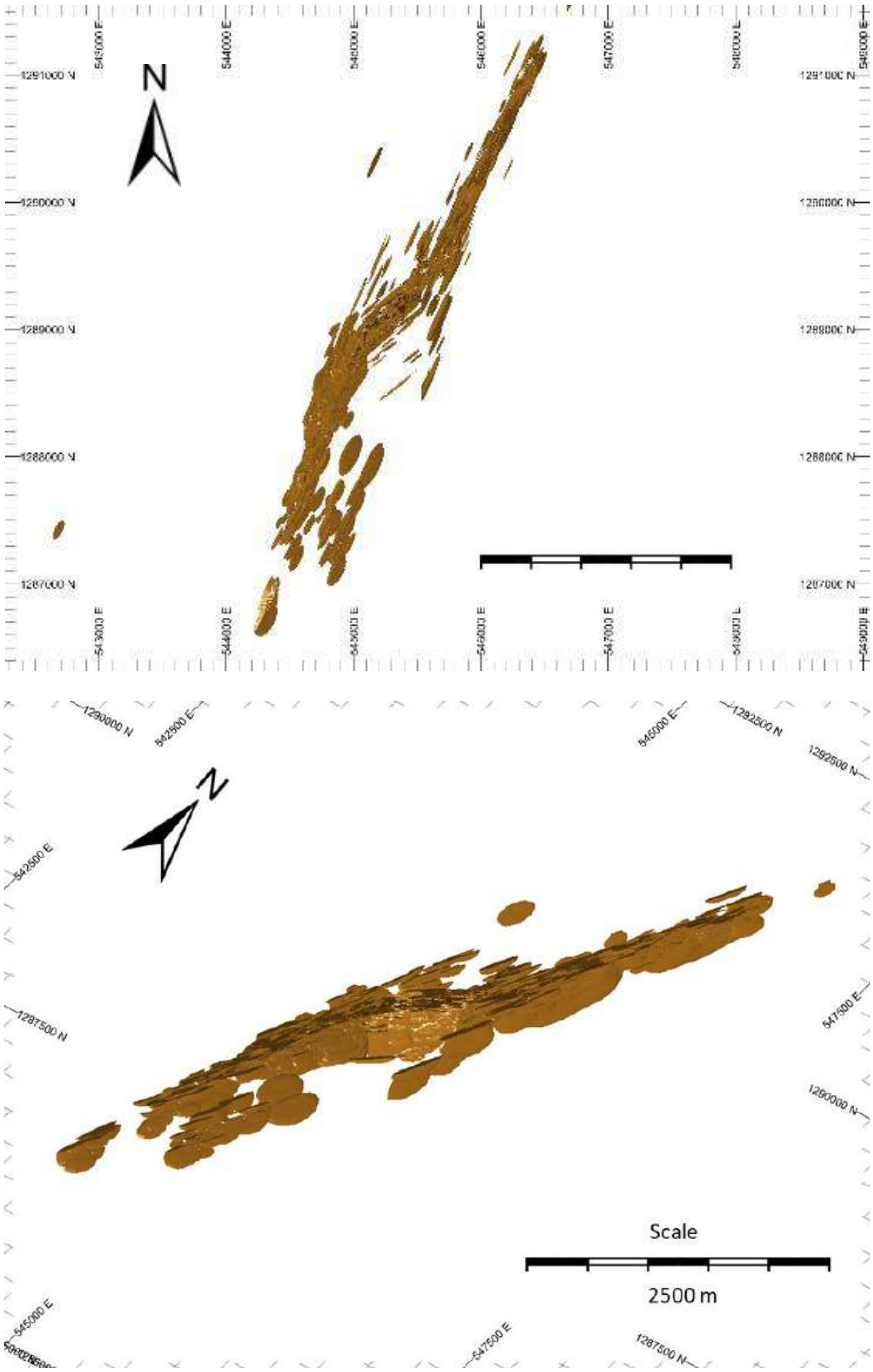


Figure 9.11: Geological Model Based on Exploration and Drilling

10 DRILLING

10.1 TYPE AND EXTENT OF DRILLING

The Kobada Project has been explored by five companies since 1988. The company, year, and corresponding number of drillholes are shown in Table 10.1.

Table 10.1: Drilling Statistics by Company and Drilling Method

Company	Number of Drillholes	Total Metres Drilled	Drilling Method		
			Diamond Core	Reverse Circulation	Air Core
BRGM (1988)	7	913	913		
La Source (1996)	50	4,804		4,804	
Cominor (2002 and 2004)	132	10,111		8,375	1,736
IAMGold (2009)	2	200	200		
AGG (2005–2012)	1,289	114,357	26,890	81,982	5,485
AGG (2015)	13	1,398	1,398		
AGG (2018)	10	1,173	482	691	
AGG (2019)	67	14,671	11,417	501	
AGG (2020)	39	5,842	1,254	4,588	
Total	**1,609**	**153,469**	**42,554**	**100,250**	**7,221**

The drillholes and samples per campaign are shown in Figure 10.1.

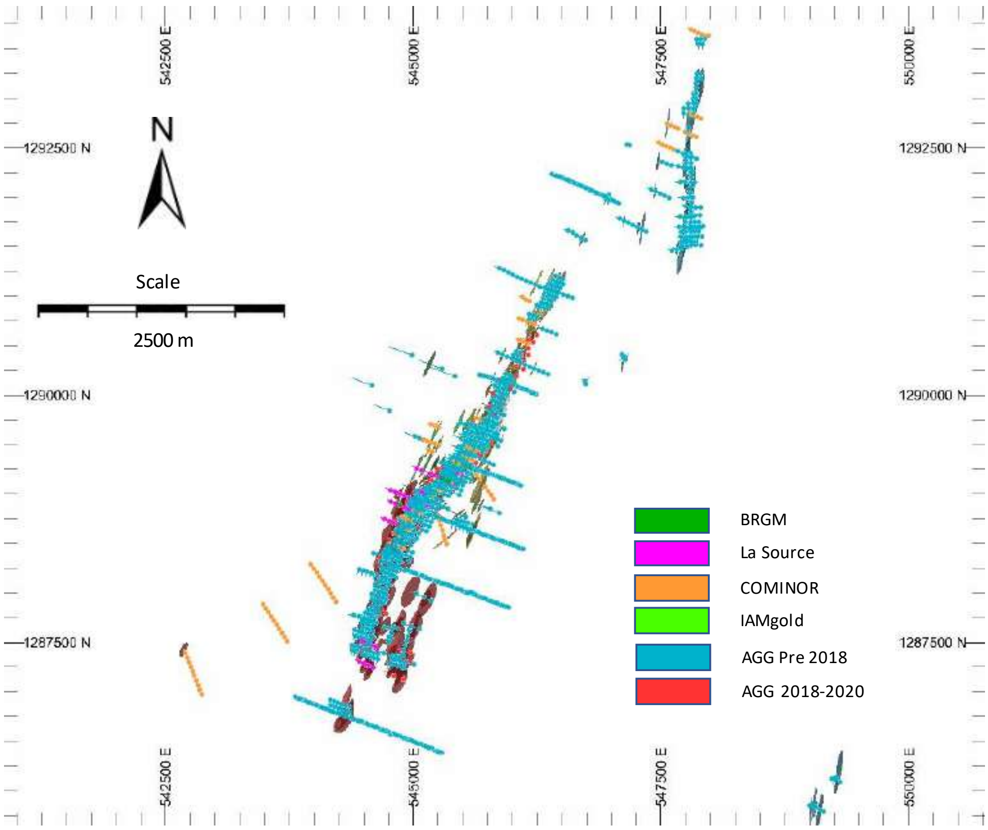


Figure 10.1: Drillhole Campaigns for Kobada

Drilling methods and sampling techniques were reported in detail in Watts, Griffis and McOuat's report of May 2008 (Wolfe et al, 2016). As almost 90 % of the drilling has been completed by AGG, the detail of the drilling completed prior to their involvement has been omitted and a brief summary presented below.

In 1988, BRGM tested a resistivity anomaly outlined during the 1987 geophysical surveys with all the diamond drillholes intersecting the mineralisation. Drilling was orientated towards 290° or 300°, with a westerly dip of 55°. In 1996, La Source completed an RC drilling campaign designed with the objective of extending the mineralisation intersected by the previous diamond drilling. RC drilling was orientated at ~290° and drilled with a westerly dip of 55° except for three drillholes, which were drilled towards the east. In 2002, Cominor conducted both RC and air core drilling for reconnaissance purposes. Air core drilling totalled 25 drillholes, but these were not utilised in the Mineral Resource estimation. RC drilling was orientated at ~290° and drilled with a westerly dip of 55°. The programme continued in 2004 with the majority of the drillholes at the same orientation and a small number oriented at ~345°. In 2009, IAMGold Corp. drilled 10 RC drillholes (1,136 m) and two DD drillholes (200 m) halfway between the previous drillholes in the main Kobada orebody to test the effects of a larger sample support and analytical aliquot, and collect a representative bulk sample for metallurgical testing at SGS Lakefield, Canada. The RC drillholes were not included in the drillhole database supplied to Minxcon for the 2020 Mineral Resource estimation.

From 2005 to 2015, AGG actively drilled the Kobada Main Shear Zone to establish the extents of the mineralisation and to model the orebody. The infill drilling campaign of 2019 was split into Phase 1 and Phase 2. Phase 1 was primarily focused on improving the understanding of and confidence in the geological model, as well as upgrading the Mineral Resource categories. Phase 2 included the Phase 1 aims but was also intended to increase the potential Mineral Resource. The 2019 drilling campaign was also used as confirmation of the previous data. The 2020 drilling campaign continued with the infill drilling process with the aim of upgrading the inferred Mineral Resource.

For the 2005 to 2015 drilling campaigns, the proposed RC and DD drill sites were located using a Garmin handheld GPS with a ± 3 m accuracy. A compass was then used to measure the azimuth of the drillholes. After drilling, the actual locations of all the drillholes were surveyed with a differential GPS (Novatel Flex Pack 6 DGPS using the Omni STAR XP-G2 service) with a ± 20 cm accuracy. The 2015 drillholes were located and surveyed with a Garmin handheld GPS with a ± 5 m accuracy. The drillholes were surveyed with a Flexit tool that measures the azimuth, magnetic field, inclination, and temperature at each depth selected. These measurements were taken at three planned depths: 6 m, 65 m, and 130 m for DD. For RC drillholes, surveys were taken at 6 m and at the end-of-hole (maximum depth).

Pre-2018, a total of 786 RC drillholes for 55,233 m were completed by AGG at Kobada. Drilling campaigns were undertaken in 2007 (110 drillholes), 2009 (22 drillholes), 2010 (163 drillholes), 2011 (258 drillholes), 2012 (228 drillholes), 2018 (5 drillholes), and 2020 (31 drillholes). Drillhole orientations were generally an approximately 290° azimuth with an approximately 55° westerly dip, which in general is orthogonal to the orientation of the shear zone mineralising structure and is optimal for defining margins in the mineralised system. In 2010, as a result of a geological review, the decision was made to re-orient the drilling with a 200° azimuth and a southerly dip. The rationale was to intersect the E-W striking veins, which were postulated to carry most of the gold at Kobada, with a northerly shallow dip. Regardless of the orientation of these veins, this drilling orientation makes the definition of a hanging wall and a footwall to the mineralisation (mineralised envelope) difficult at best in the areas where only N-S orientated drilling exists.

A total of 220 DD drillholes for 45,696 m have been completed by AGG at Kobada between 2005 and 2020. Drilling campaigns were undertaken in 2005 (6 drillholes), 2006 (13 drillholes), 2007 (86 drillholes), 2009 (2 drillholes), 2010 (6 Drillholes), 2012 (10 drillholes), 2015 (13 drillholes), 2018 (5 drillholes), 2019 (67 drillholes), and 2020 (12 drillholes). The drillholes pre-2018 have generally been collared with an HQ size, and this is continued to the base of the oxide material. The drillhole size is then reduced to NQ, and drilling is continued to the planned depth. A Reflex EZ-Shot instrument was used to collect the downhole survey data, with an EZY-mark tool used to orientate the core from mid-2006 onwards.

Several drilling contractors have been used.

The 2018 and 2019 campaigns were intended to improve the confidence in and refine the geological and Mineral Resource models. The 2018 drilling campaign was completed by Geodrill while the 2019 drilling campaign employed AMCO. Minxcon was employed to oversee and manage the drilling campaigns in 2018, 2019 and 2020. During the 2018 drilling campaign, both RC and DD drillholes were drilled. In 2019, only DD drillholes were utilised while in 2020, predominantly RC drillholes were utilised with twelve diamond drillholes. Some of the deeper RC drillholes were completed with diamond tails to drill the sulphides. The drill direction was both in a westerly direction to test NNE-trending shears and in a southerly direction to test the roughly E-W striking veins, at a general dip of 55°. This assisted in collecting more geological information in areas that were previously predominantly tested by RC drilling. This contributed to the refinement of the geological interpretation and model as described in 8.2.

During these campaigns, the drillholes were collared with HQ in the laterite and drilled as such until the transition/sulphide zone where the core was changed to NQ until the end of the drillhole. A Reflex EZ-trac downhole survey tool was used to take downhole survey measurements.

Below is an extract from the Minxcon Exploration procedures (Procedure No:020432) followed for the RC drilling during the 2018–2020 drilling campaigns.

Drill Site Establishment

The following tasks must be done prior to drilling commencing:

- *An adequately sized area around the drill rig must be demarcated by a suitable danger marking material.*
- *All necessary measures must be taken to prevent ground contamination (by oils, grease, diesel, etc.).*
- *The designated work area is the full responsibility of the drilling contractor. Implementation of all necessary safety procedures within the designated work area, including but not limited to signage, use of PPE, guarding of all dangerous, moving parts on drill machine/compressors, etc. is the drilling contractor's responsibility.*

- *A permanent site foreman/or Safety Representative with delegated responsibilities for MH & S, a dedicated vehicle, detailed knowledge of the safety file contents, appropriate first aid kit and trained personnel, as well as an explicit knowledge of first aid procedures in the event of an accident must be supplied by the drilling contractor.*

Setting Up of a Drilling Rig

- *Setting up of the drilling rig is the full responsibility of a drilling contractor and a geologist.*
- *The geologist will mark (with a peg) the position of the drillhole to be drilled with the aid of a hand-held GPS. The GPS will be checked by the project geologist to ensure the coordinates are reported in the correct system (Form 030432F0). For the Kobada project the coordinates are based on the WGS84 Datum and are in UTM format.*
- *The majority of drillholes are to be inclined at -55° and the drillhole azimuth is 294,20 True North.*
- *The drilling contractor will then align the drilling rig as per instructions from a geologist in terms of the dip and direction of the drillhole to be drilled.*
- *Before drilling commences, the drilling contractor and geologist must sign the RC Drilling Site Set-Up Form (Form 030432F0).*

Checks during Drilling

The following checks have to be performed during drilling:

- *Access roads must be kept to a minimum and kept as narrow as possible e.g. if two tracks have been made vehicles must stay within these tracks.*
- *The natural vegetation must be preserved within acceptable limits.*
- *If any trees are to be removed prior permission must be obtained from the landowner/relevant authorities wherever possible.*
- *Adequate measures to prevent or contain oil spills must be in place (e.g. PVC sheets underneath the machines). If oil spills occur an applicable oil clean up kit should be applied and the soil, sand, etc. re-claimed.*
- *There should be waste disposal bins available into which all litter etc. is placed.*
- *An acceptable waste removal system must be in place i.e. the driller must dispose of all waste into a recognised waste disposal facility.*
- *Chemical and other hazardous waste needs to be disposed of according to acceptable standards and in the correct disposal sites.*
- *An adequate latrine system, which is serviced according to acceptable standards, must be available.*
- *Natural water courses must be protected against contamination/pollution.*

Drilling Operations

General

- *Drill sites are to be indicated by the project geologist/the client's elected personnel (as is the case at the AFRICAN GOLD GROUP RC drilling project in Mali) to the drill foreman, at least 24 hours in advance.*

- *The size of the drill site should be large enough to house the drill rig, compressor and geologist's sampling area.*
- *The position of the drillholes is marked with the aid of a GPS and a peg is placed at that point. Any change to the planned coordinates needs to be updated in the database.*
- *A daily target expected per drill machine is to be discussed and agreed between the exploration manager/project geologist and the drilling contractor. In the case of the AFRICAN GOLD GROUP project, a daily target of at least 30 m has been set.*
- *The drill contractor is responsible for the full implementation and adherence to all relevant chapters of the Mine Health and Safety Act (or applicable legislation) for all operations with regards to the drilling activity, as well as compliance to the approved Environmental Management Plan.*
- *The appointed drill contractor is responsible for the successful drilling of all drillholes at the given sites.*
- *Drillholes may only be stopped by explicit instruction of the geologist in charge in liaison with the client or Minxcon exploration director. In the event of very poor ground conditions, drilling may be temporarily suspended until all relevant parties have met, discussed the implications and concluded an action plan for the drillhole.*
- *The procedure for starting a drillhole varies based on the expected end of hole ("EOH") depth and the ground conditions. In general, a drillhole is initially cased to prevent caving until stable ground conditions are encountered.*
- *The drilling supervisor needs to report back daily to the geologist on site to inform him on the day's drilling performance.*
- *Before RC-drilling can commence the main air hose connection between drill and compressor must be inspected by the drill foreman to ensure that all safety chains and equipment are in place.*

<u>Sampling Procedure</u>

The strategy to be adopted must be carefully designed so that at each stage of the process, the chance of taking biased, unrepresentative or contaminated samples is minimised. In order to achieve this:

- *All sampling activities should be positioned away from the cyclone "smoke stack" so as to avoid dust pollution.*
- *Samples are to be collected in 1 m intervals as per the instructions from the project geologist.*
- *New, clean sample bags are to be used at all times and for each sample.*
- *The geologist or site supervisor will ensure that the necessary sample bags are correctly labelled and available before the drillhole is drilled.*
- *All samples are to be collected from the cyclone outlet and as per indication from the drill operator.*
- *The procedure is to fasten the sample bag tightly to the (bottom) of the cyclone unit and keep it there for the duration of the sample drilled, catching the sample.*
- *The second sample assistant will take the full bag from the sampler and hand him the next marked sample bag.*

- *The sample assistant must always communicate with the drill operator to ensure the sample collection is done properly.*
- *The sampler under the guidance of the geologist must ensure that the cyclone is cleaned with compressed air after the sample is bagged and prior to the collection of the next sample.*
- *At the drill site, the sampler will enter all the relevant data into his triplicate book and liaise with the drilling contractor to ensure that the correct information is used on their record sheets.*
- *If any underground water is encountered during the drilling, its depth of occurrence must be recorded. If a wet sample has been obtained during drilling, a note must be recorded on the comments section of the log sheet or in the description.*
- *Each sample bag is weighed and the drillhole number, start date, end date, sample number, sample batch number, drilling system used, encountered voids and sample depth are recorded in the provided RC drilling log sheet.*
- *All samples will have to be stored in a shaded, dry storage facility or shed pending dispatch to SGS laboratory in Bamako.*

Transport to the Core Yard/Storage Facility

- *The transporting of the samples safely to the sample storage area is the responsibility of geological staff as agreed before the drill programme starts. During transport, whoever is responsible must ensure that the samples are transported in a manner that will not cause loss or cross-contamination.*
- *Loading of the samples onto the vehicle must be done with the utmost care and they must be delivered directly to the core yard, without any unnecessary travel or deviations.*
- *The security and integrity of the samples must be maintained at all times. Samples will be loaded onto the geologist's bakkie as soon as they have been ticketed, checked and re-checked against the sample information on the geologist's log sheet.*

Sample Measurement – Inspection of Drillers Marking

- *All the labelled and weighed samples are sorted and ordered at the core yard.*
- *Sample bags are put in sequence to ensure that all samples are present and the "from" and "to" correspond with the sample ticket books.*
- *The samples will be packed per drillhole in order of drilling e.g. (0 m to 10 m or project specific).*

Sample Splitting

- *The samples are split using a riffle splitter into ½, ¼, ⅛ sample units or as per instruction from the project geologist.*
- *The split should be maintained across all drillholes. Always maintain an even feed when pouring the sample into the splitter.*
- *If the splitter does not produce the desired fraction in a single pass, repeat the process until the desired minimum mass of 2 kg (as stipulated by the project geologist) is achieved.*

- *The fraction that is not destined for the laboratory should be kept in a single marked bag (or "Archive Sample" bag) for each sample for reference and further testing if required.*
- *If this is not required (instructions issued by the project geologist) the split (that is not going to be submitted for assay) can be thrown back into the hole once the hole is completed. However, this must be discussed with the drillers and geologist before drilling starts. For the Kobada project, the remaining sample for each drilled and sampled metre is kept on a 50 kg grain bag.*
- *The riffle splitter is to be cleaned with a clean brush/dry cloth after each sample has been divided so as to exclude all forms of possible contamination. Ideally an air hose should be used to clean the riffle.*
- *After splitting, a portion of each sample is laid on the ground in the order of the drilling depths. The geologist or the person to carry out the logging must take note of the colours of the dry powder of the chips. This usually gives an indication of the points where changes in rock strata or rock type occur.*

RC Chip Logging

- *The logging of rock chips from RC at African Gold Group's Kobada project will be done at the company's core shed.*
- *Strict supervision of the handling of the chips is required. The geologist or the geological technician must be able to train personnel so that the purpose and importance of the whole exercise is clearly understood by all those that are involved.*
- *After splitting the sample, scoop a portion of the chips (approximately 500 ml container).*
- *The scooped material must be cleaned thoroughly and gently in a bucket of water while being kept in the sieve. Care should be taken not to spill the chips into the bucket. Once the material is cleaned, it is placed in a chips tray where the depths are written on the side of the tray and a sample ticket is placed in the tray too.*
- *The sample chip trays must be clearly marked with the drillhole number and the appropriate depths per sample. The supervisor will ensure that the material placed in the sample chip trays is a good representation of each sample taken.*
- *The supervisor or geologist must check to ensure that every bag is properly labelled with the project number, drillhole number, depth, and batch number. He must also ensure that every drillhole has a representative sample and that all the samples that will go to the laboratory have been weighed and securely fastened.*
- *A hand lens or magnifying glass must be used to aid in the logging process. If available, hydrochloric acid must also be used to test the chips for some elements. A record of the following should be made on the log sheet:*
    - *Depth (from-to)*
    - *Colour of powder*
    - *Colour of chips when cleaned*
    - *Grain size*
    - *Alteration types*
    - *Minerals e.g. pyrite, chalcopyrite, chalcocite, bornite, etc.*
    - *Description*

- *Geological Structures, etc.*

- *Logging is done onto the standardized data capture percussion log sheet Form 030432F2 for ease of computer data capture. This data is then captured into an excel format database to which is added all relevant geological and survey data, for later use in geological modelling.*

Below is an extract from the Minxcon Exploration procedures (Procedure No:020431) followed for the diamond drilling during the 2018 and 2019 drilling campaigns. This extract is for the core handling procedures and not so much for the drill rig and site establishment (which are similar to the above procedure).

<u>Drilling Procedure</u>

Presentation of Core

- *The drilling contractor must present the core in prescribed core trays in an acceptable manner. All core trays must be clearly labelled and numbered to the satisfaction of the Minxcon Exploration site staff. Paper stickers (if used) must reflect drillhole number, box number and core lengths contained otherwise this information is written directly onto the core boxes.*
- *Driller's core blocks must clearly show drilled depths/run length, core recovery and core loss/gain. All writing on plastic depth blocks and trays must be done using an indelible black marker pen.*
- *The core must be cleaned of all drilling related impurities, greases, oils and any other foreign matter prior to being presented in the core trays. It is to be laid out in a systematic manner with due regard to the direction of the advance of core beyond the previous segment of core. However, for the Kobada drill core extra care must be taken in order to avoid washing off some of the free fine gold that can be encountered in the poorly consolidated sediments.*
- *Particular care should be taken with regard to specific "reef" targets as specified by the geologist.*
- *The contractor must ensure that the core is laid out and aligned in such a manner as to join all "breaks" in the core. This is to permit the accurate measurement of the length of run and the accurate geological record of the drillhole to be maintained and recorded. Once the core is laid out, the contractor must take all precautions necessary to ensure that the core is not disturbed until inspected or approved by the geologist.*
- *The contractor must mark the core at intervals or according to the geologist's instructions. In addition, the contractor must mark on the core, each "end of run" with a paint marker. With regards to the Kobada project, this is impossible due to the unconsolidated nature or the sediments as well as the highly weathered very clayey material which cannot be easily written on.*
- *The drill core at Kobada is required to be oriented (in the sulphides and maybe transition zones).*
- *Where a core orientation mark has been placed the driller marks "OR" on the core on which the mark has been placed.*
- *The orientation device and all the necessary training and expenses associated with the device shall be on the contractor. At the beginning of each run the core must be*

orientated and a line of orientation must be recorded on the core itself with a permanent marker. Should the orientation line be unclear or invalid, question marks must be marked on the core.

Drilling Contractor's Responsibilities

- *The contractor must ensure that accurate interval measurement between runs is adhered to, and that at the end of every coring/sample run physical measurements of the stick-up, drill rod, and tally of rods in drillhole are maintained. This data is to be recorded in an acceptable and agreed format in the contractor's field logbook.*
- *The logbook is to have numbered pages, duplicated in sufficient number to allow the contractor and geologist to each receive individual copies of each page on completion of that page or drillhole as may be required (in addition a copy of each page may be retained by the respective machine operator if required). Such field logbooks are to include space where written comment and or instruction by the geologist, or requests by the contractor, can be noted.*
- *The contractor must ascertain from the geologist prior to commencing a drillhole as to the necessity of casing retention for that drillhole, and must adjust the manner and nature of the set-up accordingly.*
- *The contractor must ascertain from the geologist what core size is to be drilled. HQ (and HQ triple Tube) then NQ (where possible) size core is to be drilled at Kobada.*
- *No drilling deflections will be done in this phase of drilling at Kobada.*
- *When the drilling contractor pulls rods or changes the drilling bit, they should record the number of rods and the stick-up on a separate yellow drill block as an additional check to eliminate mistakes or to correct stick up errors.*

Core Handling

- *Handling of the core by the drill team should be monitored at each visit. Procedures that are considered to be detrimental to core quality must be reported to the exploration manager and the drill foreman, and must be corrected immediately.*
- *Detrimental procedures include general sloppy handling, inversion of core from core barrel to core tray, excessive banging of the inner tube to remove core, etc.*
- *All aspects of the drilling operation must comply with Health and Safety Act regulations. The contractor must hold regular safety meetings and a copy of the minutes of these meetings must be forwarded to Minxcon Exploration.*

Core Quality

- *The core must be inspected for dirt and grease, correct packing, and accuracy of core marking. End of run block markers must be correctly placed and should show the depth, the run length, and any loss or gain of core (see* Figure 10.2*). In addition, a block marker must be placed in the core where core loss / core gain has occurred. This same information must be recorded in a notebook kept at the drilling rig, and should be available for scrutiny by exploration personnel at any time.*

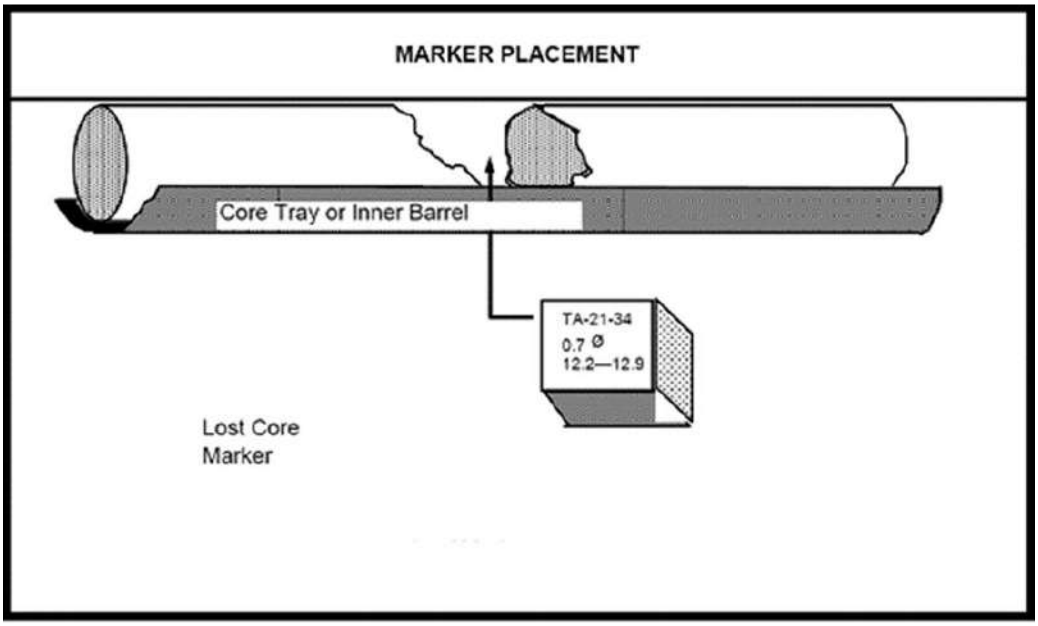


Figure 10.2: Core Loss Marker Placement

- *Core trays or boxes must be correctly marked and must be in good condition.*
- *Core must not be removed from the drill site until a preliminary inspection has been carried out to ensure that the general quality of the core marking is of the required standard, that the depths recorded on the blocks are correct and that they agree with the metre marks on the core. The core is provisionally accepted from the driller at this stage, with the understanding that, should minor discrepancies or errors be found when the core is examined at the core shed, the driller can be recalled to correct these errors.*
- *Core must not be left unattended at any stage.*

Transport to Core Yard

- *The transporting of the core trays safely to the core yard and logging area is the responsibility of either the drilling contractor or Minxcon Exploration staff (this would be determined prior to commencement of the job). During transport, whom ever is responsible must ensure that the samples are transported in a manner that must not damage, or cause loss or cross-contamination.*
- *Loading of core trays onto the transport vehicle must be done with the utmost care. All exploration and drilling personnel must handle core trays with both hands, and must avoid any tipping of the trays. An adequate number of personnel should be involved to ensure secure movement of boxes.*
- *Foam rubber spacer mats must be used between stacked core boxes, and the full load of core trays must be held securely together using two ratchet straps or rope.*
- *All trays must be delivered directly to the core yard.*
- *The security and integrity of the samples must be maintained at all times. The core collection register, Form 030431F2 (BHID, number of core trays, length and condition of core) must be filled in and signed by the driver and the driller before the sample/core is removed from the drill site.*

- *The care and custody of the accepted core samples is transferred from the drilling contractor to Minxcon Exploration once Minxcon Exploration personnel sign for the core.*

Core Shed Procedures

The following information should be known for each and every drillhole:

- *Project name*
- *Drillhole identification number (BHID)*
- *Collar X, Y and Z coordinates*
- *Date of drilling and logging*
- *Drilling contractor and driller*
- *Geologist*
- *Final depth*

Immediately when the core arrives in the core yard, the geologist or core yard manger must sign the Core Shed Register (Core) – Form 30431F2

Core Measurement – Scrutiny of Drillers Marking

- *Once core has been delivered to the core shed, the core collection register (number of core trays, length and condition of core, and "from" and "to") is signed by the geologist or core yard supervisor and the driver.*
- *The core trays are laid out in sequence on a waist high logging bench, and the accuracy of the driller's core marking is checked. Where necessary, core that has shifted or moved in the core tray during transport is re-assembled and fitted together to eliminate any gaps before measurements are scrutinised.*

General Logging Procedure

- *On a daily basis, the logging procedure begins with a review of that portion of the drillhole that may have been previously logged. This is followed by a quick review of the new core, to acquire an overall feel for the stratigraphy, and to identify significant lithological and mineralogical changes.*
- *The logging is guided by a set procedure based on identifying the 'Main' lithological units. Geological Information is captured as follows:*
    - *Lithologies and respective "from" and "to" depths. These depths must be contiguous.*
    - *Stratigraphic units and respective "from" and "to" depths*
    - *Comments*
    - *Composition*
    - *Grain size and packing*
    - *Colour*
    - *Texture*
    - *Alteration*
    - *Structure*
    - *RQD (Rock Quality Designation)*

- *Mineralisation*
- *Contact type*
- *Angle to core axis*
- *Orientation: Alpha and Beta*

Logging is done onto a standardised data capture sheet or diamond drilling log sheet, Form 030431F4 for ease of computer data capture. For each data field there are a number of set options that are available to the geologist logging the core. Coding options are project specific so must be designed for each project. This data is then captured onto an Excel spreadsheet.

10.2 FACTORS INFLUENCING THE ACCURACY OF RESULTS

Minxcon is of the opinion that there are no factors materially detracting from the accuracy and reliability of the results. Drilling and sampling have been conducted according to industry best practices. The core recoveries were measured during the 2019 drilling campaigns and have a recovery of 75 % for the laterite, 83 % for the saprolite and 96 % for the transition and sulphide zones. The recovery for the saprolite is lower because of the friable nature of the highly weathered zones. Minxcon also investigated the percentage of samples above 0.3 g/t (which informed the estimation) that had a significant core loss. The samples that had a significant core loss was 5 % (see Table 10.2). These would have been distributed amongst the previous samples (approximately 41,273 samples), and represent 0.1 % of the total; therefore, they would not have had a material impact on the Mineral Resource estimation.

Table 10.2: Core Loss Summary

Phase	Parameter	Total Samples	Samples with No Loss	Samples with Core Loss	Samples with Significant Core Loss
1	No. of samples	272	230	42	17
	Grade (g/t)	1.42	1.47	1.15	0.86
	%	100	85	15	6
2	No. of samples	349	255	94	13
	Grade (g/t)	1.74	1.68	1.91	1.38
	%	100	73	27	4
Total	No. of samples	621	485	136	30
	Grade (g/t)	1.60	1.58	1.68	1.09
	%	100	78	22	5

The RC recoveries for the 2020 drilling campaign were calculated utilising the estimated weight vs actual weight for each 1 m sample and have a recovery of 61 % for the laterite, 87 % for the saprolite, 100 % for the transition, and 92 % for the sulphide zones. The average estimated recovery for the RC drilling was 85 %.

10.3 EXPLORATION PROPERTIES – DRILLHOLE DETAILS

This section is not applicable to the Kobada Gold Project as it is an advanced exploration project with sufficient drillhole data to declare a Measured and Indicated Mineral Resource with a feasibility study.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 SAMPLE HANDLING PRIOR TO DISPATCH

There is no sample handling detail prior to 2009.

From 2009 to 2015 (Wolfe et al, 2016), collection and processing of samples was performed by employees of AGG. For DD samples, transportation from the Kobada camp to Bamako ALS Laboratory was performed by ALS. All samples for the 2015 DD programme were delivered by AGG employees to the SGS Bamako Laboratory. A record for each sample was generated, including type of sample, drillhole location, description, date, sampler, elements to be analysed and assays results.

Below is an extract from the Minxcon Exploration RC sampling procedures for the 2018 and 2020 drilling campaigns:

Sampling Procedure

The strategy to be adopted must be carefully designed so that at each stage of the process, the chance of taking biased, unrepresentative or contaminated samples is minimised. In order to achieve this:

- *All sampling activities should be positioned away from the cyclone “smoke stack” so as to avoid dust pollution.*
- *Samples are to be collected in 1 m intervals as per the instructions from the project geologist.*
- *New, clean sample bags are to be used at all times and for each sample.*
- *The geologist or site supervisor will ensure that the necessary sample bags are correctly labelled and available before the drillhole is drilled.*
- *All samples are to be collected from the cyclone outlet and as per indication from the drill operator.*
- *The procedure is to fasten the sample bag tightly to the (bottom) of the cyclone unit and keep it there for the duration of the sample drilled, catching the sample.*
- *The second sample assistant will take the full bag from the sampler and hand him the next marked sample bag.*
- *The sample assistant must communicate with the drill operator at all times to ensure the sample collection is done properly.*
- *The sampler under the guidance of the geologist must ensure that the cyclone is cleaned with compressed air after the sample is bagged and prior to the collection of the next sample.*
- *At the drill site, the sampler will enter all the relevant data into his triplicate book and liaise with the drilling contractor to ensure that the correct information is used on their record sheets.*
- *If any underground water is encountered during the drilling, its depth of occurrence must be recorded. If a wet sample has been obtained during drilling, a note must be recorded on the comments section of the log sheet or in the description.*

- *Each sample bag is weighed and the drillhole number, start date, end date, sample number, sample batch number, drilling system used, encountered voids and sample depth are recorded in the provided RC drilling log sheet.*
- *All samples will have to be stored in a shaded, dry storage facility or shed pending dispatch to SGS laboratory in Bamako.*

Sample Splitting

The samples are split using a riffle splitter into ½, ¼, ⅛ sample units or as per instruction from the project geologist. The split should be maintained across all drillholes. Always maintain an even feed when pouring the sample into the splitter. If the splitter does not produce the desired fraction in a single pass, repeat the process until the desired minimum mass of 2 kg (as stipulated by the project geologist) is achieved. The fraction that is not destined for the laboratory should be kept in a single marked bag (or "Archive Sample" bag) for each sample for reference and further testing if required. If this is not required (instructions issued by the project geologist) the split (that is not going to be submitted for assay) can be thrown back into the hole once the hole is completed. However, this must be discussed with the drillers and geologist before drilling starts. For the Kobada project, the remaining sample for each drilled and sampled metre is kept on a 50 kg grain bag.

The riffle splitter is to be cleaned with a clean brush/dry cloth after each sample has been divided so as to exclude all forms of possible contamination. Ideally an air hose should be used to clean the riffle.

After splitting, a portion of each sample is laid on the ground in the order of the drilling depths. The geologist or the person to carry out the logging must take note of the colours of the dry powder of the chips. This usually gives an indication of the points where changes in rock strata or rock type occur.

Chip Logging

The logging of rock chips from reverse circulation ("RC") drilling at African Gold Group's Kobada project will be done at the company's core shed.

Strict supervision of the handling of the chips is required. The geologist or the geological technician must be able to train personnel so that the purpose and importance of the whole exercise is clearly understood by all those that are involved. ("If you think training is expensive, try ignorance").

Procedure

- *After splitting the sample, scoop a portion of the chips from the bag using a handheld sieve. Shake the sieve gently to remove some of the fine dust. This should be done in such a way that the dust falls back into the bag from which the material was scooped.*
- *The scooped material must be cleaned thoroughly and gently in a bucket of water while being kept in the sieve. Care should be taken not to spill the chips into the bucket. Once the material is cleaned, it is placed in a chip tray (see* Figure 11.1*) where the depths are written on the side of the tray and a sample ticket is placed in the tray too.*

- *The sample chip trays must be clearly marked with the drillhole number and the appropriate depths per sample. The supervisor will ensure that the material placed in the sample chip trays is a good representation of each sample taken.*
- *The supervisor or geologist must check to ensure that every bag is properly labelled with the project number, drillhole number, depth, and batch number. He must also ensure that every drillhole has a representative sample and that all the samples that will go to the laboratory have been weighed and securely fastened.*

Figure 11.1: Rock Chips in Trays

- *A hand lens or magnifying glass must be used to aid in the logging process. If available, hydrochloric acid must also be used to test the chips for some elements. A record of the following should be made on the log sheet:*
    - *Depth (from-to)*
    - *Colour of powder*
    - *Colour of chips when cleaned*
    - *Grain size*
    - *Alteration types*
    - *Minerals e.g. pyrite, chalcopyrite, chalcocite, bornite etc.*
    - *Description*
    - *Graphical log*
    - *Geological Structures, etc.*

Logging is done onto the standardised data capture percussion log sheet Form 030432F2 for ease of computer data capture. This data is then captured into an excel format data base to which is added all relevant geological and survey data, for later use in geological modelling.

Transport to the Core Yard/Storage Facility

- *The transporting of the chip trays and samples safely to the sample storage area is the responsibility of geological staff as agreed before the drill programme starts. During transport, whoever is responsible must ensure that the samples are transported in a manner that will not cause loss or cross-contamination.*
- *Loading of the samples and chip trays onto the transport vehicle must be done with the utmost care and they must be delivered directly to the core yard, without any unnecessary travel or deviations.*
- *The security and integrity of the samples must be maintained at all times. Samples will be loaded onto the geologist's vehicle as soon as they have been ticketed, checked and re-checked against the sample information on the geologist's log sheet.*

Storage Facility Sampling Procedures

For each and every drillhole the following information should be known:

- *Project name*
- *Drillhole identification number (BHID)*
- *X, Y and Z coordinates*
- *Date of drilling and logging*
- *Drilling contractor and driller*
- *Geologist*
- *Final depth*

Sample Measurement – Inspection of Drillers Marking

- *All the labelled and weighed samples are sorted and ordered at the core yard.*
- *Sample bags are put in sequence to ensure that all samples are present and the "from" and "to" correspond with the sample ticket books.*
- *The samples will be packed per drillhole in order of drilling e.g. (0m to 10m or project specific).*
- *A series of standard duplicate sample ticket numbers is chosen for the sampling purpose.*

Reference materials (from African Mineral Standards) are included in the sampling sequence. Each drillhole's samples start with a blank (AMIS Blank) followed by Low Grade Standard (AMIS0432 (0,36g/t); a Duplicate and ends with a blank. Low grade and medium grade standards will be alternated. At least one High Grade (AMIS0537 (7,04g/t)) must be placed in every set of samples collected for each drillhole.

For the AFRICAN GOLD GROUP project, every 10th sample will be a QAQC reference material. In the placement of the QAQC samples, the following logic will be used at the core yard:

- *Blank sample, then 9 physical samples, a Standard (CRM), 9 physical samples, duplicate, 9 physical samples and so on, then a Blank or Duplicate at the end of the sequence.*
- *Consecutive ticket numbers are then applied to each successive individual sample section or reference material sample.*
- *The geologist/core yard geotechnician/sampler must fill in a sampling sheet.*
- *These will then be placed in a big bag and labelled with the project number, drillhole number and batch number for dispatch to laboratory. This is to ensure that the smaller samples do not get lost between all the large bags and that the lab receives them intact.*
- *The sample sequence numbers must be added to all the samples. This will also be a double check of all samples before they are dispatched.*
- *One duplicate sample ticket is placed in the bag, and the other one is stapled onto the left-hand corner inside the sample bag, after which it is closed and prepared for dispatch to the laboratory.*

Sample Submission

Samples are to be dispatched directly from African Gold Group's Camp core storage yard in Kobada to SGS analytical laboratory in Bamako.

Before sample shipment, the following steps are followed:

- *Samples are sequenced within the secure storage area and the sample sequences examined to determine if any samples are out of order or missing.*
- *The sample sequences and numbers shipped are recorded on the sample submission form, Form 030432F5. The project geologist keeps copies of all pages.*
- *The samples are placed according to sequence into large plastic/woven grain bags. The following information is recorded on the outside of the large bag:*
    - *The sample ticket number series, identified by the starting sample and end sample numbers.*
    - *The total number of samples contained within the large bag and the total samples dispatched.*
    - *The analytical request sheet/Sample Submission From (in this case, provided by SGS Laboratory) is completed, signed and dated by the project geologist before the samples are removed from secured storage.*
- *Once the above is completed and the sample shipping bags are sealed, the samples may be removed from the secured area.*
- *The geologist must sign a sample dispatch sheet, Form 030432F5 page 1.*
- *The bagged samples are collected from the core shed and checked out.*
- *The driver, and an AGG appointed assistant, then proceed directly to the laboratory and deliver the sample batch together with the Sample Submission sheet. Note that for security purposes, it is best practice that two people are used to transport the samples.*

- *The laboratory sample submission/control sheet, electronically embedded within the primary database, is filled out to accompany the samples and contains the following information:*
  - *Date*
  - *Project Name*
  - *Unique Submission/Control Sheet Number*
  - *Sample number series*
  - *Drillhole Number*
  - *Sample description: e.g. Rock chips; soil; Core etc.*
  - *Destination Laboratory*
  - *Assay instructions and Sample Storage instructions.*
- *A copy of the sample submission/control sheet, signed by the exploration manager or project geologist, accompanies the samples to the analytical laboratory and a duplicate of this form is kept by the geologist for record purposes.*
- *This sample submission sheet is signed and stamped at the laboratory on receipt of the samples. An email sample receipt is sent from the lab to the project geologist if possible.*

Below is an extract from the Minxcon Exploration diamond drilling sampling procedures for the 2019 drilling campaign:

Detailed Logging and Sample Marking and Cutting

- *Drill core trays are laid, in order, onto a logging table.*
- *The geologist checks all the core boxes to see they are all laid out in order. After this, the geologist inspects the core blocks to check if they are in their correct positions and if all the required marks are in place.*
- *If satisfied, the geologist goes on to reconcile the drillhole depths and writes/places “metre marks” onto both the rock core and the core tray. These are depth markers placed at 1 metre interval throughout the whole length of the drillhole. In the case of the Kobada drill core, the material is either too wet and clayey or very broken such that it is hard to write on it, so the marks will need to be written on the inside of the tray. To counter this, the geologist writes the metre marks onto white paper (see Figure 11.2), and where possible on the core and on the inside of the core box.*

Figure 11.2: Metre Marks

- *The geologist makes a visual inspection of the core and determines the start and end point of each horizon to be sampled and marks these points on the core with a red marker (see Figure 11.3). The geologist highlights all significant geological contacts, decides on sample lengths, and assigns sample numbers to each section of the core. In the case of the Kobada project, the sample length will be 1m except where the sample zone is guided, geologically, by two different lithologies meaning the sample length will be shorter than 1m in this instance. Samples are numbered consecutively from the top of the sampled section using commercially available numbered triplicate sample tickets.*

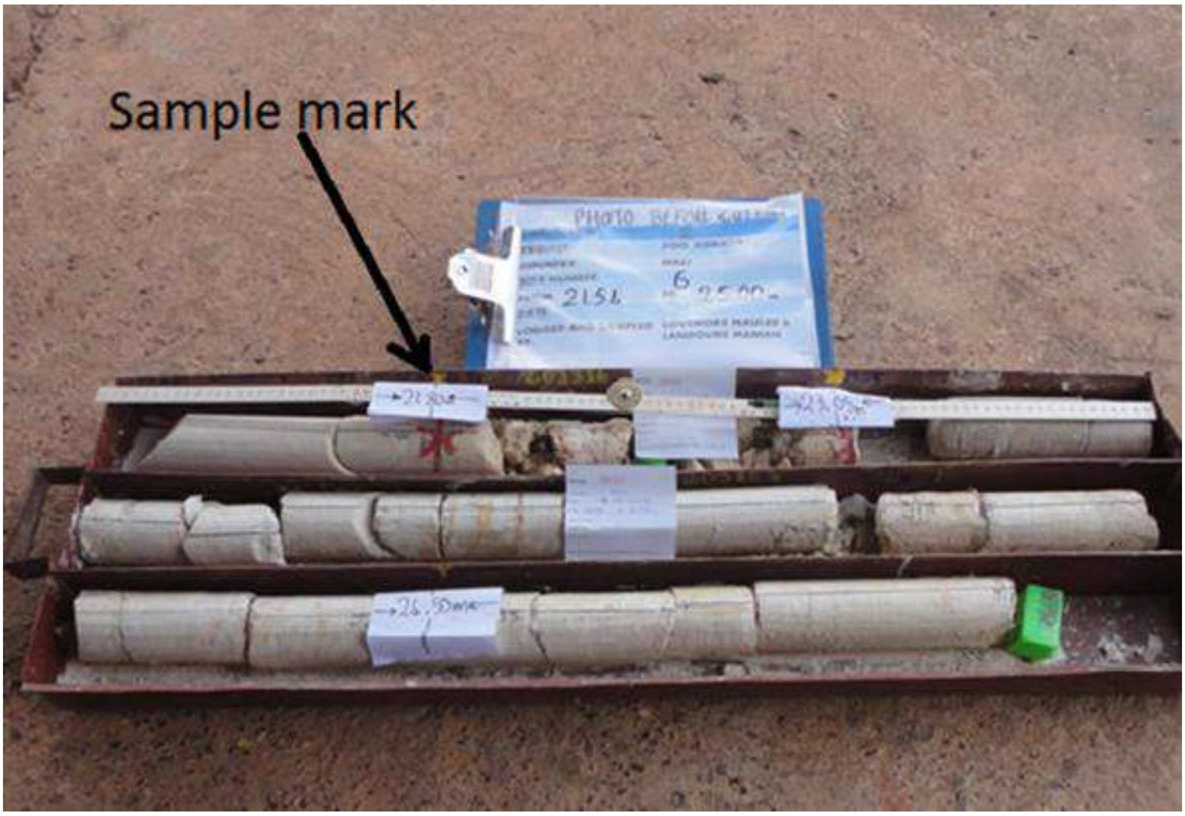


Figure 11.3: Sample Marks

- *Geologist marks the sample boundaries with a red paint marker on the core and yellow/white paint on the core tray. The sample number or identity is written in yellow/white on the inside of the core tray at the corresponding point.*
- *Where possible, the sample numbers are written on the relevant sample using the white paint marker.*
- *The sample depths ("from" and "to") are written on the core using a paint marker where possible.*
- *Cutting line is drawn down the length of the core in the plane of the observed dip of the layering (if this is visible) such that the cut must provide two identical half-cores. This would enable the geologist, the core cutter and the sampler to clearly identify the separate halves of the split core and mixing of the core would be largely avoided.*
- *Photographs of the samples are taken noting the drillhole number, date, the depth and the core box number.*
- *Marking of samples shall commence inwards from both reef contacts and each end sample must include up to a maximum of 2cm-3cm of hanging wall/footwall rock.*
- *Reference materials (CRMs--blanks, standards) and field duplicates must be included in the sampling sequence. The actual number, intervals and sequence must depend on the nature of the drilling and the target reef.*

Core Cutting

- *Broken core, wherever possible, should be taped together with masking tape before cutting into halves commences. A core-cutting machine with a specially designed diamond blade should be used to split the core.*
- *Once individual samples have been cut in half the samples are broken on the marks with a chisel and a hammer. The sample depth marks and sample numbers are marked onto the remaining half core.*
- *It is important not to use the core cutter to separate samples as one would lose material that is sawn away on the sample lines. When that material, the width of the blade, is mineralised, it could have a detrimental influence on assay values.*
- *It is imperative that the whole core is not sent for assay, i.e. physical specimens of the core are kept for future reference.*
- *Note – the operator must wear safety goggles, gloves and ear protection whilst cutting core.*
- *A series of standard triplicate sample ticket numbers is chosen for the sampling of the drillhole.*
- *Reference materials are included in the sampling sequence. Starting from the top-most section of core to be sampled, the number of the first ticket is written on the core.*
- *Consecutive ticket numbers are then applied to each successive individual sample section or reference material sample, down the core.*
- *The depth of the top and bottom boundary of each individual sample is also recorded on the remaining half core.*
- *The sampling information (description, from, to, completeness / representativeness is recorded on a specially designed diamond drilling sampling sheet, Form 030431F5 that facilitates digital capture into the database.*
- *Representativity and weight of the samples are noted on the sampling sheet.*
- *One numbered ticket, corresponding to the number written on the core, is placed into a plastic sample bag together with the sample.*
- *The second ticket, showing sample number and the sample depths, is placed in the core tray where it will be stored for future reference.*
- *The sample is weighed and the weight recorded onto the sampling sheet.*
- *The counterfoil retained in the ticket book is identified with the drillhole number and the "from" and "to" depths of the sample and project name.*
- *Individual sample bags are then placed into large poly-weave or plastic bags and sealed with a cable tie.*
- *Not more than 15 sample bags of half core samples are placed into each poly-weave bag.*
- *The sample information is captured into the database, to which is added all relevant geological and survey data, for later use in geological modelling.*

Sample Submission

On a weekly basis (or more often if required) samples are despatched directly to the SGS analytical laboratory in Bamako, Mali.

Before sample shipment, the following steps are followed:

- *Samples are sequenced within the secure storage area and the sample sequences examined to determine if any samples are out of order or missing.*
- *The sample sequences and numbers shipped are recorded on the analytical request form.*
- *The samples are placed according to sequence into large plastic/grain bags. The following information is recorded on the outside of the large bag:*
    - *The sample series identified by the starting sample and end sample numbers.*
    - *The total number of samples contained within the large bag.*
- *The sample submission form is supplied by SGS, Mali and it includes portions to be filled with relevant information.*
- *Once the above is completed and the sample shipping bags are sealed, the samples may be removed from the secured area.*
- *The bagged samples are collected from the core shed and checked out on a dispatch note. The driver and assistant then proceed directly to the laboratory and deliver the sample batch with the lab request sheet. Note that for security purposes, AGG sends one of its geological technicians to accompany the samples to the laboratory where he will also be responsible for checking them in.*
- *The laboratory sample submission/control sheet electronically embedded within the primary database, is filled out to accompany the samples and contains the following information:*
    - *Date*
    - *Project Name*
    - *Unique Submission/Control Sheet Number*
    - *Sample number series*
    - *Drillhole Number*
    - *Sample description: e.g. half core or quarter core*
    - *Destination Laboratory*
    - *Assay instructions*
- *A copy of the sample submission/control sheet, signed by the exploration manager or senior geologist, accompanies the samples to the analytical laboratory and a duplicate of this form is kept by Minxcon Exploration for record purposes.*
- *This sample submission sheet is signed and stamped at the laboratory on receipt of the samples. An email sample receipt is sent from the lab to the Minxcon geologist.*

Quality Control Procedures

Three types of quality control materials will be included in sample consignments viz. standards, blanks and duplicates, discussed in Section 11.3.

11.2 SAMPLE PREPARATION AND ANALYSIS PROCEDURES

Limited data is available for sample preparation and analysis procedures, and often the laboratory package code is the only indication of analytical procedures. This applies especially from 1988 to 2009. From 2009 onwards, detailed information and reports are available (see Table 11.1).

Table 11.1: Analytical Procedure per Company, Year and Drillhole Type

Company	Year	Hole Type	Laboratory Package Code	Analysis	Analytical Procedure	Laboratory
BRGM	1988	DD	FA10-FA15	Fire Assay	Fire Assay, 10 g to 15 g aliquot	Dakar or Orleans
La Source	1996	RC	FA30 at BRGM	Fire Assay	Fire Assay, 30 g aliquot	unknown
Cominor	2002	RC/AC	FA_cominor	Fire Assay	Fire assay	unknown
Cominor	2004	RC	FA_cominor	Fire Assay	Fire assay	unknown
AGG	2005	DD	FA30 at ETL/ SFA at ETL	Fire Assay and SFA	Fire assay/screen fire assay, 30 g aliquot, AAS finish	ETL
AGG	2005	DD	ICP at ETL	ICP	Inductively coupled plasma	ETL
AGG	2006	DD	FA50	Fire assay	Fire Assay, 50 g aliquot, AAS finish	ABI/ALS
AGG	2006	DD	SFA1000	Screen Fire Assay	Screen fire assay	ABI/ALS
AGG	2006	DD	CNLW	Possibly Fire Assay	Leachwell	ABI/ALS
AGG	2007	DD	FA50	Fire Assay	Fire Assay, 50 g aliquot, AAS finish	ABI/ALS
AGG	2007	RC	FA50	Fire Assay	Fire Assay, 50 g aliquot, AAS finish	ABI/ALS
AGG	2007	DD	SFA1000	Screen Fire Assay	Screen fire assay (assumed 50 g aliquot, AAS finish)	ABI/ALS
AGG	2007	DD	SFA1000_freshrx	Screen Fire Assay	Screen fire assay (assumed 50 g aliquot, AAS finish)	ABI/ALS
IMG	2009	RC/DD	AGG_leachwell	Leachwell	Leachwell 2 kg	ALS2
AGG	2009–2010	RC/DD	AGG_leachwell	Leachwell	Leachwell 2 kg	ALS2
AGG	2011	RC	AGG_leachwell	Leachwell	Leachwell 2 kg	ALS2
AGG	2012	RC	AGG_leachwell	Leachwell	Leachwell 2 kg and Fire assay, 50 g (assumed AAS finish)	ALS2
AGG	2012	DD	AGG_leachwell	Leachwell	Leachwell 2 kg and Fire assay, 50 g (assumed AAS finish)	ALS1, ALS2
AGG	2015	RC	AGG_Leachwell	Leachwell	Leachwell 2 kg	ALS2

AAS: Atomic Absorption Spectroscopy
AC: Air Core
ALS/ABI: Abilab Bamako, renamed ALS Bamako
ALS: ALS Bamako
ALS2: ALS Ouagadougou
CNLW: Unknown
ETL: Eco tech Lab, Kamloops, BC, Canada
FA: Fire Assay
ICP: Inductively coupled plasma mass spectrometry
IMG: IAMGold
LW: Leachwell
SFA: Screen Fire Assay

Sample preparation and analysis were performed at the laboratories specified in Table 11.1, apart from AGG in 2015, where sample preparation was performed at SGS Bamako, and sample analysis was performed at ALS Ouagadougou. Where the size of the sample taken was documented, it is also mentioned in Table 11.1.

During the 2018–2020 drilling campaigns, the RC and DD samples were submitted to the SGS Laboratory in Bamako, Mali, which is a SANAS (South African National Accreditation System) accredited laboratory (facility accreditation number T0762). The laboratory operates a quality system in accordance with ISO/IEC 17025. The samples were tested by fire assay with an AAS finish. Samples < 3.0 kg are dried in trays, crushed to a nominal 2 mm using a jaw crusher, and then < 1.5 kg is split using a Jones-type riffle splitter. The reject sample is retained in the original bag and stored. The sample is pulverised in an LM2 pulveriser to a nominal 85 % passing 75 µm. An approximately 200 g subsample is taken for assay, with the pulverised residue retained in a plastic bag. All the preparation equipment is flushed with barren material prior to the commencement of the job. A 50 g subsample is fused with a litharge-based flux, cupelled, and the prill is dissolved in aqua regia, and gold is determined by flame AAS (Detection Limit 0.01 ppm).

All the samples were core samples of HQ or NQ size. The laterite, saprolite and transition zones were drilled using HQ size, and the fresher sulphide zone was drilled using NQ size.

The deeper 2020 RC drillholes were completed with diamond drill tails to test the sulphide zone.

11.3 QUALITY ASSURANCE AND QUALITY CONTROL

Varying levels of quality assurance and quality control (QAQC) were performed for each of the years and companies. The more recent programmes were better documented, and the actual procedures followed were more complete, while earlier programmes focused only on duplicate analyses.

The years are summarised as follows:

- < 2004:
    - No QAQC data is available to comment on.
- 2004:
    - Some detail on check samples/duplicate assays for DD drillholes, by FA, showing poor results.
- 2005:
    - Duplicate analysis by ICP is available, showing a good correlation with the original.
- 2006, 2007:
    - Only duplicate samples are available, no actual QAQC results or standards can be sourced, duplicates show only minor correlation.
    - CRMs and blanks plotted in other reports for 2007 show only the results, which are good. The actual data is not available to examine.

- 2009, 2010, 2011, 2012:
    - Detailed QAQC from two CRMs, blanks and duplicates, as well as FA to LW comparisons are available.
    - Blanks show a reasonable pass rate with minor failures.
    - TheCRM show significant failures from 2009 to 2012, analytical techniques are predominantly LW. Potentially weighting and assigning of weighting to these results were done incorrectly. This is seen consistently over four years for both standards.
    - AGG reports, as well as independent plots by Minxcon, confirm these results.
    - LW shows poor repeatability. While FA to FA show good results, FA versus LW tests do not show a good correlation. This is possibly due to the high variability and nugget effect inherent in the samples and is exacerbated by errors in sampling and analytical procedures.
    - No criteria were specified for failure of batches due to exceeding specified limits, and when checked, the original assays were retained in the final database, thus no re-assaying was performed on failing batches.
- 2015:
    - QAQC data results, including blank and CRM analyses, are seen in the reports, but the actual data is not available.
    - Results are good.

A detailed discussion on each individual year is given below.

11.3.1 QAQC 2004

A total of 64 check assays are available; it is unclear if these are duplicate analyses or umpire analyses at a different laboratory. These do not show a good correlation or repeatability (see Figure 11.4 and Figure 11.5). It is also unclear what the analysis methodology is for the check assays; the analysis methodology for the original assay is FA.

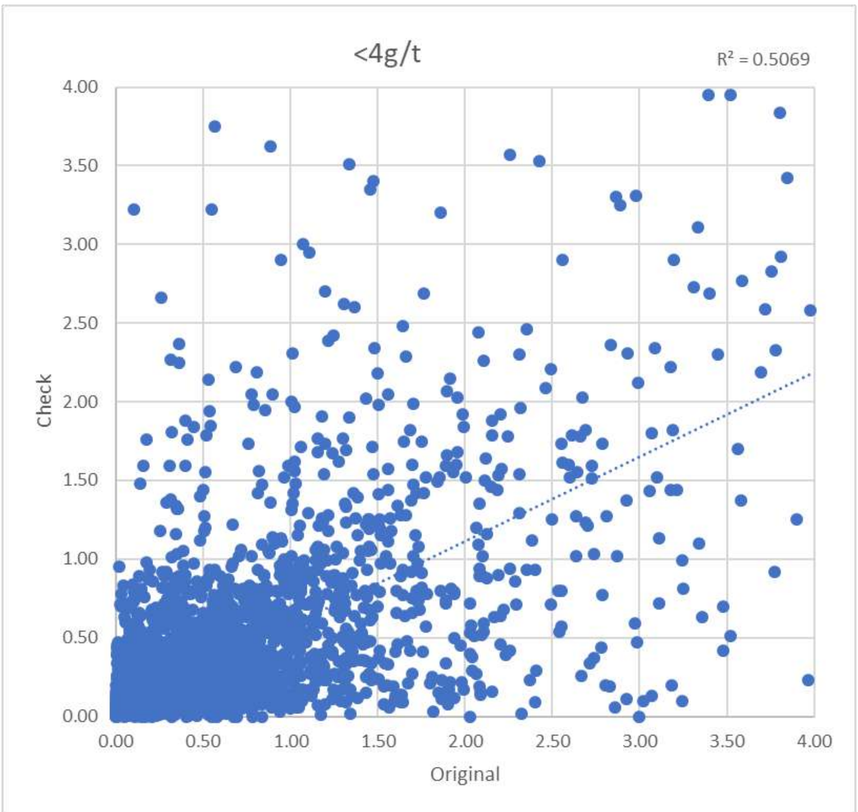


Figure 11.4: Check Assays for 2004 Cut-off at 4 g/t

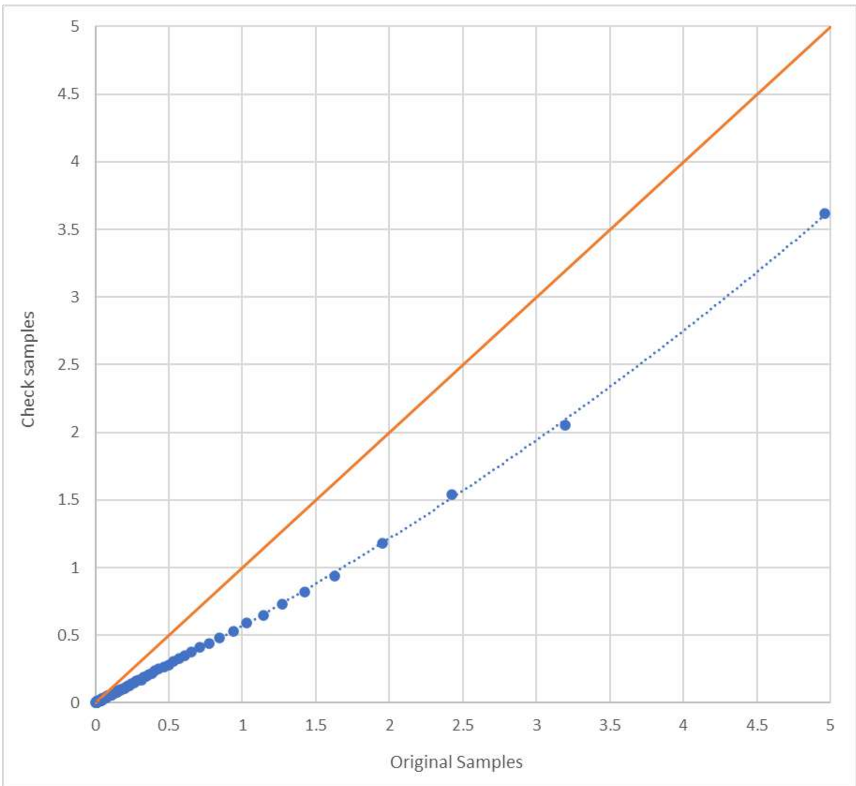


Figure 11.5: Quantile Plot for Check versus Original Samples

11.3.2 QAQC 2005

For 2005, there were ICP and FA samples. The comparison between the two is shown in Figure 11.6. The scale is limited to 1 g/t to display the bulk of the data; other higher-grade values are in the minority. The reproducibility of the results is within acceptable limits. At higher grades, repeatability is more erratic; however, this is expected from this style of mineralisation, where the methodology of sample splitting and the distribution of the grade/vein in the sample in question will affect the reproducibility greatly.

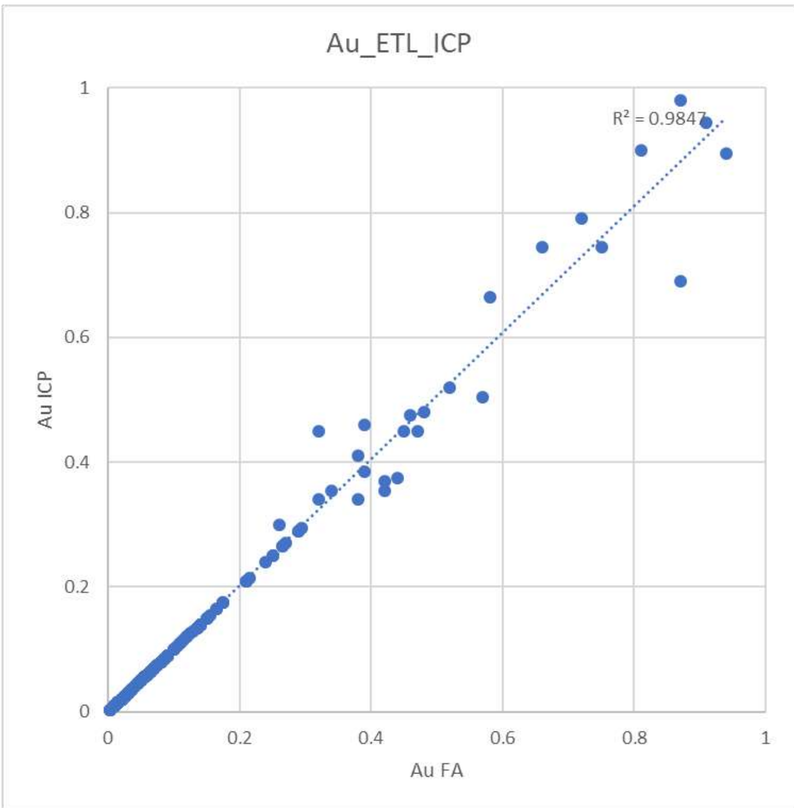


Figure 11.6: ICP to FA Comparison for 2005

11.3.3 QAQC 2006–2007

For 2006 and 2007, FA results were taken as the final values, with one set of duplicates (also FA) to test these values (see Figure 11.7). Some SFA samples are available for 2007, but these did not correlate well with the original.

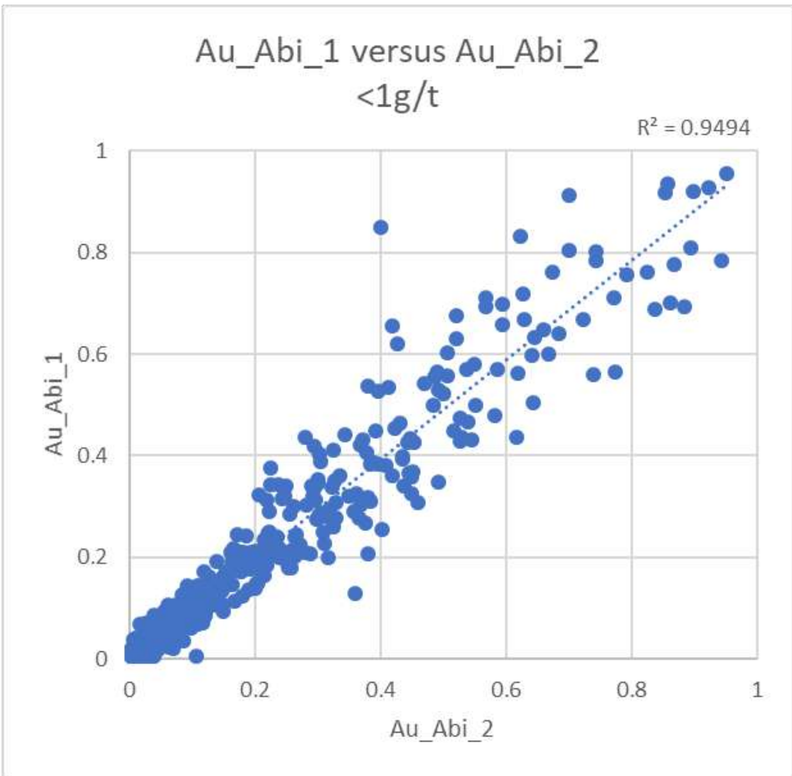


Figure 11.7: Duplicate Analysis for 2006–2007, Both FA

CRM and blank information is available in the previous feasibility report (Wolfe et al, 2016); however, the actual data was not available for review. No issues were identified, and no bias or issues were identified with the blanks and CRMs presented.

11.3.4 QAQC 2009–2012

For the period 2009 to 2012, CRMs, blanks and duplicates are available. In addition, every 30th and 100th sample was assayed with an additional method (FA/SFA). For this period, all the CRMs failed consistently.

CRMs were diluted with blank material and blended to be used with a Leachwell analysis (as part of the same sample run): typically 100 g of CRM was mixed with 3 kg of blank, and the relevant weights were used to weigh and calculate the expected value (as the original CRM is a portion of the total 3.1 kg sample).

The opinion from AGG (2015):

The fault is likely to be inherent in the method of preparation of the standard. The balance used for weighing the blank material is likely to have not been correctly calibrated and the homogenisation of the blended material is likely to be poor, also no weighted average calculation was used when the standard was blended with blank material. The possibility should not be excluded that the low mean assay value is related to the fact that Leachwell is a partial analysis and fire assay is considered total.

It is agreed that it is likely that the blending process for CRMs for Leachwell may be the cause of this. Measuring a 100 g sample versus a 3 kg sample within the same process run is likely something the laboratory is not set up to do, and thus greater inaccuracy would occur in measuring this smaller sample size. In addition, when taking various measurements and sample proportions, other errors can occur. For example, if the CRM is incorrectly tagged as a 50 g CRM and 3 kg blank versus a 100 g CRM, the correct result cannot be determined.

If there is inconsistency in the weighing of the blank sample, as well as inadequate blending, then the results will differ greatly. Note, only QAQC of CRMs and blanks of limited drillholes are available, for example, in 2012, only nine RC drillholes out of 228 RC and 10 DD drillholes are available.

Two examples are shown for the two CRMs used, G306-4 (see Figure 11.8) and G901-8 (see Figure 11.9). All the years and all the CRMs show the same trend, with some minor failures lying well outside the bulk of the population, which would constitute batch failures as part of any QAQC programme. However, the bulk of the data consistently shows a lower-than-expected value (negative bias).

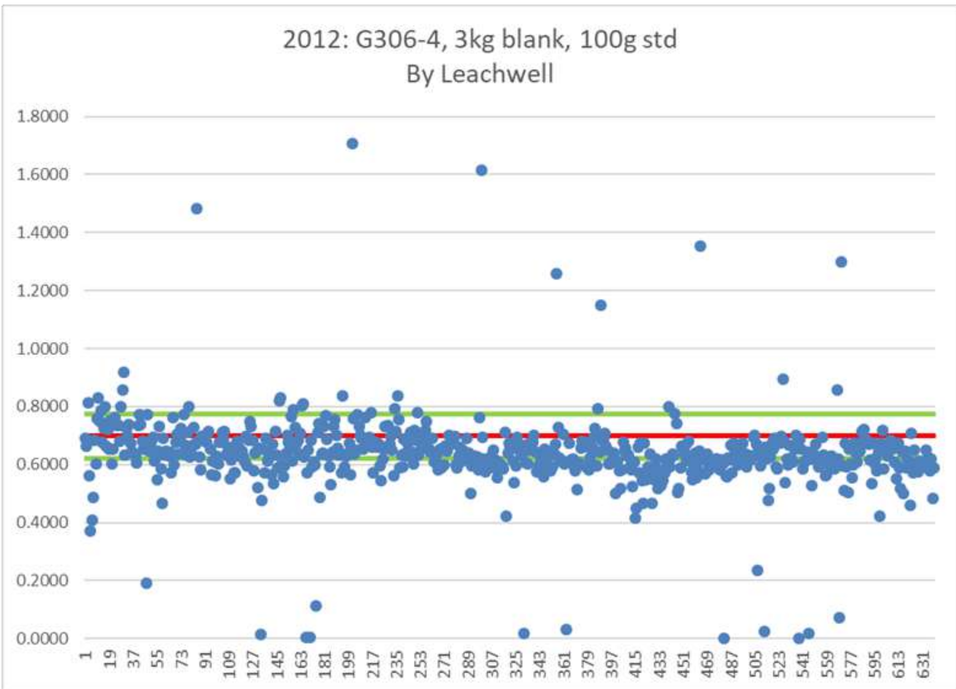


Figure 11.8: CRMs Results for G306-4 for 2012

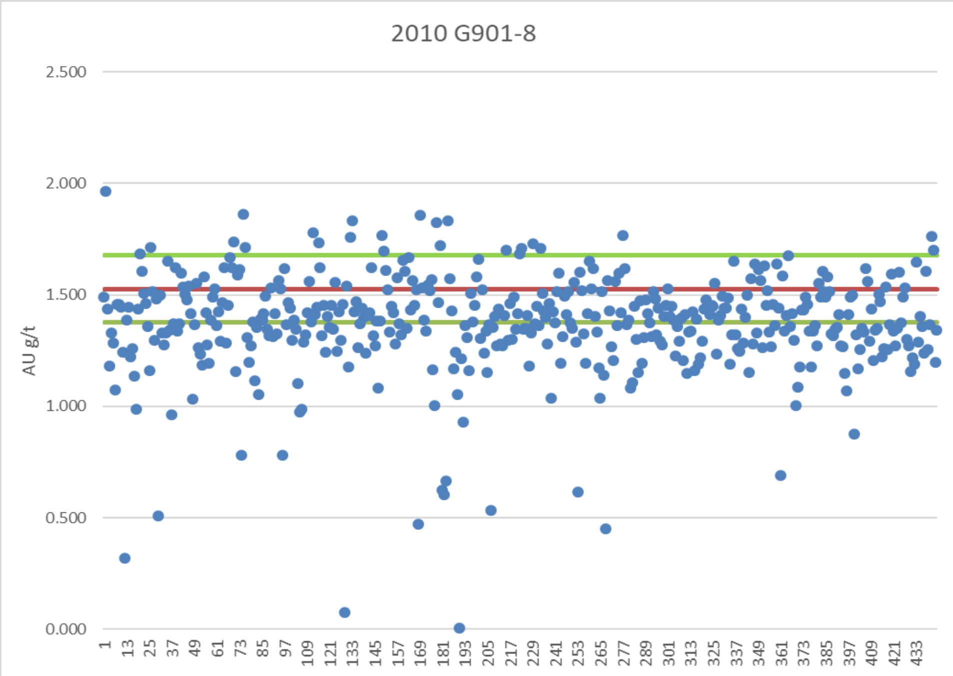


Figure 11.9: CRM Results for G901-8 for 2010

During 2011, various ratios of CRM to sample were tested (see Table 11.2). This was likely to identify the cause of the poor performance of the results. No difference in these various methods for 2011 were observed. Table 11.2 and Table 11.3 summarise the results of the standards for 2009–2012. There is a very low pass rate for all the years. In addition, the percentage deviation below the confidence limits is significantly higher than the deviation above the confidence limits. There is consistent under-evaluation of the CRMs through all years.

Table 11.2: CRMs Results and Passing Rates

Year	CRM	Ratio	Mean (g/t)	Lower (g/t)	Upper (g/t)	Results Total		Passing	
						Average	Count	Count	Percentage
2009	G306-4	3 kg/100 g	0.70	0.62	0.77	0.76	34	21	62 %
2009	G901-8	3 kg/100 g	1.53	1.38	1.68	1.50	33	20	61 %
2010	G306-4	3 kg/100 g	0.70	0.62	0.77	0.63	465	205	44 %
2010	G901-8	3 kg/100 g	1.53	1.38	1.68	1.37	444	208	47 %
2011	G306-4	3 kg/100 g	0.70	0.62	0.77	0.64	762	415	54 %
2011	G306-4	3 kg/50 g	0.36	0.32	0.39	0.41	26	11	42 %
2011	G306-4	1.5 kg/20 g	0.29	0.25	0.32	0.26	292	97	33 %
2011	G901-8	3 kg/100 g	1.53	1.38	1.68	1.00	657	398	61 %
2011	G901-8	3 kg/50 g	0.78	0.70	0.85	0.70	27	9	33 %
2011	G901-8	1.5 kg/20 g	0.62	0.56	0.68	0.55	220	70	32 %
2012	G306-4	3 kg/100 g	0.70	0.62	0.77	0.64	643	340	53 %
2012	G901-8	3 kg/100 g	1.53	1.38	1.68	1.39	604	304	50 %

Table 11.3: CRMs Above the Confidence Limits and Below the Confidence Limits

Year	CRM	Ratio	Above 3StdDev				% Change to Total	Below 3StdDev				% Change to Total
			Excluded True Outliers	Count	% of Total	% Deviation		Excluded True Outliers	Count	% of Total	% Deviation	
2009	G306-4	3 kg/100 g	2	4	12 %	5 %	0.6 %	2	5	15 %	−23 %	−3 %
2009	G901-8	3 kg/100 g	2	3	9 %	2 %	0.2 %	2	6	18 %	−23 %	−4 %
2010	G306-4	3 kg/100 g	9	14	3 %	3 %	0.1 %	0	237	51 %	−28 %	−14 %
2010	G901-8	3 kg/100 g	1	23	5 %	4 %	0.2 %	15	197	44 %	−25 %	−11 %
2011	G306-4	3 kg/100 g	13	57	7 %	7 %	0.6 %	33	244	32 %	−28 %	−9 %
2011	G306-4	3 kg/50 g	3	4	15 %	6 %	0.9 %	2	6	23 %	−28 %	−6 %
2011	G306-4	1.5 kg/20 g	2	47	16 %	14 %	2.3 %	5	141	48 %	−31 %	−15 %
2011	G901-8	3 kg/100 g	1	36	5 %	9 %	0.5 %	25	197	30 %	−25 %	−7 %
2011	G901-8	3 kg/50 g	1	4	15 %	11 %	1.7 %	4	9	33 %	−25 %	−8 %
2011	G901-8	1.5 kg/20 g	3	23	10 %	8 %	0.9 %	7	117	53 %	−30 %	−16 %
2012	G306-4	3 kg/100 g	3	33	5 %	21 %	1.1 %	14	253	39 %	−25 %	−10 %
2012	G901-8	3 kg/100 g	3	31	5 %	9 %	0.4 %	17	249	41 %	−24 %	−10 %

NOTE:
An additional calculation is performed to determine the effect of this deviation on the total data set (% change to total) This is done by multiplying the percentage of the total database affected by these outliers by the percentage of the deviation.

True outliers are excluded from Table 11.2 and Table 11.3, as the purpose of this exercise is to assess the bulk of the data set that is not passing the QAQC standards (> 3StdDev or < 3StdDev). As a general trend is seen where the results are shifted as a whole, the purpose is to quantify this shift, if possible, without it being skewed by a few outliers (which are expected). If a consistent trend or bias is seen, then it can be decided if this bias can be quantified and applied to the confidence in the data set.

Reviewing the percentage change to the total, for the samples plotting above the 3rd standard deviation, the effect is 0.8 % on average to a maximum of 2.3 % This deviation is not deemed to have a significant effect on the total database. For the samples plotting below the 3rd standard deviation, the effect is 10 % on average, with three samples plotting above 14 %.

This data set shows a consistent negative bias, it is thus proposed that the high analytical precision be seen with low accuracy. With a consistent bias averaging −10 %, the bias may mean that the estimates are conservative (underestimating). It is proposed that any area estimated by samples during this period be confirmed by samples from other drilling programmes with higher confidence. This is discussed in the sections below.

The blank values show good results, with minimal failures (see Figure 11.10).

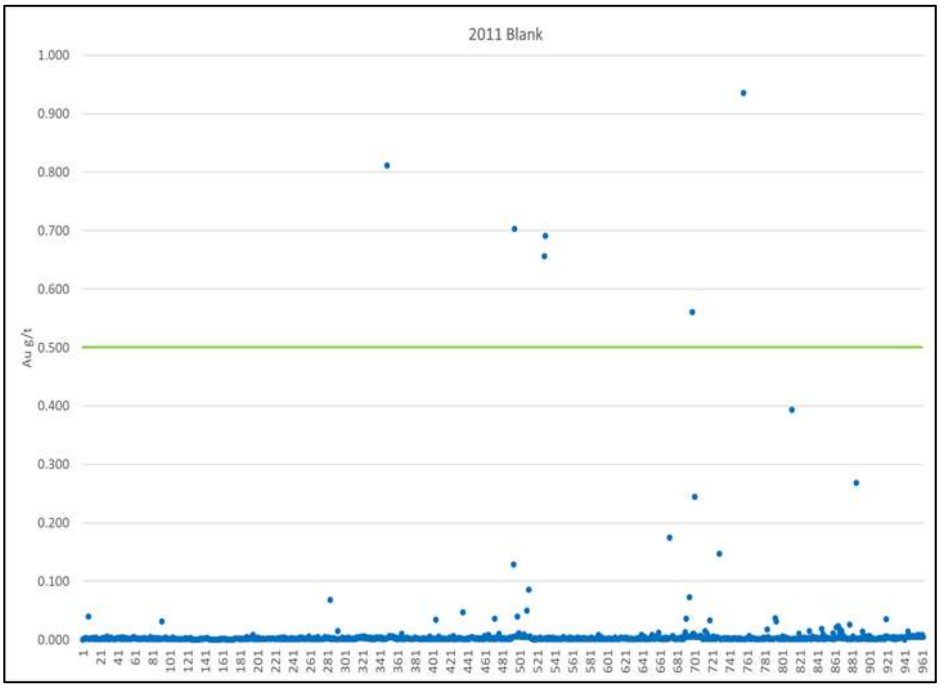


Figure 11.10: Analysis of Blanks Results for 2011

A duplicate analysis was performed for the years 2009 to 2012, and all the years showed poor results (see Figure 11.11). However, it must be noted that due to the type of deposit and narrow veins hosting the mineralisation, duplicates and twinning typically show poor results. Thus, the reproducibility of the results must be considered with this in mind.

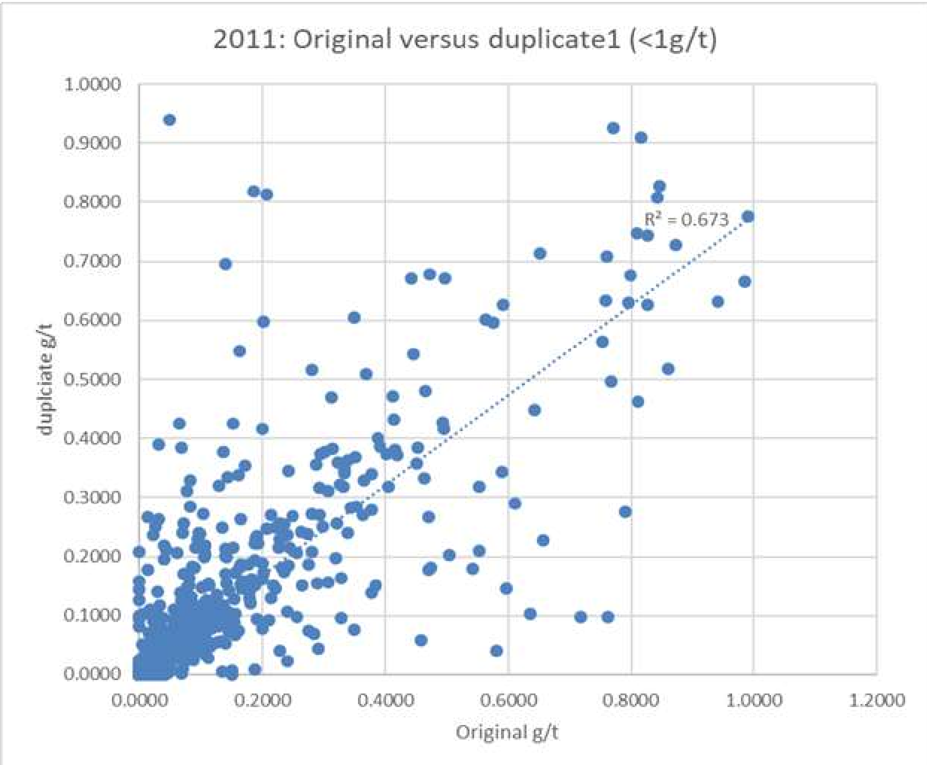


Figure 11.11: Leachwell Duplicate Results for 2011

An umpire analysis was performed at SGS, and a poor reproducibility (see Figure 11.12) of results is seen between the LW results and the FA results from SGS. The umpire analysis was performed on the 30th and 100th samples. Where internal duplicates were analysed, some of these were also sent to the umpire laboratory.

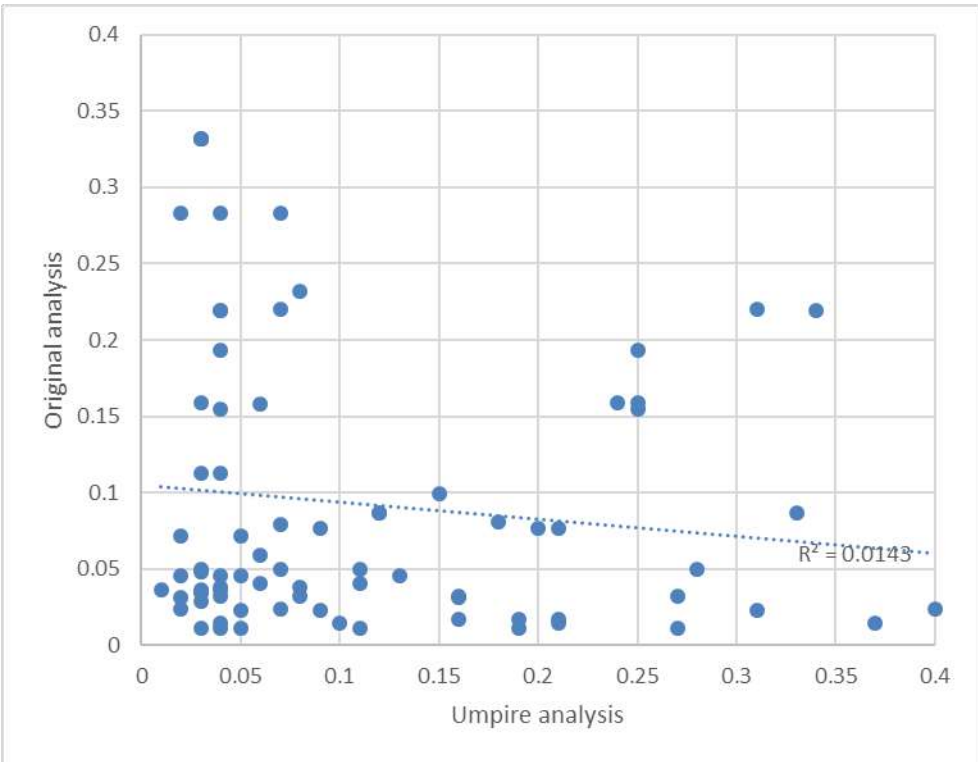


Figure 11.12: Umpire Laboratory Results Compared to Original Results for 2011

In contradiction to AGG's conclusions in their 2015 report, the LW and FA results do not compare well (see Figure 11.13). This is seen throughout all the types of duplicate analysis and may be a function of the high variability inherent in the deposit. There is a negative bias in all the LW results that are being compared, so a good correlation is not expected. FA to FA comparisons show the best correlations. All the types of duplicate analysis show poor repeatability and high variability due to the style of mineralisation.

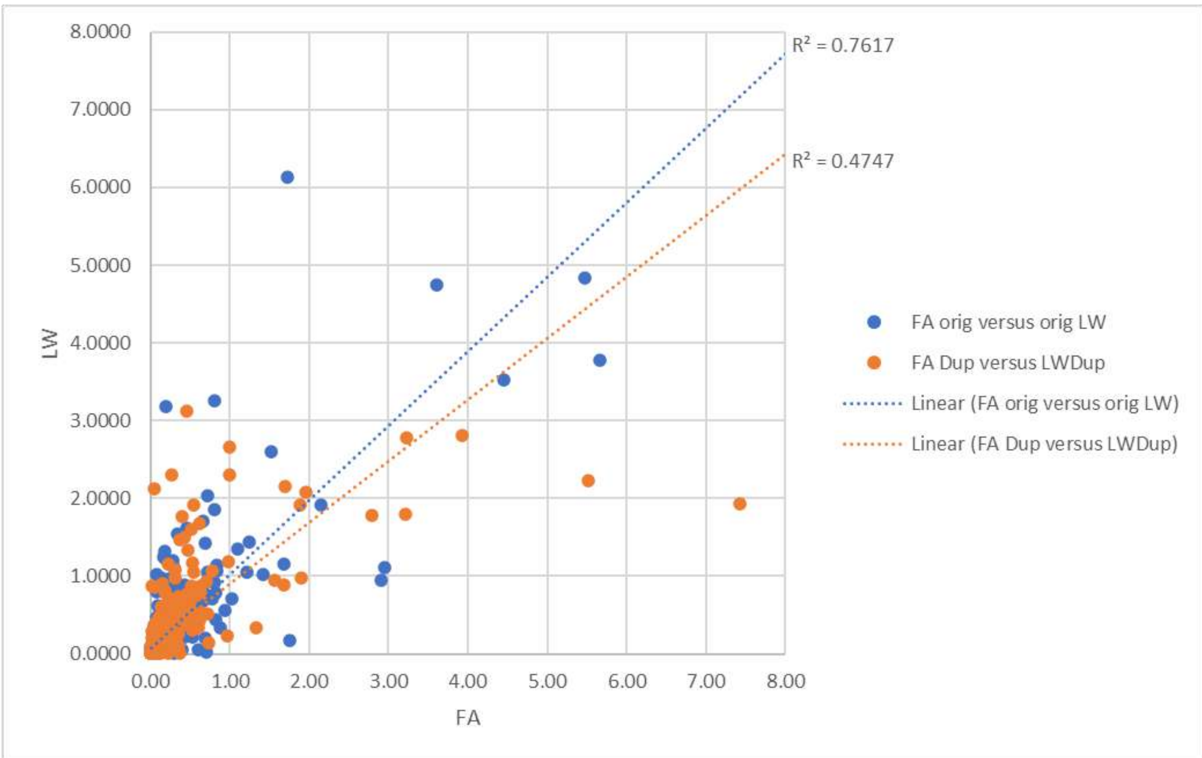


Figure 11.13: Leachwell to Fire Assay Comparisons

11.3.5 QAQC 2015

A detailed QAQC programme was performed for 2015 with blanks, duplicates and CRMs inserted (see Table 11.4) into the sample sequence.

Table 11.4: QAQC Programme for 2015

Sample Number	Type of QAQC	Selected QAQC Material
10	Std1	Std. OxG103
19/20	Field Dup	Parent/daughter
30	Std2	Std: OxD87
39/40	Field Dup	Parent/daughter
50	Blank	Waste material
59/60	Field Dup	Parent/daughter
70	Std3	Std G901-7
79/80	Field Dup	Parent/daughter
90	Std1	Std. OxG103
100	Blank	Waste material

No QAQC data is available for this period. The AGG report (2015) states that the results for all the CRMs and blanks are good. The previous bias seen in 2009–2012 is absent. Sample preparation was performed at SGS and the LW analysis at ALS, and all the results showed a good correlation. However, the duplicates analysis showed a poor correlation.

11.3.6 Leachwell and Screen Fire Assay

SFA should be routinely performed following LW to test the residual grade. LW is a partial analysis technique while fire assay is a total analysis for the precious metal. The SFA grade should be added to the LW result to fully account for the total gold content. At Kobada, SFA was performed randomly on too few samples to provide reliable ratios of leachable gold to total gold. The LW results are currently being used with a non-quantitative understanding that there will be slight underestimation due to the LW technique.

Based on the limited SFA results following LW, it is currently believed that the unleachable gold is insignificant. Table 11.5 shows the effect of the addition of the SFA results to the existing LW results in the database. The “samples with SFA samples” column shows the samples that had an SFA analysis performed following the LW analysis (it is only a portion of the total). While the “samples with SFA samples in the database” column shows what exists in the final database. The difference between these two can be accounted for by trace samples in the SFA results, often not carried over or entered into the final database. Thus, there is no real loss of actual samples or missing samples as these trace values for SFA are not used to recalculate the final Au. These SFA results from the original analysis were then added to the LW samples, and the statistics were calculated to determine the effect on the database by the inclusion or exclusion of these SFA results after LW. By comparing the mean values, the addition of the SFA samples has a minimal effect on the mean values of the SFA database, and when comparing this to the total data set, a very minor difference is seen. Also noted is the very small number of SFA results above trace in relation to the total samples (see Table 11.5). These SFA analyses were performed only on a small portion of the LW samples. For this reason, it was decided to exclude these SFA samples from the final data set.

Table 11.5: SFA Sample Comparison

Year	Total Samples	Samples with SFA Samples	SFA Values Above Trace	Samples with SFA Samples in Database	Samples Changed from Original	Total Database		SFA Samples Only	
						Original Mean	New Mean	Original Mean	New Mean
2009	2,427	134	15	134	15	0.413	0.414	0.206	0.216
2010	18,944	1,179	165	1,000	191	0.317	0.318	0.340	0.375
2011	26,449	1,415	163	1,359	419	0.226	0.235	0.241	0.256
2012	32,888	1,415	266	1,188	322	0.366	0.373	0.360	0.406
Total	**80,708**	**4,143**	**609**	**3,681**	**947**	**0.331**	**0.335**	**0.287**	**0.313**

11.3.7 QAQC 2004–2015 Conclusions

The low level of repeatability in the field duplicates and different laboratory methods may be due to the nature of the mineralisation in the deposit where large samples containing high-grade mineralised vein quartz might not have been adequately blended.

11.3.8 QAQC 2018–2019

AGG implemented the QAQC protocols for the drilling campaign to ensure consistent quality. Every 10th sample was either a blank sample, a CRM of various grades, or a duplicate sample. This equates to approximately 10 % of the samples.

11.3.8.1 Standard Reference Materials

Standard reference materials (CRMs) are used to assess the accuracy and possible bias of the assay values. The CRMs are stored in sealed packets (explorer packs – 110 g), and considerable care is taken to ensure that they are not contaminated in any manner. These explorer packs are placed sealed into the sample bag but with the label removed.

The CRMs were purchased from African Mineral Standards (https://www.amis.co.za), and the details of the CRMs are as follows:

- AMIS0569: Greenstone belts are zones of variably metamorphosed mafic to ultramafic volcanic sequences with associate sedimentary rocks that occur within Archaean and Proterozoic cratons between granite and gneiss bodies. Greenstone belts are primarily formed of volcanic rocks, dominated by basalt, with minor sedimentary rocks inter-leaving the volcanic formations. The material comes from the Greenstone belt in Barberton in the Mpumalanga Province, South Africa. Gold: 0.271 g/t ± 0.031 g/t two standard deviations (2s).
- AMIS0441: The material for AMIS0441 was provided by Vantage Goldfields from the Taylors Mine section of the Barbrook Mine, which is located 25 km east-northeast of the town of Barberton in the Mpumalanga Province, South Africa. The mine is situated in the Archaean Barberton Greenstone Belt. Gold is contained in mesothermal veins associated with late tectonic shears and fractures within rocks of the Barberton Supergroup, which comprises an assortment of ultramafic and mafic submarine volcanics, turbiditic greywacke sandstones and shales. Intense shearing, massive quartz veining, silicification of BIF and sulphide enrichment characterise the larger orebodies. Gold: 2.44 g/t ± 0.23 g/t.
- AMIS0559: Greenstone belts are zones of variably metamorphosed mafic to ultramafic volcanic sequences with associate sedimentary rocks that occur within Archaean and Proterozoic cratons between granite and gneiss bodies. Greenstone belts are primarily formed of volcanic rocks, dominated by basalt, with minor sedimentary rocks inter-leaving the volcanic formations. Gold: 12.01 g/t ± 0.82 g/t (2s).

The results for the above CRMs are presented in Figure 11.14 to Figure 11.16.

QAQC Results for AMIS0569 (see Figure 11.14): No low-grade QAQC samples failed, and all fell within the 3rd standard deviation. However, the samples did consistently plot above the certified mean, and hence a selection of samples were sent as umpire samples to ALS Chemex South Africa (SANAS accredited – T0387) in Edenvale, Johannesburg, South Africa to test for bias. No significant bias was observed from the umpire samples.

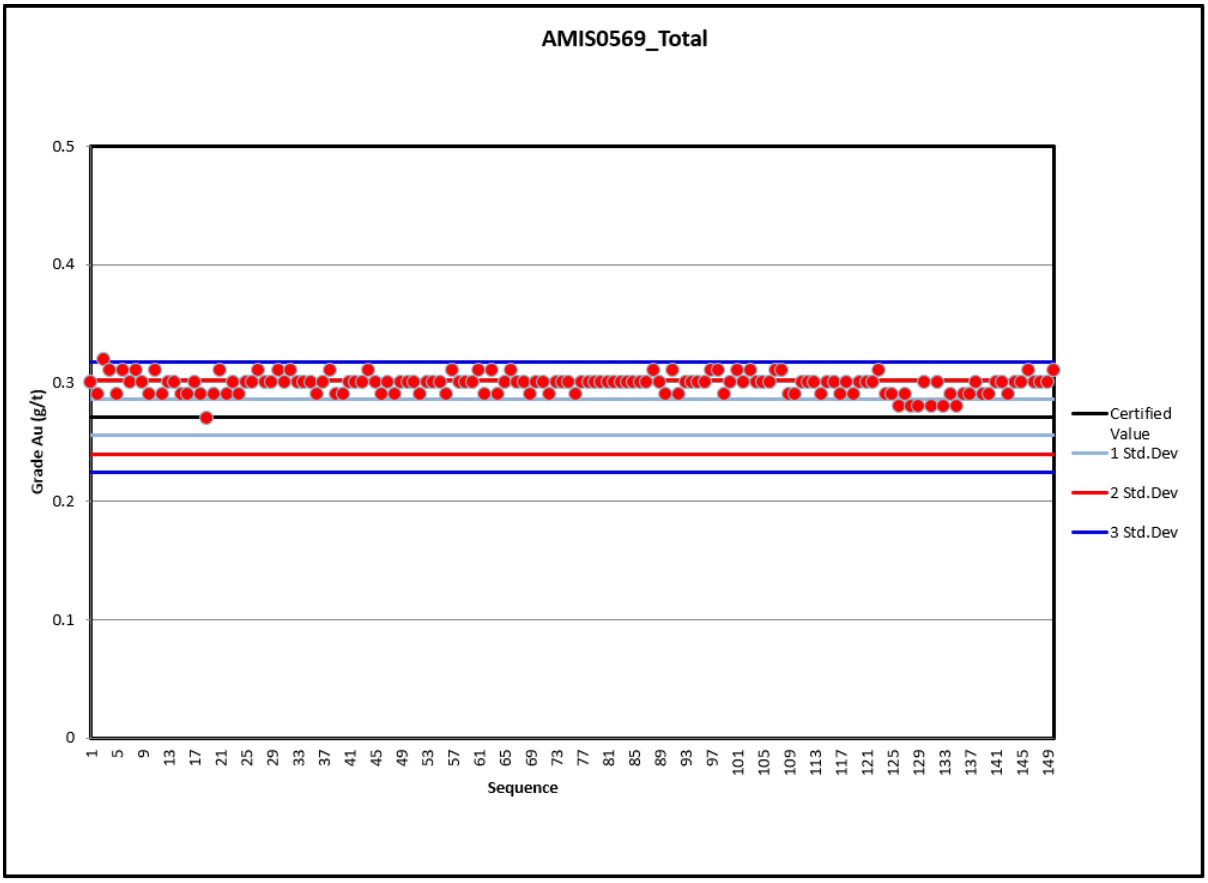


Figure 11.14: QAQC Results for AMIS0569

QAQC Results for AMIS0441 (see Figure 11.15): No medium-grade QAQC samples failed, and all the samples fell within the 2nd standard deviation.

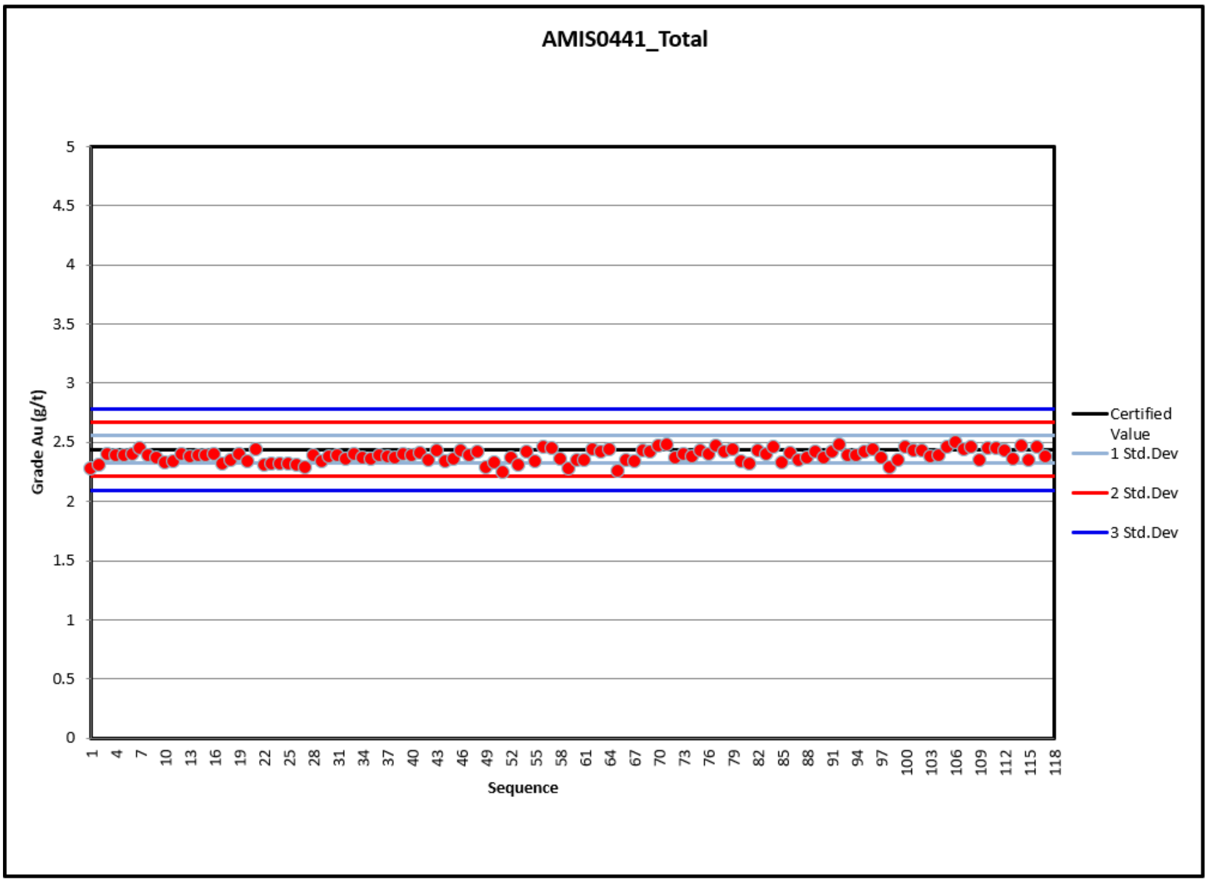


Figure 11.15: QAQC Results for AMIS0441

QAQC Results for AMIS0559 (see Figure 11.16): No high-grade QAQC samples failed, and all fell within the 3rd standard deviation. However, the samples did consistently plot above the certified mean, and hence a selection of samples were sent as umpire samples to ALS Chemex South Africa to test for bias. No significant bias was observed from the umpire samples.

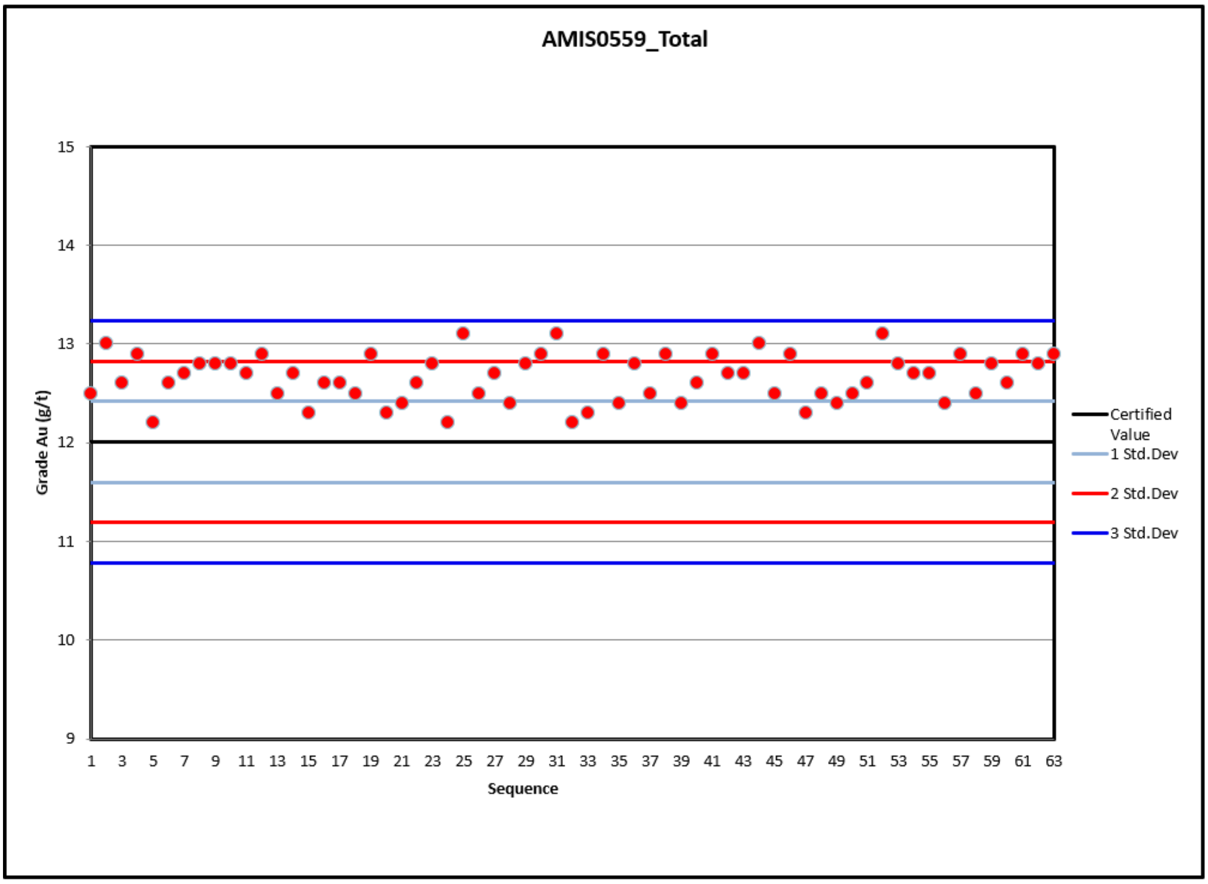


Figure 11.16: QAQC Results for AMIS0559

11.3.8.2 Blanks

The insertion of blanks provides an important check of the laboratory practices, especially potential contamination or sample sequence mis-ordering. Each drillhole started and ended with a blank. No blank samples failed.

The blank was purchased from African Mineral Standards (https://www.amis.co.za), and the details are as follows: AMIS0681: Blank – barren silica sand. Gold: 0 g/t.

The results for the above blank are presented in Figure 11.17.

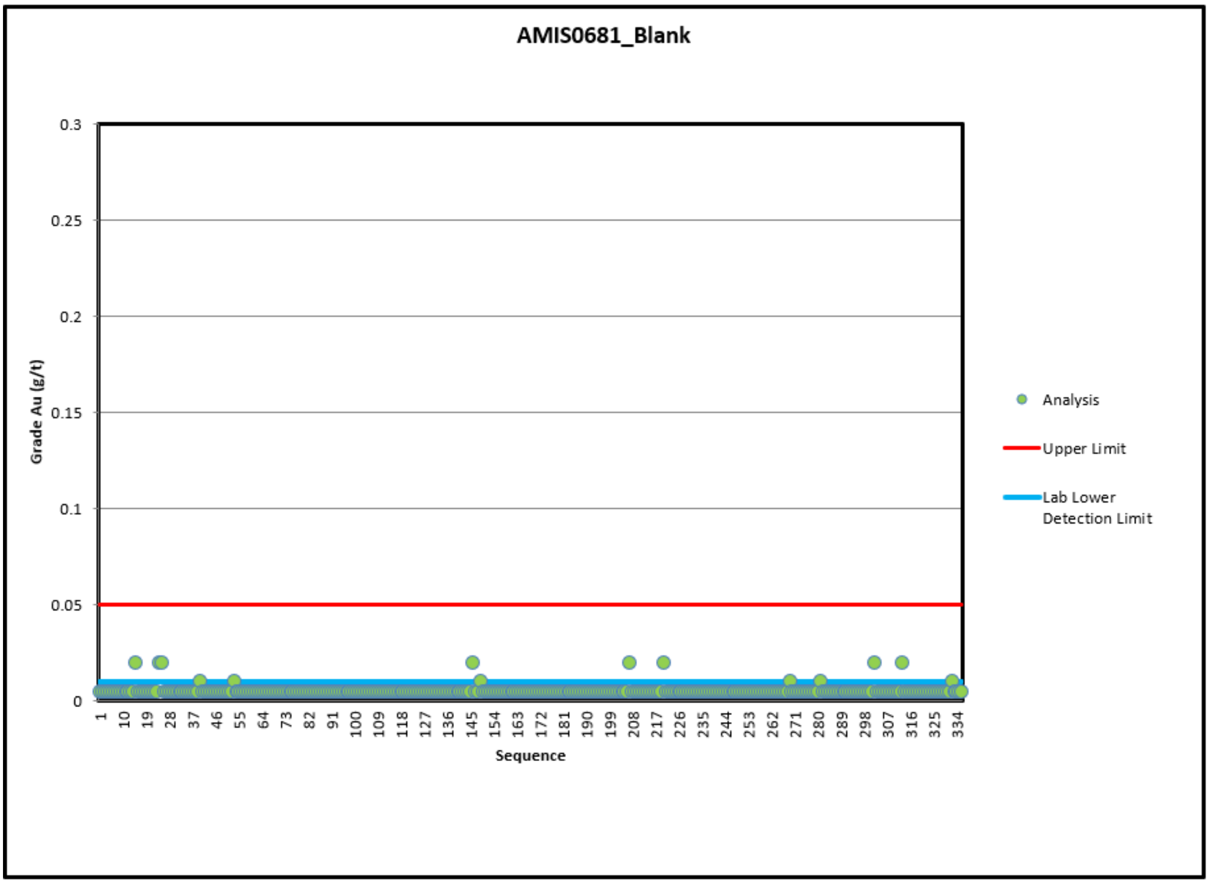


Figure 11.17: QAQC Results for AMIS0681 (Blank)

11.3.8.3 Duplicates

The SGS laboratory in Bamako was asked to regularly assay split pulp samples as duplicate samples to monitor analytical precision. Empty, labelled sample bags were inserted into the sample sequence for the laboratory to split the sample. The results of the duplicate samples are shown in Figure 11.18 and show a good correlation with a coefficient correlation (R^2) of 0.9648. Two outliers were removed from a data set of 293 duplicates.

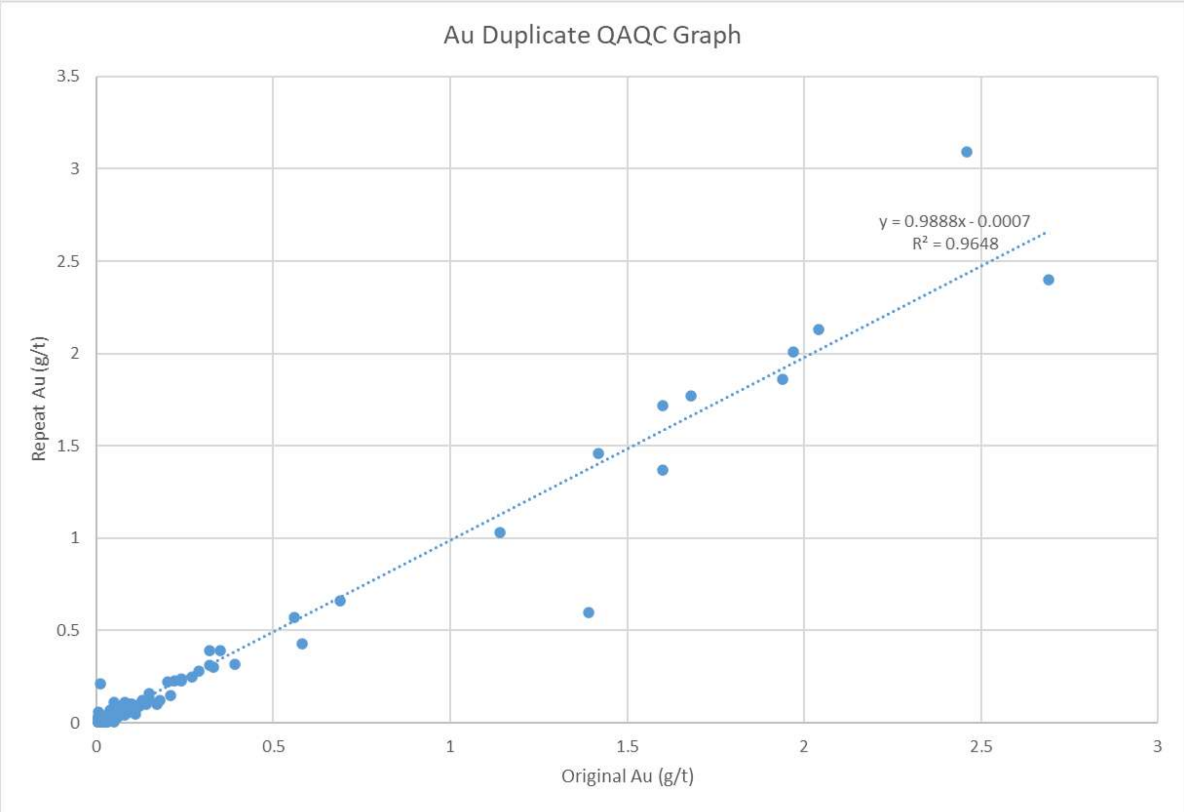


Figure 11.18: Duplicate Results

11.3.8.4 Umpire Samples

A total of 209 pulp samples were submitted as umpire samples to ALS Chemex South Africa.

The samples were fire assayed (50 g aliquot) with an AAS finish. The results of the umpire samples are shown in Figure 11.19. There was a good overall correlation with an R^2 of 0.9556. The umpire samples were a mix in order to be a representation of low-grade, medium-grade, high-grade and QAQC samples. No significant bias was observed.

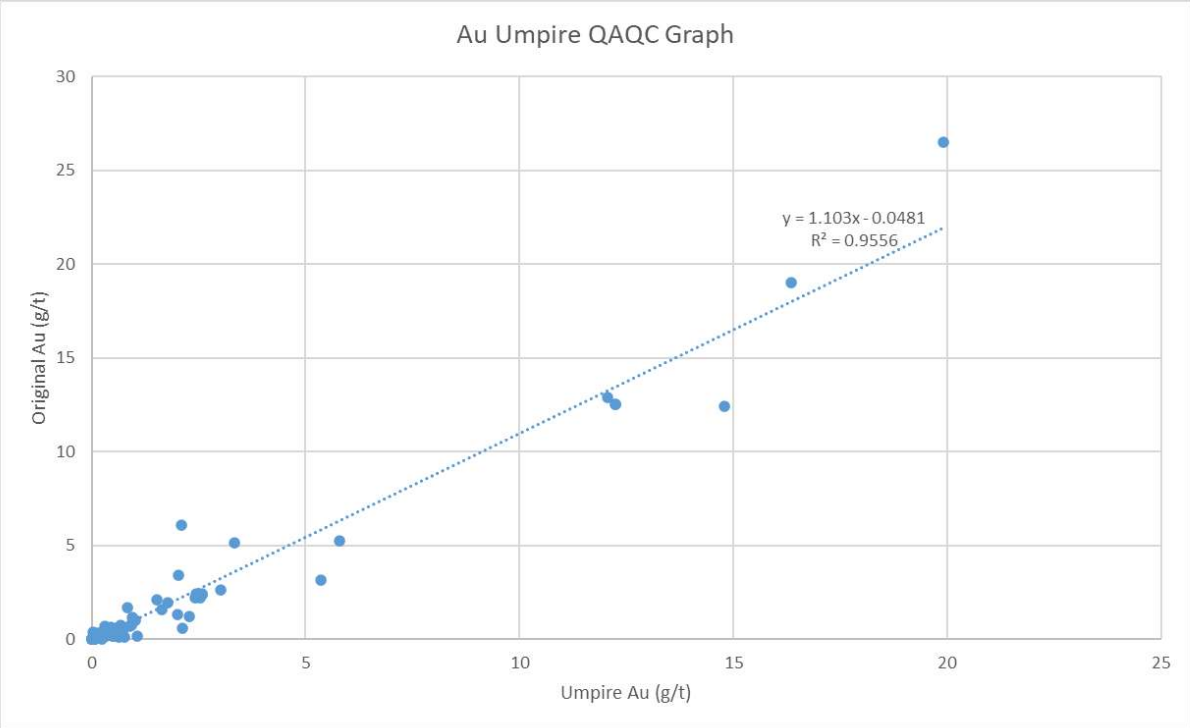


Figure 11.19: Umpire Sample Results

11.3.9 Assigning Confidence by QAQC

Assigning confidence by QAQC was as follows:

- Prior to 2009, limited or no QAQC data is available; thus, there is a low level of confidence in the data and QAQC.
- From 2009 to 2012, detailed QAQC data is available; however, a negative bias is observed in the QAQC results. Thus, these samples will be considered with a medium level of confidence. This could however be due to erroneous CRMs being utilised.
- 2007 and 2015 show good QAQC results and can be assigned a medium to high level of confidence (however, none of this raw data is available, only the results in reports).
- 2018 and 2019 shows good QAQC results and can be assigned a high level of confidence.

The years are shown in Figure 11.20, illustrating an even distribution of the high confidence with the low confidence years in terms of QAQC. The 2019 drilling covers most of the strike of the orebody. With this information available, three levels of confidence can be assigned to the QAQC data based on the corresponding year (see Figure 11.21). Where all three confidence ranges occur together, the area is considered to be a higher confidence area. This allows an area that was previously drilled and has a lower QAQC confidence range to have improved confidence due to the more recent results and QAQC data that confirms the results in the area.

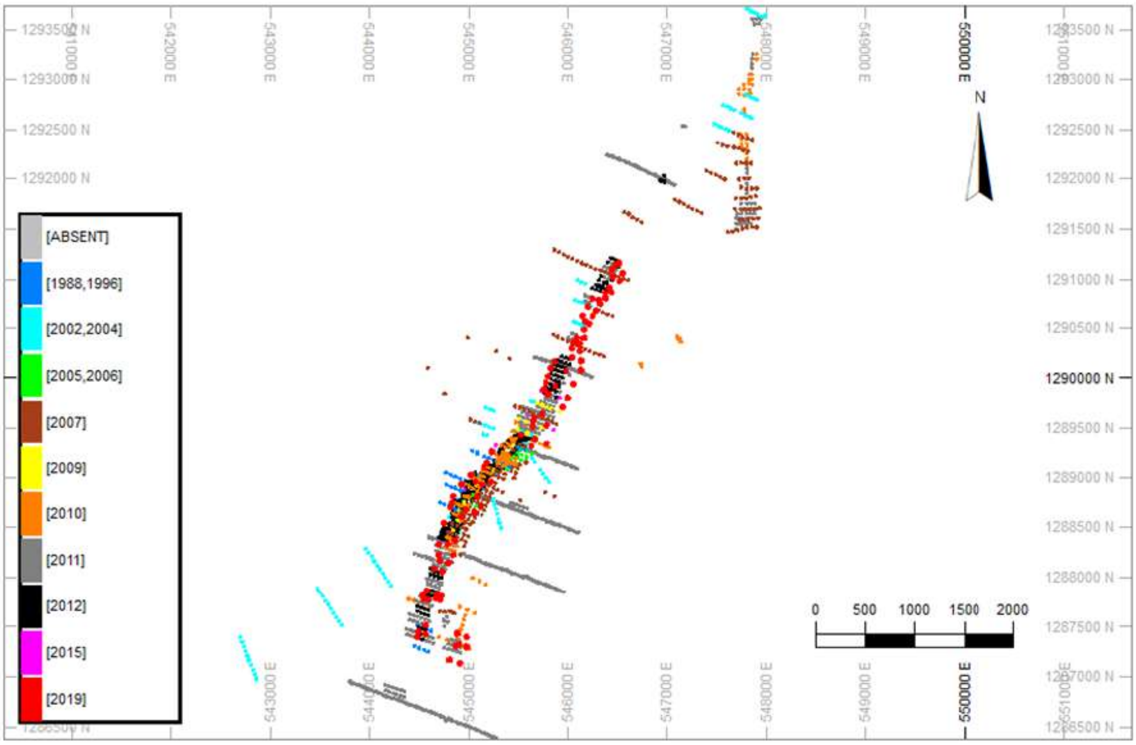
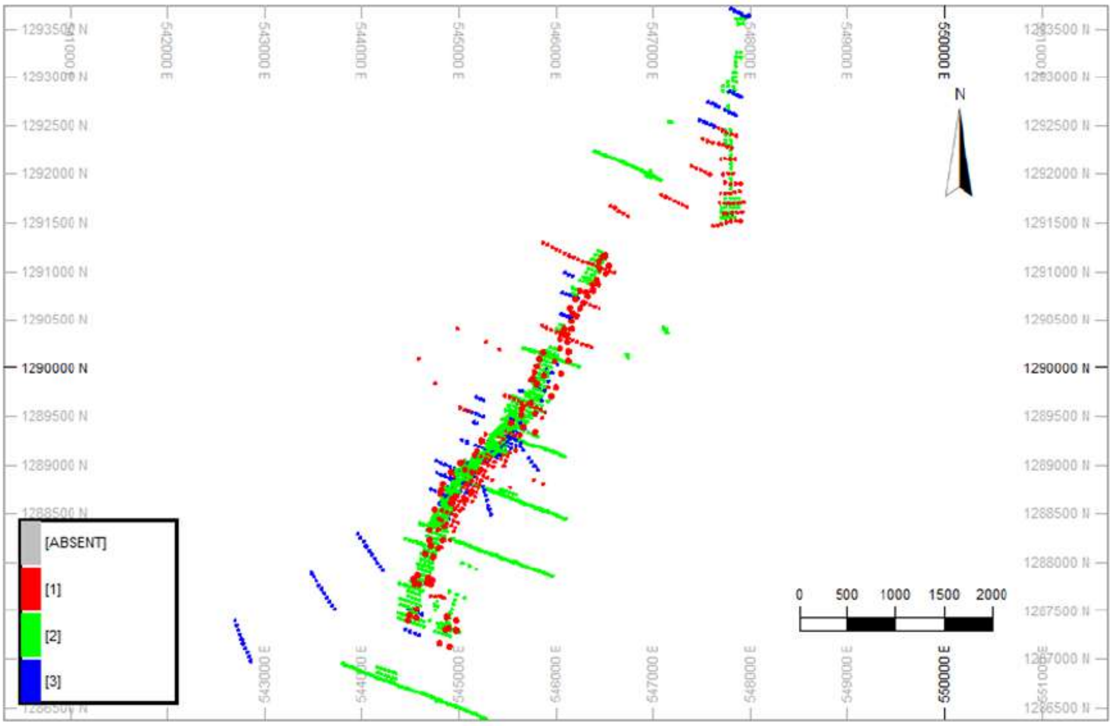


Figure 11.20: Years of Each Drillhole



NOTES:

1. Confidence level 1: higher confidence, 2007, 2015, 2018 and 2019
2. Confidence level 2: medium confidence: 2009-2012
3. Confidence level 3: low confidence: prior to 2009

Figure 11.21: QAQC Confidence by Year

To investigate this further, the data for the pre-2018 and post-2018 drilling was presented in histograms to understand the grade distribution and the mean grades and to see if there was a big variance. If the discrepancy was low, then one could assume that the average grade and grade distribution of the pre-2018 and post-2018 drilling were similar and that therefore there was no major bias. The histograms are shown in Figure 11.22 to Figure 11.25.

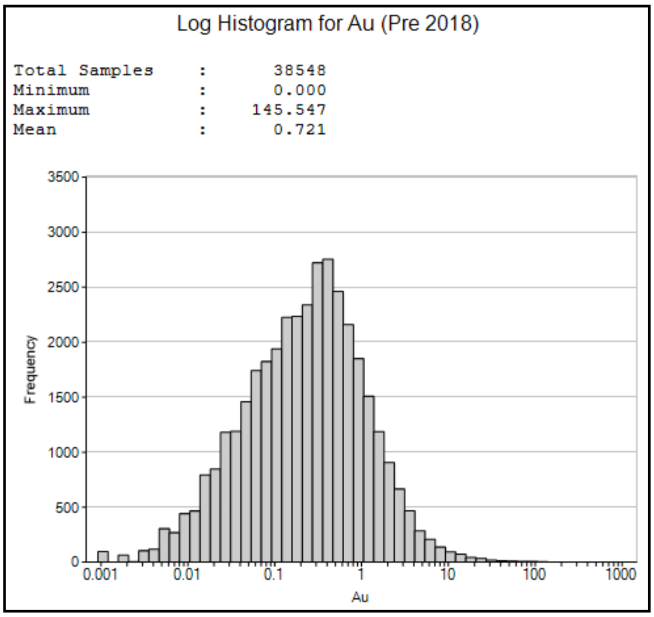


Figure 11.22: Histogram Plot of the Pre-2018 Data within the Grade Shells

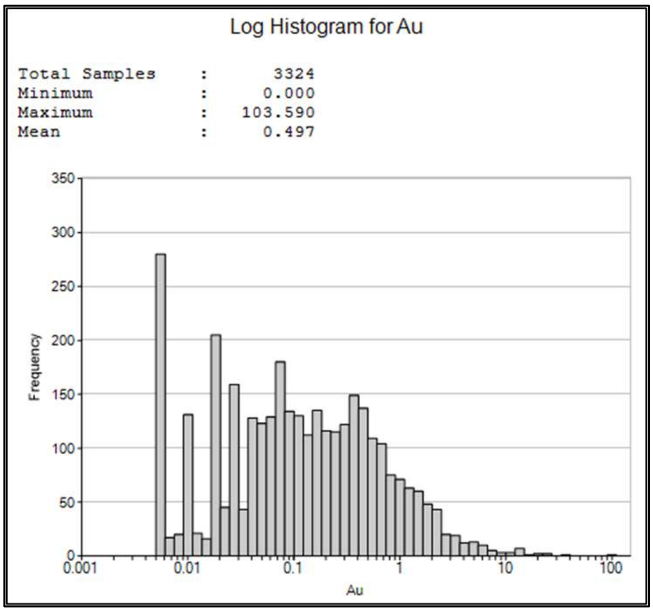


Figure 11.23: Histogram Plot of the Post-2018 Data within the Grade Shells

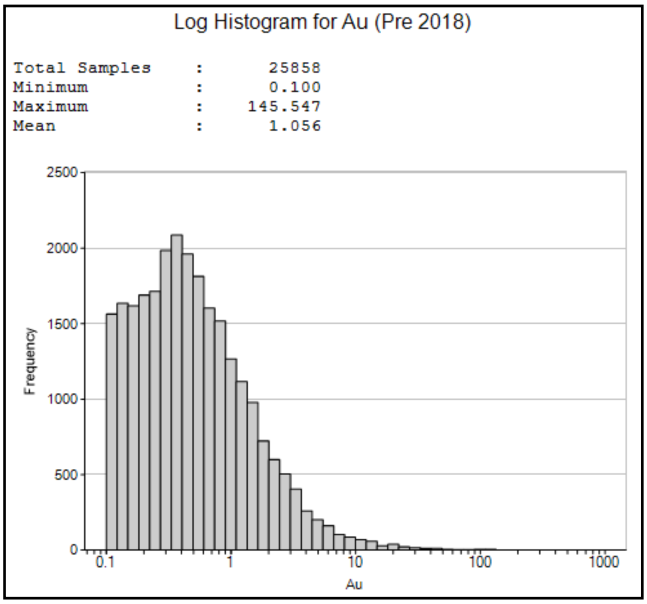


Figure 11.24: Histogram Plot of the Pre-2018 Data within the Grade Shells, > 0.1 g/t

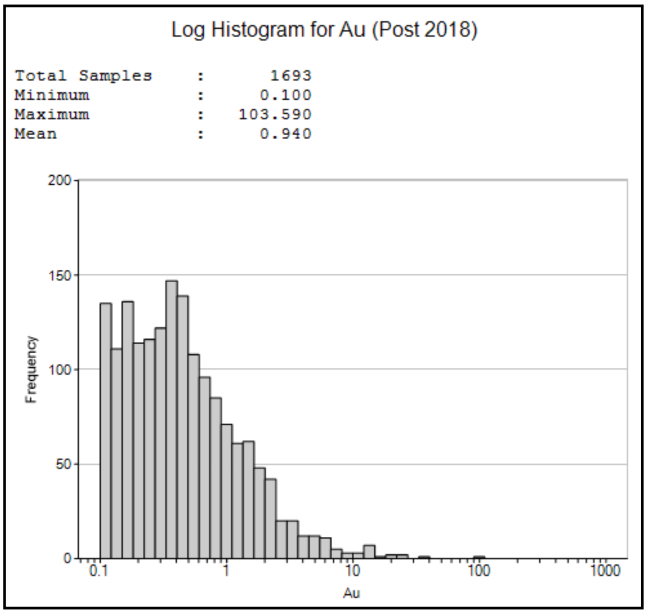


Figure 11.25: Histogram Plot of the Post-2018 Data within the Grade Shells, > 0.1 g/t

When considering the total data set within the grade shells of the geological model, the mean grades for the pre-2018 data set are higher than those for the post-2018 data set. This is due to the more negatively skewed histogram, as well as the bimodal nature, of the post-2018 drilling, including more very low-grade samples. This could be due to the fact that the majority of the pre-2018 samples are LW samples and therefore could be detecting the very low-grade mineralisation while the smaller diamond core samples are not detecting these very low levels of mineralisation because of the smaller sample size. This affects the mean grade of the data set when trying to compare the total grade. Therefore, to reduce this “noise”, the data sets were cut at 0.1 g/t to investigate the grade distribution and mean grade of the samples contributing toward the Mineral Resource since the cut-off grade is 0.35 g/t. The grade distributions are very similar, and the mean grade is within 9 %, which is within reason for this type of mineralisation and the difference in the size of the two data sets (25,858 samples vs 1,693 samples). From the above, it is concluded that there is no bias in the 2009 to 2012 database.

This suggests that the larger LW samples are better when detailed geological samples are not required. However, if LW analysis is used, the preparation of the CRMs should be by an accredited laboratory for QAQC purposes. The tails of the LW process should also be analysed by fire assay to obtain the total gold content. The 2009 to 2012 samples are LW-based content only and not total gold. If the cyanidation leach recovery in a sample is poor, then the grade could be understated. Table 11.5 shows the results of a review of the detailed assay database, which contains information from approximately 5 % of the 2009 to 2012 database, which includes SFA on the tails of the LW assay grade. In Table 11.5, the last two columns show the data for this exercise. They show the average grade of the samples for LW only (original mean) and the same samples with the SFA tails included for a total gold grade (new mean). The results show that the grade is approximately 9 % higher if all the samples, including the trace samples, are included. If the trace samples are excluded, then the percentage difference increases to 14 %. This suggests that the LW samples may be understating the sample grade as they are not total gold values.

11.3.10 Total QAQC Conclusion Pre 2020 Drilling

As can be seen in Figure 11.21, the higher confidence years and, in particular, the 2019 drilling have taken place over the entire strike of the deposit, adding greater confidence to using the database and all the samples in the estimation for Measured and Indicated resources. During the course of geological modelling, it was also observed that the new models generated from the existing data needed no major edits or changes when the new “infill/confirmatory” drilling was introduced. The 2019 drilling served to update the new revised model and add better resolution to specific areas in the model that were previously uninformed; it did not show a large difference or variation from the existing database. This also confirms the suitability of using historical data along with newer data. Figure 11.26 compares historical assay results with the 2019 Phase 1 drilling results, showing a good correlation of the 2019 Phase 1 drillholes and the closest historical drillholes. This helps to confirm that the existing database can be used with the new data. The metre-by-metre comparison with neighbouring holes does show some variance; however, the plots in Figure 11.26 serve to confirm that broader mineralised zones are common in old versus new drillholes. This was also proved as newly drilled holes were added to update the existing grade shells created only from historical holes. There were minor changes and local differences between the initial grade shells and the revised grade shells.

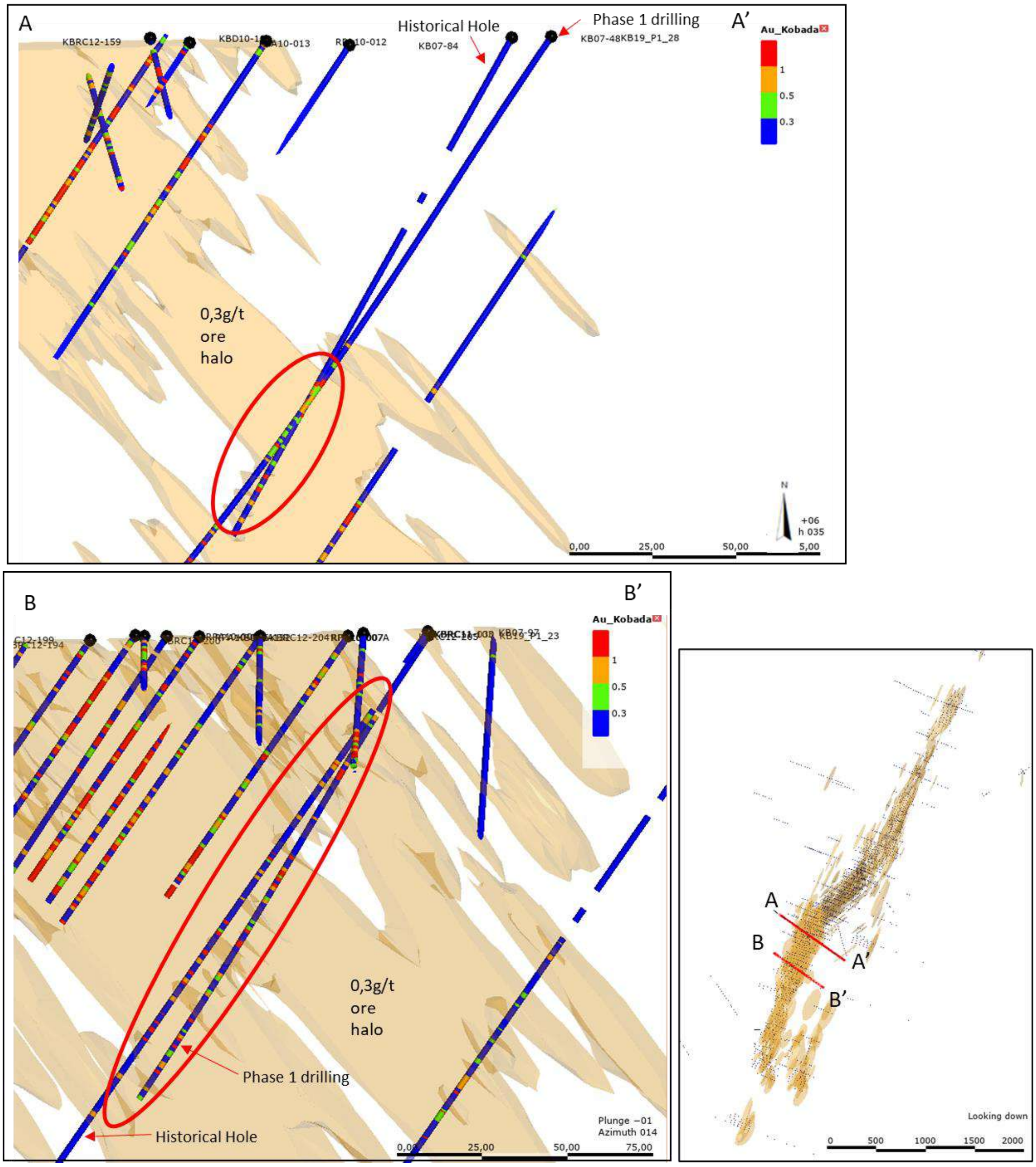


Figure 11.26: Comparison of Historical Drillholes and Nearby Phase 1 Drillholes

11.3.11 QAQC 2020

AGG implemented QAQC protocols for the 2020 drilling campaign to ensure consistent quality. The QAQC protocol that was implemented was similar to the one implemented during the 2018–2019 QAQC (Section 11.3.8). Every 10th sample was either a blank sample, a CRM of various grades, or a duplicate sample. This equates to approximately 10 % of the samples.

11.3.11.1 Standard Reference Materials

The CRMs were purchased from African Mineral Standards and included AMIS0569, AMIS0441 and AMIS0559 (see Section 11.3.8.1). The results for the CRMs are presented in Figure 11.27 to Figure 11.29.

A total of 114 AMIS0569 samples were utilised as part of the QAQC during 2020 sampling campaign. No low-grade QAQC samples failed, and all fell within the 2nd standard deviation (see Figure 11.27).

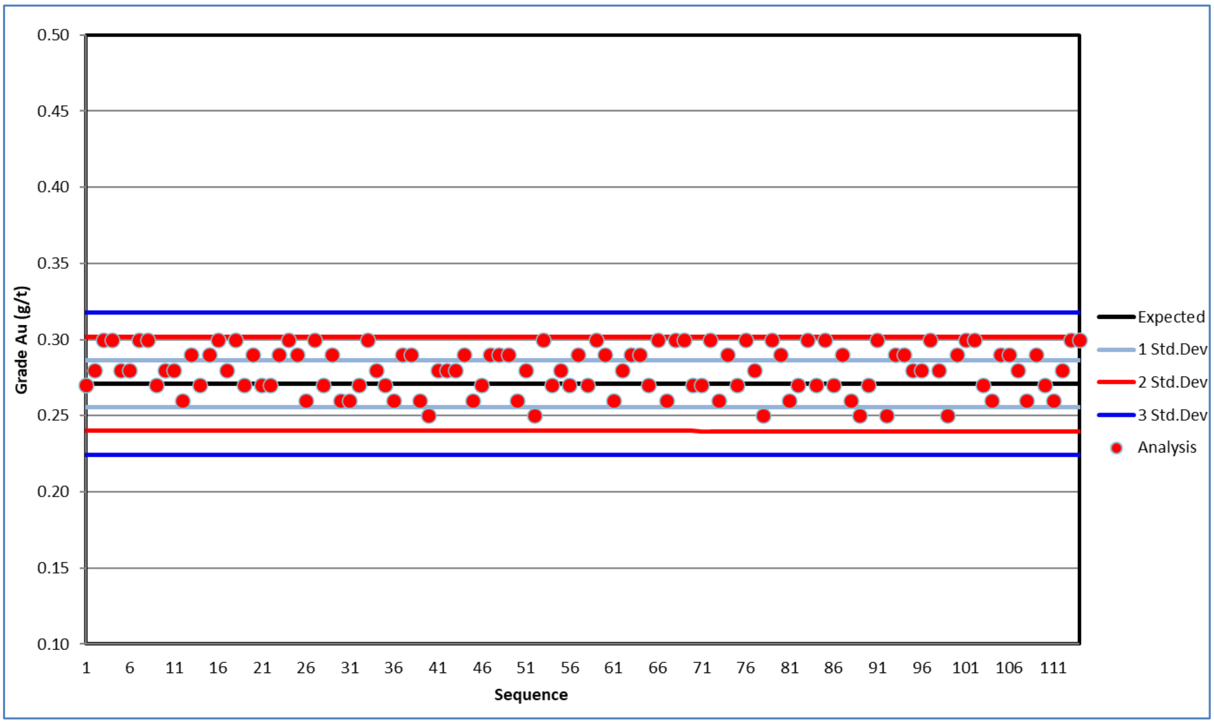


Figure 11.27: QAQC Results for AMIS0569

A total of 97 AMIS0441 samples were analysed. No medium-grade QAQC samples failed, and all the samples fell within the 2nd standard deviation (see Figure 11.28).

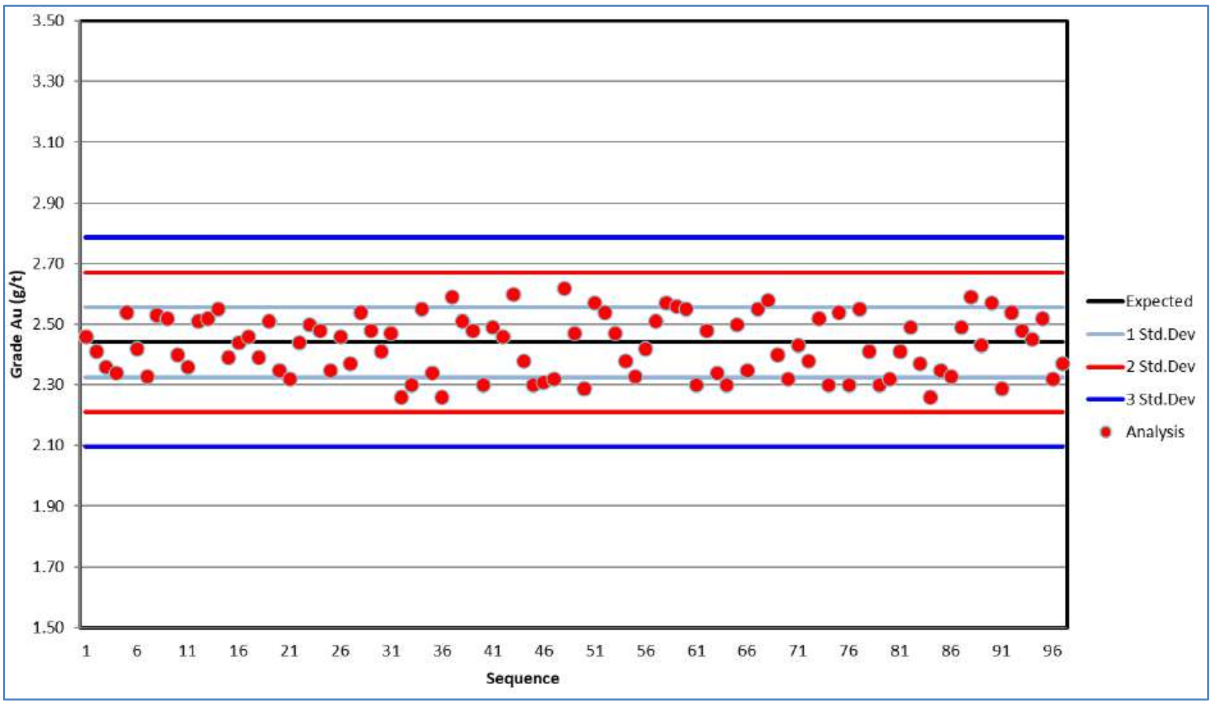


Figure 11.28: QAQC Results for AMIS0441

A total of 43 AMIS0559 samples were analysed. No high-grade QAQC samples failed, and all fell within the 2nd standard deviation (see Figure 11.29).

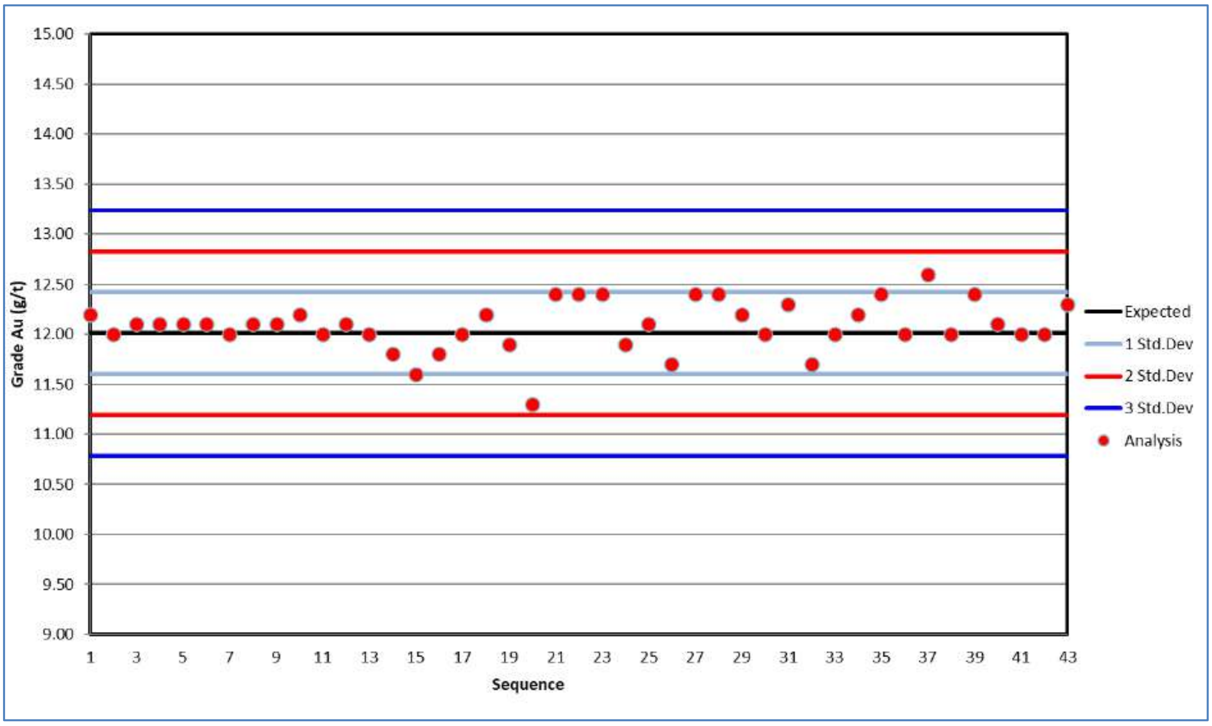


Figure 11.29: QAQC Results for AMIS0559

11.3.11.2 Blanks

Each drillhole started and ended with a blank. No blank samples failed. Two types of blanks were utilised during the 2020 sampling campaign:

- AMIS0681, purchased from African Mineral Standards (see Section 11.3.8.2)
- Pool sand, purchased from the SGS laboratory in Bamako

The expected gold grade of the blanks is 0 g/t.

The results for the AMIS0681 blank sample are shown in Figure 11.30. All the blank samples passed the blank QAQC. The upper limit for the blank samples was set at three times the detection limit (0.05 g/t).

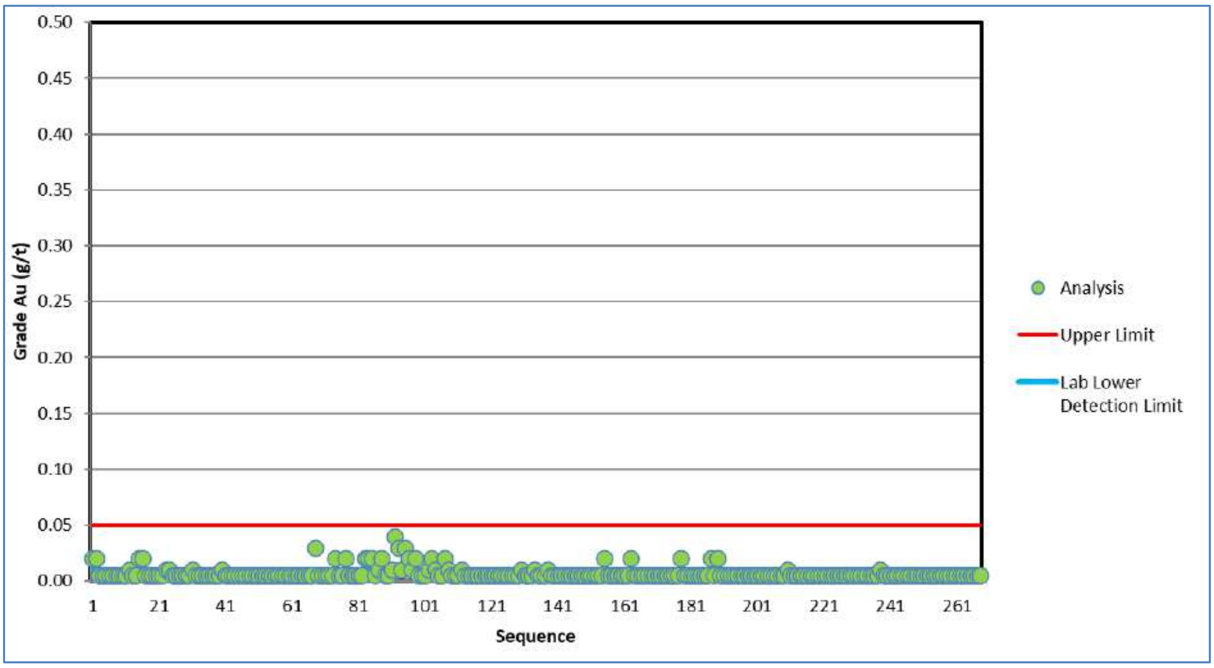


Figure 11.30: QAQC Results for AMIS0681 (Blank)

11.3.11.3 Duplicates

The SGS laboratory in Bamako was asked to regularly assay split pulp samples as duplicate samples to monitor analytical precision. Empty, labelled sample bags were inserted into the sample sequence for the laboratory to split the sample. The results of the 196 duplicate samples are shown in Figure 11.31 and show a good correlation with an R^2 of 0.9337.

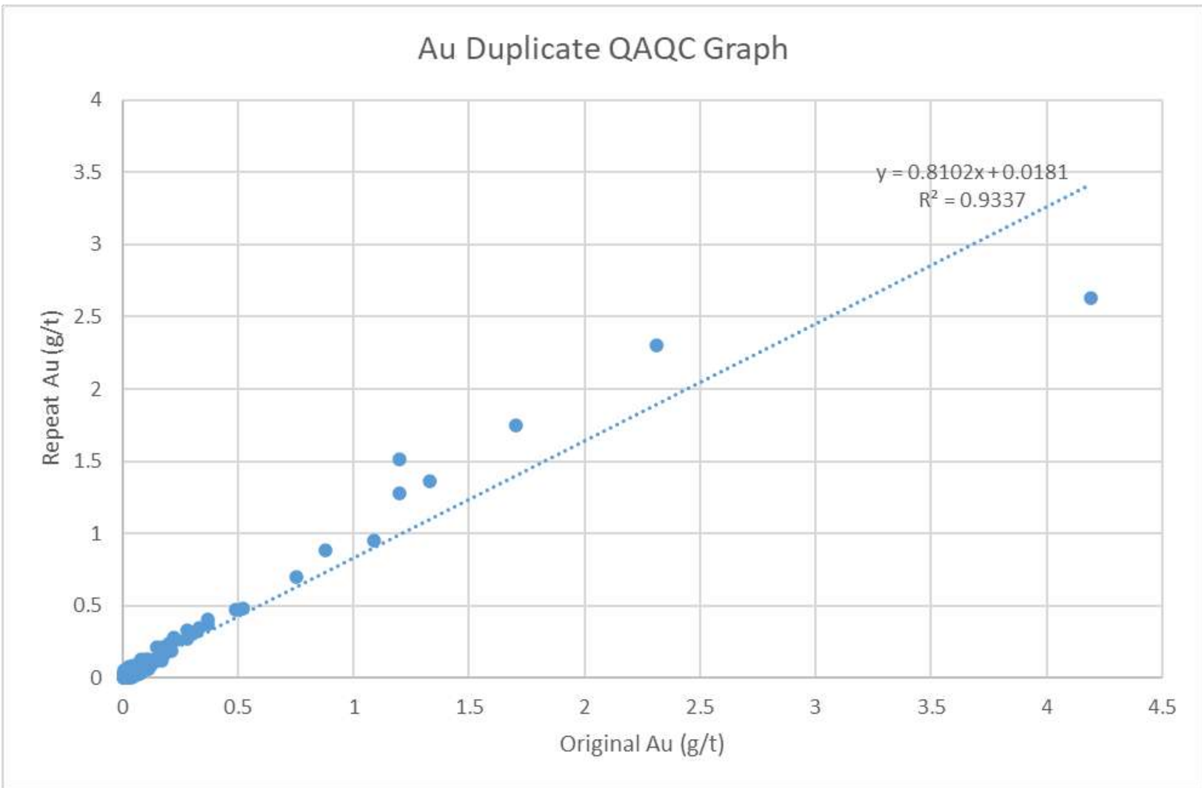


Figure 11.31: Duplicate Results

11.4 ADEQUACY OF SAMPLE PREPARATION

CRMs from 2009–2012 show a significant bias. This is believed to have been due to the sampling preparation procedure of diluting a 100 g CRM with a 3 kg blank sample to provide the diluted standard required for LW. Inaccurate weighing of the blank or CRM portions could also bias the result. However, this problem potentially affects only a limited portion of the database.

Due to the potential bias of the standard preparation for LW, it is suspected that the bias applies to the CRMs verification and *not to the general drill sample assays*.

When the sample preparation is performed at the SGS Laboratory and the subsequent analysis at the ALS laboratory, none of this bias is seen, and the CRM results are acceptable. This suggests that the error in the LW was with the preparation and inaccuracies in the weighting of these samples.

It is the QP’s opinion that the sample preparation, security, and analytical procedures of the total database are adequate and reliable enough to be utilised in the estimation of a Mineral Resource.

12 DATA VERIFICATION

12.1 DATA VERIFICATION PROCEDURES

An exercise was undertaken to define the final assay values and codes for absent values that had been assigned in the existing final database, as well as to check the previous assay entries. The final database was recompiled from original data where it was available (assay databases) and compared to merged files exported from Microsoft Access. Where samples were missing from the final database, these were corrected and added where required (six drillholes that were missing were added). The final assay results are summarised in Table 12.1.

Table 12.1: Values Utilised as the Final Au Value

Company	Year	Hole Type	Analysis	Values used	Trace Values
BRGM	1988	DD	Fire Assay	Used au_plot_ppb (FA results)	0.01
La Source	1996	RC	Fire Assay	Used au_plot_ppb (FA results)	0.0005
Cominor	2002	RC/AC	Fire Assay	Used au_plot_ppb (FA results)	0.01
Cominor	2004	RC	Fire Assay	Used au_plot_ppb (FA results)	None
AGG	2005	DD	Fire Assay and Screen Fire Assay	If SFA lab code_SFA results used (AU_ETL_MA), if FA lab code, FA results used (AU_ETL_FS), previously applied trace/1000, I reverted to original FA result	0.015
AGG	2005	DD	ICP	AU_ETL_ICP	0.005
AGG	2006	DD	Fire assay	"AU_ABI or AU_als_FA" is populated, but final AU is not, copied over to final Au	0.005
AGG	2006	DD	Screen fire assay	"pondere_g-per_t" weighted G/t result from SFA, copied over to final…only irregularly are these results included3/217? The remainder is copied over	None
AGG	2006	DD	possibly Fire assay	Half of "AU_ABI or AU_als_FA" is populated throughout, and Au _plot_pbb identical to Au_Abi, copied remainder to final Au	None
AGG	2007	DD	Fire assay	AU_ABI or AU_als_FA values used only partially; remainder included	0.005
AGG	2007	RC	Fire assay	Only final values, unclear which lab, Au_plot_ppb also populated	0.005
AGG	2007	DD	Screen fire assay	Weighted g/t values or au_plot_ppb values used (identical), copied from weighted, more decimal points. Au_ABI/ ALS_FA present but values not used	None
AGG	2007	DD	Screen fire assay	Weighted g/t values, copied from weighted, more decimal points.	None
IMG	2009	RC/DD	Leachwell	Au_ppm_AA15c used as final au value, SFA results added to LW where available, original results imported as check	0.0005
AGG	2009–2010	RC/DD	Leachwell	Au_ppm_AA15c used as final au value, SFA results added to LW where available, original results imported as check	0.0005
AGG	2011	RC	Leachwell	Au_ppm_AA15c used as final au value, SFA results added to LW where available, original results imported as check	0.0005
AGG	2012	RC	Leachwell	Au_ppm_AA15c used as final au value, SFA results added to LW where available, original results imported as check	0.0005
AGG	2012	DD	Leachwell	Au_ppm_AA15c used as final au value, SFA results added to LW where available, original results imported as check	0.0005
AGG	2015	DD	Leachwell	Au_LWL69M_ppm used as final AU	0.005
AGG	2019–2020	DD	Fire assay	Used final AU	0.005

In addition to other checks, a number of absent values/no samples (as seen from the original assays sheets) were defaulted to trace values “tr” in the original database. All of these were changed to absent “–” for consistency to facilitate further calculations.

It was noted that for absent samples, all 0 values were coded to absent and there are no true 0 (zero) values:

- Absents include an “absent”/blank result for that sample (not populated or no sample).
- Absent samples are also described as “no sample”, “empty bag”, etc.
- Everything that is truly trace is recorded and corrected to ½ the detection limit in the final database (0.0005 g/t).

After checking, less than a 0.5 % change in the grade of the total database was noted.

The current database consists of 1,609 drillholes inclusive of historic drilling. The estimation database only considers drillholes within the Project Area and was validated for the following:

- Collars located within the Project Area
- Azimuth between 0° and 360°
- Sample interval overlap
- Sample duplication
- Assay and density within acceptable ranges
- Downhole survey
- Typographical errors

A total of 280 drillholes were excluded; therefore, 1,329 drillholes were used in the generation of the deposit model and the estimation. Intercepts totalling 41,179 lie within the grade halos.

12.2 LIMITATIONS ON/FAILURE TO CONDUCT DATA VERIFICATION

Minxcon has verified the historical data were possible. The 2018 and 2019 drilling campaigns have been used to verify the previous drill data and improve the confidence in the geological model and Mineral Resource model.

12.3 ADEQUACY OF DATA

It is the QP’s view that the volume, quality, and density of the reviewed database used in the Mineral Resource estimation are adequate for the purposes of conducting a Mineral Resource estimation and for the declaration of a Mineral Resource.

13 METALLURGY

13.1 OXIDE TEST WORK

13.1.1 Introduction

In the previous feasibility study conducted in 2016, the test work conducted was to support a process flowsheet based on recovering gold through gravity means only, and other recovery options were not assessed. SENET proposed a metallurgical test work programme to support all the possible process flowsheets and to use the results to select the optimum process route.

The objectives of the DFS metallurgical test work were to

- Conduct test work to select the optimum process route for treating the Kobada ore.
- Conduct optimisation test work on the selected route to obtain the optimum parameters for maximum gold dissolution.
- Conduct variability comminution and recovery test work using the selected process route and optimum parameters established.

The test work was conducted at Maelgwyn Mineral Services South Africa. The test work was conducted on samples selected from the North, Central and South zones to cover the entire deposit.

The Kobada (measured and indicated) resource, based on a 0.35 g/t gold cut-off grade, contains 29.5 Mt of ore at a grade of 0.89 g/t and is summarised in Table 13.1. It can be noted from the table that the saprolite ore constitutes more than 80 % of the resource, and thus most of the test work was performed mainly on saprolite ore with only a small part of the samples being a mixture of the laterite and transition ore.

Table 13.1: Ore Type Abundance

Ore Type	Tonnes	Tonnes Less Geological Losses	Au	Percentage of the Resource
	Mt	Mt	g/t	%
Laterite	1.96	1.81	0.93	6
Saprolite	27.10	24.88	0.89	84
Transitional	3.07	2.82	0.87	10
TOTAL	**32.13**	**29.50**	**0.89**	**100**

13.1.2 Recommended Metallurgical Test Work

Figure 13.1 shows the recommended DFS metallurgical test work.

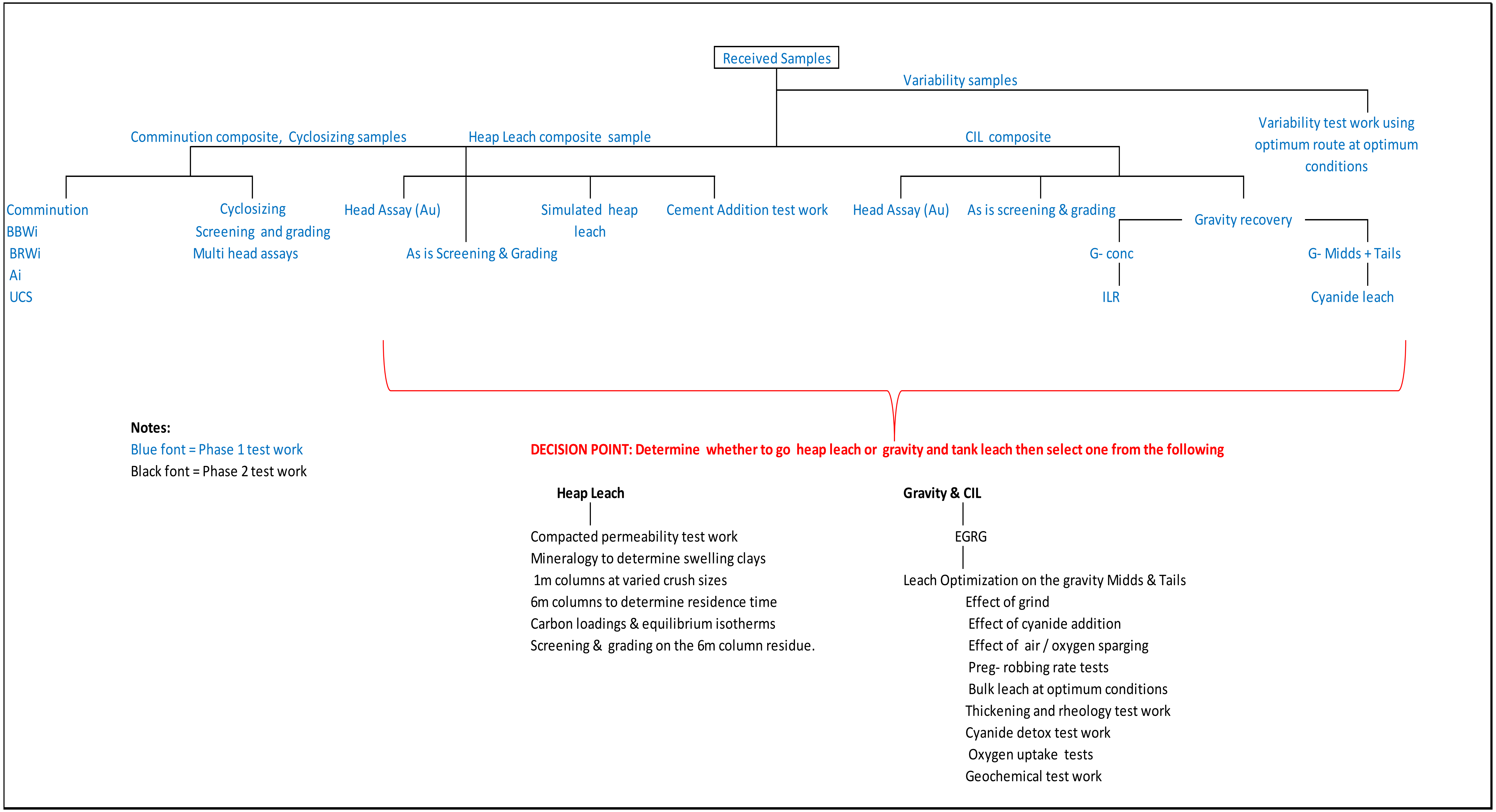


Figure 13.1: DFS Test Work Flowsheet

13.1.3 Sample Selection

Following the identification of the metallurgical test work to be conducted for the DFS, this information was submitted to Minxcon Pty Ltd (Minxcon) to select samples that were representative of the entire mineralised zone (orebody) that was intended for mining. Minxcon specified the metallurgical samples as shown in Table 13.2.

Table 13.2: Metallurgical Samples Specified for DFS

Type of Tests	Sample Requirements					
	Master Composite		Variability		Waste Ore	
	Type	Mass (kg)	Type	Mass (kg)	Type	Mass (kg)
Cyclone Sizing				10 × 10 kg Samples		
Comminution	½ or full HQ or PQ core	100	½ or full HQ or PQ core	3 × 100 kg	½ or full HQ or PQ core	100
CIL Recovery Tests	½ core HQ or PQ or Grab Samples	200	½ core HQ or PQ or Grab Samples	3 × 20 kg		
Heap Leach Tests	½ core HQ or PQ or Grab Samples	300	½ core HQ or PQ or Grab Samples			

The samples were taken from the remaining historical HQ half core that was drilled during 2005 and 2018. The half core was used for all the samples because the historical RC chips had been destroyed. The samples were taken predominantly from the saprolite, and a small portion was taken from the laterite and transition zone in order to mimic the intended mining proportions.

Minxcon selected the samples to represent the entire mineralised zone intended to be mined. This was done by taking the drillhole database and flagging the samples that fell within the Minxcon mineralised wireframes. The waste comminution test samples were taken in the lower grade areas of these wireframes to ensure that they were from the intended mineralised zone.

Thereafter, Minxcon divided the database into four geodomains based on the initial Minxcon model:

- One North (Wireframe 1)
- Two Central (Wireframes 2 and 3)
- One South (Wireframe 4)

Samples were taken from each geodomain, which ensured a good spread across the entire mineralised zone.

The locations of the individual core samples (see Figure 13.2 to Figure 13.5) show the spread of the composite samples that represent the entire mineralised zone.

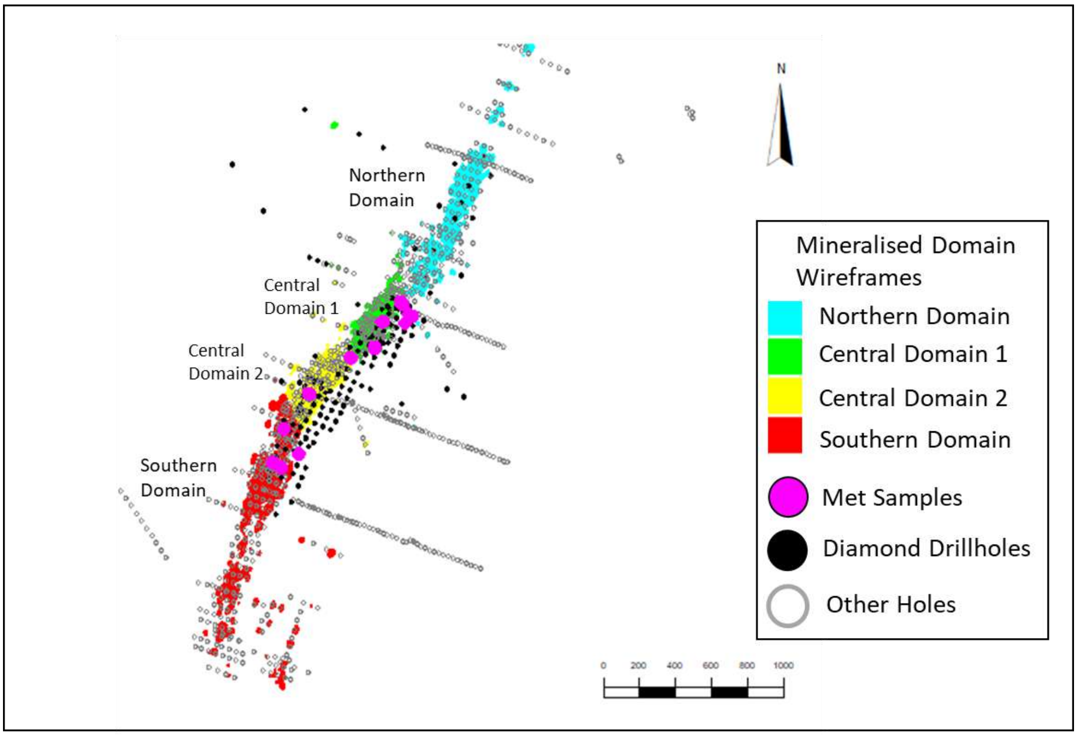


Figure 13.2: Cyclone Test Sample Positions

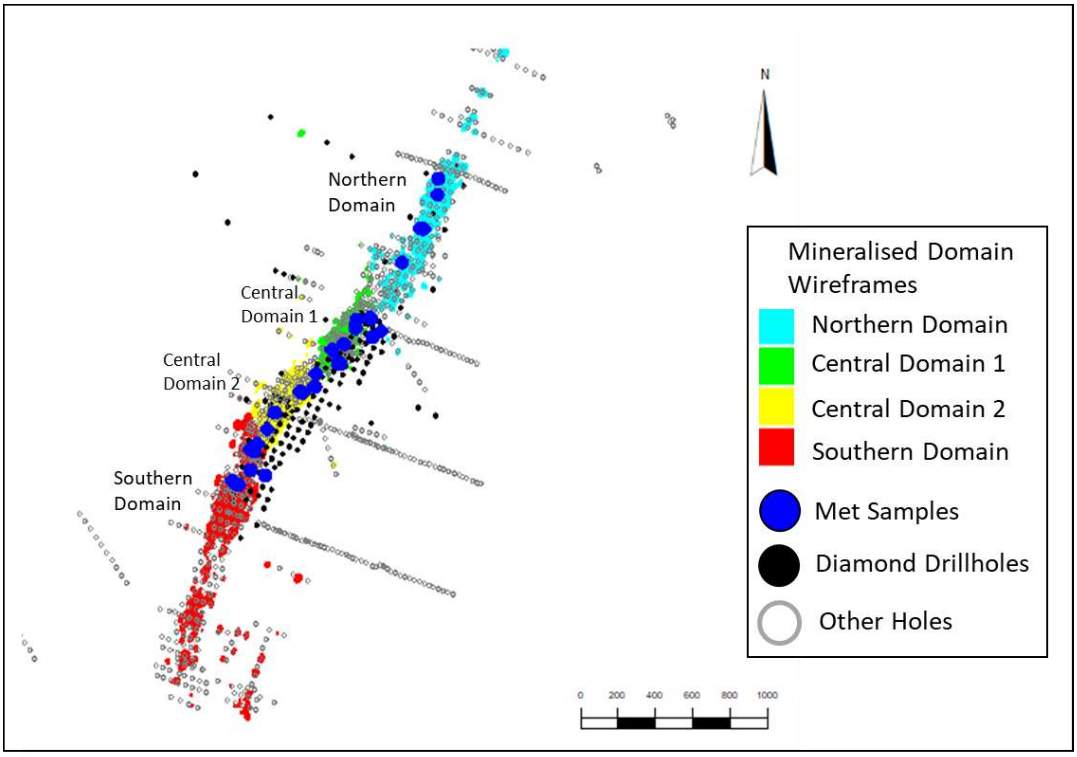


Figure 13.3: CIL Test Sample Positions

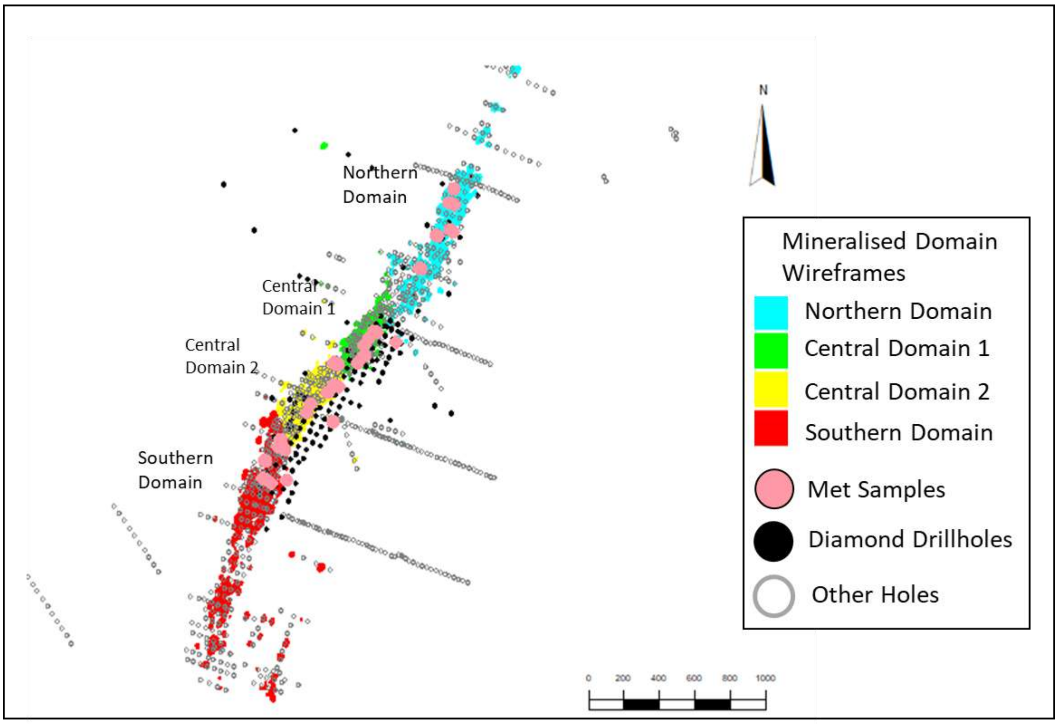


Figure 13.4: Heap Leach Test Sample Positions

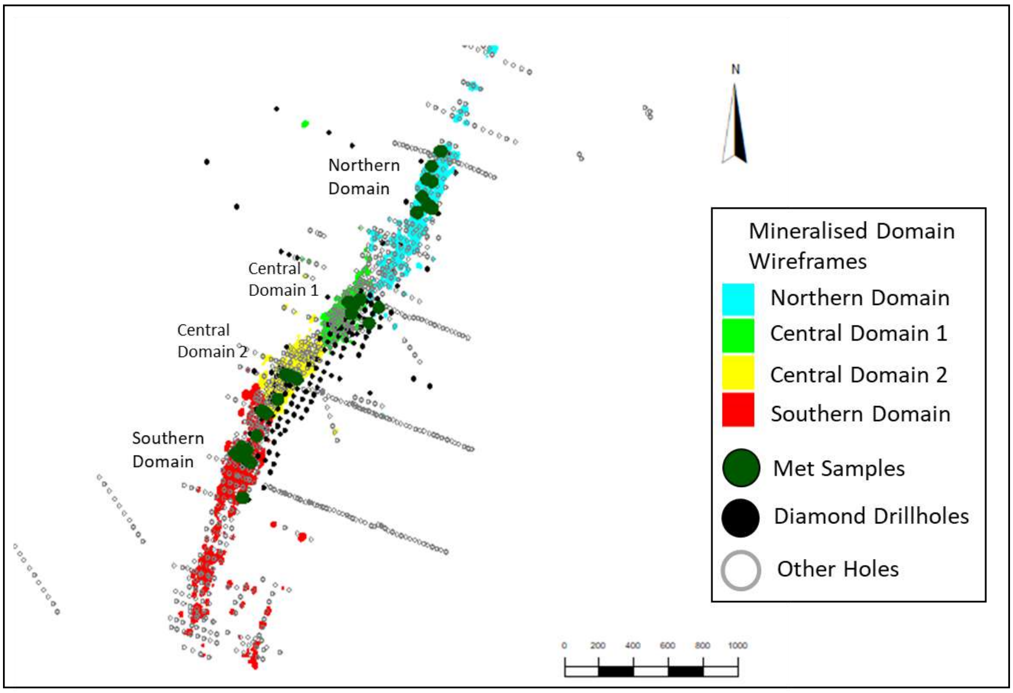


Figure 13.5: Comminution Test Sample Positions

13.1.4 Summary of DFS Results

Table 13.3 shows a summary the DFS test work result outcomes.

Table 13.3: Summary of the DFS Test Work Results

<table>
<tr><th>Test Work</th><th>Outcome</th></tr>
<tr><td>Comminution</td><td>The comminution test work indicated the following values for oxide ore:
BBWi ranged from 0.52 kWh/t to 1.20 kWh/t
BRWi ranged from 5.1 kWh/t to 6.7 kWh/t
Ai ranged from 0.087 to 0.146
CWi ranged from 4.6 kWh/t to 7.9 kWh/t
The Kobada ore has low comminution indices (Bond ball work index (BBWi), Bond rod work index (BRWi), abrasion index (Ai), and crushability work index (CWi)); therefore, the energy requirements for crushing and milling are not expected to be high. Liner wear and media wear are also not expected to be high.</td></tr>
<tr><td>Screening and Grading</td><td>Screening and grading analysis on the feed samples crushed to 1 mm showed that the −10 µm fraction by mass (% w/w) varied from 37.54 % to 73.01 % with an average of 51.85 %. Gold distribution in the −10 µm size fraction ranged from 2.14 % to 41.95 % with an average of 14.39 %. The results indicated that there was significant gold in the slimes (−10 µm) thus a gravity-only plant would not give optimum recovery.</td></tr>
<tr><td>Head Assays</td><td>Multi-head assays did not show the presence of deleterious elements except for arsenic, which ranged from 552 ppm to 2 330 ppm. Arsenic was, therefore, monitored in the test work and an arsenic balance was performed.
Copper levels were low < 150 ppm indicating that cyanide consumption is not expected to be high due to complexing with copper.
Silver levels were low at < 1 g/t. Therefore no issues due to co-loading are expected in elution.</td></tr>
<tr><td>Clay Mineralogy</td><td>Clay mineralogy on the as-received sample showed that the ore contained a high proportion of clays, and the clay fraction was made up of kaolinite. Kaolinite is not a swelling clay and should thus, in theory, not cause any excess dewatering issues.</td></tr>
<tr><td>Cyclosizing</td><td>Actual cyclosizing test work indicated that 42 % (w/w) of the feed reported to the slimes or overflow and contained approximately 28 % of the gold. This is higher than the 4 % predicted in the previous study. The recoveries predicted in the previous study will be difficult to achieve.</td></tr>
<tr><td>Gold Recovery Processes</td><td>Recovery test work comparing heap leach and agitated leach showed the following:
Heap leach is not a favourable process route due to the high cement addition requirements of > 50 kg/t and the presence of swelling clays. The following can be achieved via heap leach:
Gold recovery: 79 % (laboratory results discounted by 3 %)
Cement consumption: > 50 kg/t
Cyanide consumption: 0.26 kg/t
Gravity followed by leach can achieve gold recoveries above 90 % with low cyanide and lime consumptions. The following can be achieved via gravity and leach:
Overall gold recovery: 96 %
Lime consumption: 0.50 kg/t
Cyanide consumption: 0.55 kg/t</td></tr>
</table>

Test Work	Outcome
	Gravity followed by intense leach on the gravity concentrate and leach on the gravity middlings and tailings was selected as the gold recovery route.
Extended Gravity Recoverable Gold (EGRG)	EGRG test work was conducted, followed by modelling of the results. The EGRG results indicated a high gravity recoverable gold (GRG) of 80.2 %. However, approximately 64 % of the GRG was finer than 38 µm. This presents a challenge as the gold will be transferred to the gravity tails, i.e. the gold will not be recovered by gravity. Modelling of the EGRG test work results to predict the expected plant recovery indicated that the plant gravity recovery was 34.8 % when feeding the gravity circuit from the cyclone underflow.
Intensive cyanidation	Intensive cyanidation on the gravity concentrate showed the following: • Gold dissolution: 96.92 % • Residence time: 24 h
Leach Optimisation	Leach optimisation tests were conducted on the gravity middlings and tailings. The following optimum conditions were determined for maximum gold dissolution: • Grind to leach: 80 % passing 150 µm • Cyanide addition: 0.7 kg/t NaCN (with a cyanide consumption of 0.45 kg/t from the bulk leach) • Leach process: CIL due to the presence of preg-robbers • Air sparging: required during leach • pH: 10.5 (with a lime consumption of 0.82 kg/t from the bulk leach) • Residence time: 16 h
Bulk Leach at Optimum Conditions	A bulk leach was performed at the established optimum conditions to generate slurry for further test work. The bulk leach showed the following: • Gold dissolution: 93.94 % • Cyanide consumption: 0.45 kg/t • Lime consumption: 0.82 kg/t • Intense leach conducted on the gravity concentrate: 96.18 %
Variability Test work	Variability test work confirmed the high gold recovery and consistently low cyanide and lime consumption. The variability results showed the following: • Gold head grades ranged from 0.33 g/t Au to 3.13 g/t Au with an average of 1.07 g/t Au. • Cyanide consumption ranged from 0.19 kg/t to 0.49 kg/t in line with the optimised cyanide consumption of 0.45 kg/t • Lime consumptions ranged from 0.21 kg/t to 0.82 kg/t in line with the optimised lime consumption of 0.82 kg/t • Overall gold recovery ranged from 90.97 % to 98.39 % in line with the optimised overall recovery of 97.45 %.
Thickening and Rheology	Thickening and rheology test work conducted on the leached slurry showed that the ore did not settle easily, and the following was determined: • No flocculant polymer was able to produce clear supernatant liquor without coagulation. After coagulation with inorganic and organic coagulants, the flocculant Flomin 934VHM gave the fastest settling rates. • The optimum feed well density was 5 % w/w. • The pilot thickener achieved an underflow density of 49 %. • The solids flux rate was 0.2 t/h/m^2.

Test Work	Outcome
Oxygen Uptake	The oxygen uptake rate tests showed an oxygen demand of 0.034 mg/L/min. An oxygen demand of less than 0.15 mg/L/min indicates that only aeration will be required during leach. This outcome ties in well with the decision from the effect of air and oxygen sparging tests, which showed that air sparging was sufficient for the Kobada ore.
Cyanide Detoxification	Cyanide detoxification test work indicated the following: • Batch detoxification test work showed that the sodium metabisulphite (SMBS) process reduced the weak acid dissociable (WAD) cyanide to less than 1 ppm, but copper addition will be required. The batch tests indicated that at an SMBS addition of 2.5 equivalent, a copper addition of 50 ppm and a residence time of 60 min, the WAD cyanide was reduced from 98 ppm to low to undetectable levels. • Batch peroxide detoxification test work showed that the method was not effective on the slurry but was effective on the leach filtrate. On the leach filtrate, the peroxide method reduced the WAD cyanide from 98 ppm to low to undetectable levels in 60 min at two times the equivalent peroxide addition and a 20 ppm copper addition. Continuous cyanide detoxification on the slurry (SMBS and copper) showed that the WAD cyanide was reduced from 115 ppm to 0.004 ppm in 60 min at an SMBS addition of 1.5 equivalent and a copper addition of 25 ppm. The following is required: • 5.48 g SMBS per gram of WAD cyanide • 25 ppm to 50 ppm copper sulphate
Arsenic Balance	An arsenic balance showed that arsenic dissolved into solution and that the final arsenic level in solution after detoxification was 1.3 ppm arsenic.

Following completion of the DFS test work, design values were selected to size the process plant (see Table 13.4).

Table 13.4: Recommended Values for Design Purposes

Test Work	Unit	Saprolite	Comment
BBWi (Ore Feed) – 150 µm	kWh/t	1.09	
BBWi (Scrubber Oversize) – 212 µm	kWh/t	12.58	
BRWi	kWh/t	6.6	
Ai	g	0.1236	
CWi	kWh/t	6.55	
Tested Head Grade			
Au	g/t	1.30	85th percentile of DFS samples received
Ag	g/t	< 2	
Sb	ppm	< 5	
S^{2-}	%	< 0.01	
SG – Design		2.45	
SG – Structural Design		2.73	
Laboratory GRG	% Au	80.2	
Simulated Plant GRG – Design	% Au	34.8	Cyclone underflow

Test Work	Unit	Saprolite	Comment
Simulated Plant GRG – Maximum	% Au	42.7	Mill discharge
ILR			
Residence Time	h	24	
Au Dissolution	%	96.92	85th percentile
Leach on Gravity Middlings and Tails (Optimum conditions)			
Grind to Leach	P_{80} (µm)	150	
Leach Residence Time – Minimum	h	12	
Leach Residence Time – Design	h	16	
Oxygenation During Leach		Air sparging required	
Au Dissolution	% Au		
Oxygen Uptake Rate	mg/L/min	0.0335	
CIL Au Dissolution	% Au	91.04	
Cyanide Consumption – Design	kg/t	0.62	
Cyanide Consumption – OPEX	kg/t	0.35	
Lime Consumption – Design	kg/t	0.82	
Lime Consumption – OPEX	kg/t	0.51	
Cyanide Detoxification			
Detoxification Method (Slurry)		INCO	
WAD Cyanide after Detoxification	ppm	5	
Residence Time	min	60	
SMBS Addition	g SMBS/g WAD CN	5.48	
Thickening and Rheology			
Solids Flux Rate	$t/h/m^2$	0.2	
Flocculant Selected		Flomin 934VHM	
Underflow Solids Density	% m/m	49	
Viscosity	% solids at which ore is viscous	55	
Plastic Viscosity – Range	Pas	0.03 to 1.3	
Plastic Viscosity – Average	Pas	0.2	
Shear Stress – Range	Pa	1 to 400	
Shear Stress – Average	Pa	200	
Shear Rate – Range	S^{-1}	-	
Shear Rate – Average	S^{-1}	100	
Site Raw Water Analysis			
Cu	ppm	2.47	
As	ppm	0.44	
Chloride	ppm	1.42	
CO_3	g/L	< 0.01	
Hardness	ppm	12.20	
pH		6.36	
TDS	mg/L	16.82	
TSS	ppm	17.75	

13.1.5 Comminution

Comminution test work was conducted on the following:

- Composite ore sample
- Variability samples from the North, Central and South zones
- Waste composite to determine the impact of waste ore on the comminution circuit

Comminution test work included determination of the following:

- Bond Rod Mill Work Index (BRWi)
- Bond Ball Work Index (BBWi)
- Bond Abrasion Index (Ai)
- Bond Crushing Work Index (CWi)

The comminution test work results are summarised in Table 13.5.

Table 13.5: Comminution Results Summary

Sample	Ai (g)	BBWi (kwh/t)		BRWi (kwh/t)	CWi	
		106 µm	150 µm		kWh/t	Specific gravity (SG)
South Variability	0.1457	0.9	0.8	6.7	4.6 ± 1.6	2.19
North Variability	0.0965	1.40	0.95	6.4	4.9 ± 0.7	2.16
Comminution Composite	0.0885	0.41	0.51	5.1	4.9 ± 2.6	2.57
Central Variability	0.0871	1.18	1.20	5.6	7.9 ± 3.2	3.06
Comminution Waste	0.0955	1.55	0.65	6.8	3.2 ± 1.3	2.28

The results indicated the following:

- The BBWi values were low, indicating that the power requirements for milling will be low. Since the ore was fine, the BBWi test work was done via the Levin method.
- The BRWi values were less than 7 kWh/t, indicating that the ore is soft. Therefore, the power requirements for rod milling are not expected to be high.
- The Ai values were all less than 0.2 g, indicating that the Kobada ore is low abrasive. Therefore, liner wear and media wear are not expected to be significant.
- The CWi values were all less than 10 kWh/t, indicating that the ore is very soft. Therefore, power requirements during crushing are not expected to be significant.

13.1.6 Head Assays

Multi-head assays were conducted on the samples received, and the results are summarised in Table 13.6.

Table 13.6: Multi-Head Assay Results Summary

Samples		S2-_S	ORG C	GRAP_C	Au	Ag	As	Bi	Ca	Cu	Hg	As	Sb	SG
		%	%	%	g/t	ppm	ppm	ppm	%	ppm	ppm	ppm	ppm	
Cyclone Samples	C2C1 Head sample	0.01	< 0.05	< 0.05	1.30	< 2	1 760	< 5	0.03	169	0.08	1 610	4.2	2.83
	C2C2 Head sample	0.02	< 0.05	< 0.05	0.46	< 2	1 330	< 5	0.02	42.5	0.15	873	2.5	2.78
	C2C3 Head sample	0.01	< 0.05	< 0.05	1.49	< 2	617	< 5	0.03	33.7	0.07	552	2	2.77
	C3C1 Head sample	0.15	0.07	< 0.05	1.32	< 2	2 410	< 5	0.03	58.1	0.19	2 330	2.2	2.85
	C3C2 Head sample	0.01	< 0.05	< 0.05	1.71	< 2	1 840	< 5	0.05	49.4	0.17	1 690	4.5	2.79
	C3C3 Head sample	0.01	< 0.05	< 0.05	1.99	< 2	1 480	< 5	0.03	68.9	0.07	1 190	2.7	2.78
	South Cyc 1	0.02	< 0.05	< 0.05	1.82	< 2	624	< 5	0.04	69.9	0.09	417	1.3	2.79
	South Cyc 2	< 0.01	< 0.05	< 0.05	1.02	< 2	593	< 5	0.04	85.2	0.06	434	1.2	3.02
	South Cyc 3	< 0.01	< 0.05	< 0.05	0.18	< 2	703	< 5	0.04	69.5	0.05	430	1.8	2.86
	North Cyc 1	< 0.01	< 0.05	< 0.05	1.79	< 2	319	< 5	0.09	116	0.4	208	1.3	2.55
	North Cyc 2	< 0.01	< 0.05	< 0.05	0.67	< 2	878	< 5	0.05	75	0.06	633	1.5	2.83
	North Cyc 3	< 0.01	< 0.05	< 0.05	0.76	< 2	542	< 5	0.06	115	0.15	396	1.3	2.89
Composite Samples	CIL Composite	< 0.01	< 0.05	0.07	9.45	< 2	1 370	< 5	0.02	64.3	0.1	1 550	3.8	2.82
	Heap Leach Composite	< 0.01	< 0.05	0.07	0.37	< 2	936	< 5	0.03	59.9	0.1	875	2.2	2.83
Initial Variability Samples	North Variability	< 0.01	< 0.05	0.08	0.44	< 2	753	< 5	0.02	62.1	0.22	827	4.6	2.82
	South Variability	< 0.01	< 0.05	0.08	0.36	< 2	877	< 5	0.04	51.2	0.04	1 010	2.8	2.84
	Central Variability	< 0.01	< 0.05	0.07	0.38	< 2	856	< 5	0.02	47.4	0.14	1 070	4	2.81
Additional Variability Samples	North Com Var	< 0.01	< 0.05	0.06	0.77	< 2	284	< 5	0.02	51.7	0.26	241	5.6	2.78
	South Com Var	< 0.01	< 0.05	0.06	0.85	< 2	859	< 5	0.08	72.8	0.32	772	9.3	2.88
	Central Com Var	0.01	< 0.05	0.06	1.09	< 2	1 080	< 5	0.04	60	0.24	1 070	10.2	2.84
	Waste	0.01	< 0.05	0.06	0.55	< 2	1 220	< 5	0.04	83.7	0.22	1 230	6.8	2.86
Analysis	Maximum	-	-	0.08	9.45	-	2 410	-	0.09	169.00	0.40	2 330.00	10.20	3.02
	Minimum	-	-	0.06	0.18	-	284	-	0.02	33.70	0.04	208.00	1.20	2.55
	Average	-	-	0.07	1.37	-	1 016	-	0.04	71.68	0.15	924.19	3.61	2.82
	85th percentile	-	-	0.08	1.79	-	1 480	-	0.05	85.20	0.24	1 550.00	5.60	2.86

The results indicated that

- The samples had a low sulphide content (< 0.02 % S^{2-}), which is expected since the ore is saprolite.
- Gold head grades ranged from 0.18 g/t Au to 9.45 g/t Au.
- The arsenic levels ranged from 284 ppm to 2 410 ppm. Arsenic will, therefore, need to be monitored in the final tails to ensure that it is within the acceptable environmental limits.
- The copper levels were low and varied from 34 ppm to 169 ppm. Therefore, cyanide consumptions, due to complexing with copper, are expected to be low.
- Antimony levels were low, ranging from 208 ppm to 2 330 ppm. Therefore, the ore is expected to be non-refractory.

As a result of the high arsenic levels, an arsenic balance was included in the test work programme to determine the distribution of arsenic in the products from the recovery process.

13.1.7 Mineralogy

In the 2016 feasibility study, SGS Lakefield (SGS) conducted a gold deportment study on an oxide sample from the Kobada deposit. The study identified 26 gold grains in the sample submitted, with over 97 % found as liberated grains with an average size of 12 µm. The largest gold grains observed were approximately 30 µm. Native gold was the only gold mineral identified.

13.1.8 Cyclosizing

The design of the 2016 feasibility study was based on the concept that the Kobada ore can be pre-treated by desliming with the slimes being rejected to the tails because they contain relatively insignificant gold (< 6 %) at a high mass rejection. Cyclosizing test work was conducted to prove/disprove that desliming the ore will result in a high mass rejection with less than 6 % gold loss. The results from this test work are summarised in Table 13.7.

Table 13.7: Cyclosizing Summary Results

Parameter	Overall Average
% Passing 75 µm (% w/w)	94 %
% Passing 10 µm (% w/w)	59 %
Overflow mass rejection (% w/w)	42 %
% Gold lost to overflow	12 %
Underflow % passing 75 µm (% w/w)	68 %
Underflow mass (% w/w)	58 %
% Gold distribution to the underflow	72 %

The DFS test work results indicated the following:

- The slimes mass rejection averaged 42 % against the previous study of more than 70 %, so any downstream treatment facility would have to be bigger than the one in the previous study.
- The gold losses associated with the rejected slimes averaged 12 % against less than 6 % in the previous study. These losses varied from 4 % to 41 %, thus rendering the desliming process a high risk due to the loss of gold to the tailings dam.

13.1.9 Gold Recovery Options

The test work and flowsheet design of the 2016 feasibility study, which was conducted by Gekko, were biased towards a gravity-only process. In the DFS, SENET proposed that several recovery methods be investigated, followed by a trade-off study to determine the optimal process route for treating the Kobada ore. Metallurgical test work to investigate recoveries using other possible process routes was conducted. The following recovery methods were investigated:

- Gravity only
- Heap leach
- Gravity followed by cyanidation of gravity middlings and tails

From the test work results, a trade-off study was conducted to compare the process options.

Four process options were investigated, with each option treating 1.6 Mt/a ROM feed. The ROM feed selection was based on the annual throughput in the 2016 feasibility study. Depending on the process route selected, different tonnages are processed in either leach or gravity.

The four process options that were assessed are as follows:

- **Option 1 – Gravity Only**
  This option treats 1.6 Mt/a ROM feed through crushing, scrubbing and desliming, resulting in less material being treated through gravity. For this option, SENET considered the mass rejection that was suggested by Gekko (high slimes rejection) and also what was determined through the DFS test work (low slimes rejection).

- **Option 2 – 0.5 Mt/a Carbon-In-Leach (CIL)**
  This option treats 1.6 Mt/a ROM feed through crushing, scrubbing and desliming, resulting in less material (approximately 0.5 Mt/a) being treated through gravity and CIL. For this option, SENET considered the mass rejection that was suggested by Gekko (high slimes rejection).

- **Option 3 – 1.6 Mt/a Gravity and CIL (Whole Ore)**
  This is a conventional cyanide leach process route, where the full 1.6 Mt/a ROM ore undergoes the crushing, milling, gravity, CIL and adsorption, desorption and recovery (ADR) processes. The desliming process is not part of this option, hence, this option has no rejection of fines.

- **Option 4 – 1.6 Mt/a Heap Leach (HL)**
  This is a conventional cyanide heap leach process route, where the full 1.6 Mt/a ROM ore undergoes the crush, agglomeration, cyanide heap leach and ADR processes. This option, as with Option 3, has no rejection of fines.

The overall gold recovery and reagents consumptions summarised in Table 13.8 were determined from the initial tests, and these were used in the trade-off study.

Table 13.8: Recovery Options Test Work Results Summary

Parameter	Unit	Gravity Only	0.5 Mt/a CIL	1.6 Mt/a CIL	1.6 Mt/a HL
Overall Gold Recovery	%	77.0	86.3	96.0	79.3
Cyanide Consumption	kg/t	0.10	0.253	0.55	0.26
Lime Consumption	kg/t	0.78	0.78	0.5	
Cement Consumption	kg/t				50

Capital and operating costs for each scenario were determined, and a high-level economic analysis was conducted using the 2016 mining plan. The results of the economic analysis are summarised in Table 13.9.

Table 13.9: Summary of Economic Analysis

Description	Unit	Gravity Only	0.5 Mt/a CIL	1.6 Mt/a CIL	1.6 Mt/a HL
LOM Tonnage Ore Processed	t (000)	12 735	12 735	12 735	12 735
LOM Feed Grade Processed	g/t	1.14	1.14	1.14	1.14
LOM Gold Recovery[a]	%	77.0	86.3	96.0	79.2
LOM Gold Production	oz (000)	393.5	441.3	490.8	404.9
Gold Price	US$/oz	1 450	1 450	1 450	1 450
Revenue	US$ million	571	640	712	587
LOM Operating Costs	US$/oz	680	651	601	785
Total Capital Costs	US$/oz	242	229	219	247
Total LOM Production Costs	US$/oz	922	881	820	1 033
Pre-Tax Net Present Value (NPV)	US$ million	141	173	206	50
Internal rate of return (IRR)	%	34.8	38.2	40.7	18.4
Discounted Payback Period	Years	3.39	3.17	3.04	6.04
Project Net Cash Flow Pre-Tax	US$ million	209.8	253.2	298.6	87.3

[a] Results from MMS Test Work
The yellow shading highlights the critical numbers to look at.

From the high-level economic analysis, it was established that the gravity followed by leaching of the gravity middlings and tailings (CIL) option (equivalent to Option 3 in the trade-off study) was the most economically viable option, and as such it was selected as the preferred processing route.

Consequently, optimisation test work was conducted on the CIL recovery option.

13.1.10 Gravity Recoverable Gold and Intensive Cyanidation on the Gravity Concentrate

Gravity laboratory tests conducted at Maelgwyn Mineral Services Laboratory indicated moderate to high gravity recoverable gold (GRG) results. It was against this background that caution was taken in the interpretation of the GRG results as it is a well understood concept that laboratory Knelson recovers gold very efficiently and to higher mass pulls than full-scale installed gravity centrifugal units, whose actual plant performance is expected to be inferior to the laboratory determined GRG. An extended gravity recoverable gold (EGRG) test was, therefore, performed at varied grinds using the Gravity Concentrators Africa (GCA) method, and a simulation of the results using the KCMOD*Pro model was performed to predict circuit recovery. The results are summarised in Table 13.10.

Table 13.10: EGRG Test Work Results

Grind Size P_{80} (µm)	Product	Mass (g)	Mass (%)	Assay Au (g/t)	Units Au	Distribution (%)	Cumulative Distribution (%)
850	Stage 1 Concentrate	197.15	1.0	238.03	46.93	74.1	74.1
212	Stage 2 Concentrate	96.23	0.5	33.79	3.25	5.1	79.2
75	Stage 3 Concentrate	144.32	0.7	4.35	0.63	1.0	80.2
	Final Tails	19 562.3	97.8	0.64	12.54	19.8	100.0
	Total (Feed)	**20 000.0**	**100.0**	**3.17**	**63.35**	**100.0**	
	Knelson Concentrator	**437.70**		**116.08**	**50.81**	**80.2**	

The EGRG test work showed a GRG value of 80.2 %, but the majority of the GRG was less than 38 µm in size. This fine-sized GRG presents a challenge in gravity recovery as most gravity units cannot efficiently recover ultrafine particles. Peacocke & Simpson was requested to run a simulation in order to predict the expected gravity gold plant recoveries.

The model predicted a GRG recovery of 34.8 % when treating the bleed of the cyclone underflow. The GRG recovery when treating the mill discharge was also modelled, and this gave a slightly higher recovery at 42.7 %. In addition, various grind sizes were modelled as shown in Table 13.11.

Table 13.11: GRG Simulation Results

Grind P_{80} (µm)	Cyclone Underflow GRG Recovery (%)	Mill Discharge GRG Recovery (%)
150 (base case)	34.8	42.7
125	36.3	43.7
100	38.0	44.8
75	42.7	48.5

The gravity concentrates produced from the variability samples and composite sample were subjected to intensive cyanidation to assess the leach kinetics of the concentrates. Test work results indicated that intense cyanidation achieved 96.92 % gold dissolution in 24 h.

13.1.10.1 CIL Gold Extraction Optimisation Test Work

Cyanidation tests were conducted on the gravity middlings and tailings samples with the aim of optimising the grind, leach time and cyanide addition, and determining the effect of preg-robbing.

13.1.10.2 Effect of Grind

Leach tests were performed on representative subsamples milled to the following grinds: 80 % passing 212 µm, 150 µm, 106 µm and 75 µm. Table 13.12 shows a summary of the effect of grind results.

Table 13.12: Effect of Grind Results

Grind	Reagent Consumption (kg/t)		Residue Assay	Gold Dissolution Based on Calculated Head Solids (%)
	Cyanide	Lime	g/t Au	
80 % passing 212 µm	0.62	0.57	0.10	88.10
80 % passing 150 µm	0.61	0.59	0.08	91.77
80 % passing 106 µm	0.58	0.57	0.08	91.60
80 % passing 75 µm	0.60	0.55	0.08	91.77

Milling finer than 80 % passing 150 µm did not significantly improve gold dissolution. Therefore, an optimum grind of 80 % passing 150 µm was selected.

13.1.10.3 Effect of Cyanide Addition

Leach tests were performed on representative subsamples milled to 80 % passing 150 µm. The leach tests were performed at the following cyanide additions: 0.1 kg/t, 0.3 kg/t, 0.5 kg/t, 0.7 kg/t, 1.0 kg/t and 2.0 kg/t. Table 13.13 shows a summary of the cyanide addition results.

Table 13.13: Effect of Cyanide Addition Results

Cyanide Addition	Reagent Consumption (kg/t)		Residue Assay	Gold Dissolution Based on Calculated Head Solids (%)
	Cyanide	Lime	g/t Au	
0.1 kg/t	0.10	0.78	0.48	48.05
0.3 kg/t	0.30	0.79	0.27	71.15
0.5 kg/t	0.35	0.78	0.15	83.82
0.7 kg/t	0.38	0.78	0.08	91.05
1.0 kg/t	0.43	0.79	0.08	91.74
2.0 kg/t	0.54	0.78	0.08	91.41

Gold dissolution did not significantly increase by adding more cyanide than 0.7 kg/t, which was, therefore, selected as the optimum cyanide addition. The corresponding cyanide consumption was low at 0.38 kg/t.

13.1.10.4 Effect of Air or Oxygen Sparging

Tests were conducted to determine if air or oxygen sparging was required during leach. The tests were conducted as follows:

- Leach tests without air or oxygen sparging
- Leach tests with air sparging only
- Leach tests with oxygen sparging only

Figure 13.6 shows a summary of the results, which indicates that air sparging improves the leach kinetics and will be required during leach.

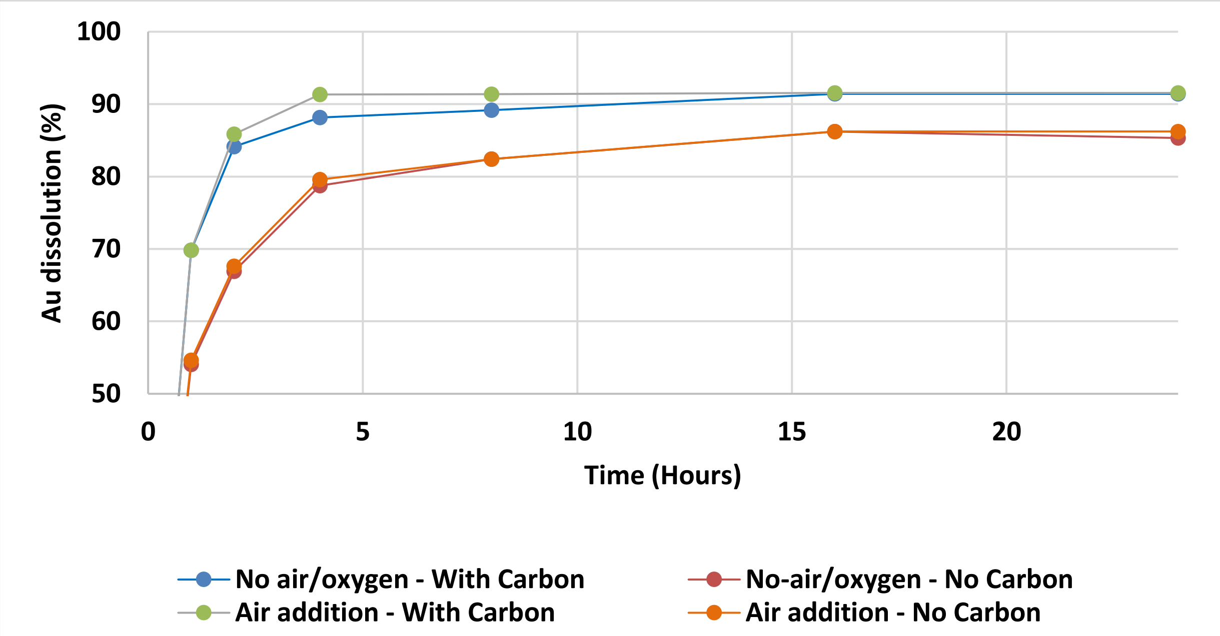


Figure 13.6: Effect of Air and Oxygen Sparging

13.1.10.5 Effect of Preg-Robbing and Residence Time (Preg-Robbing Rate Tests)

Table 13.14 shows a summary of the preg-robbing rate tests.

Table 13.14: Preg-Robbing Rate Tests

Residence Time (h)	Cyanide Consumption (kg/t)		Lime Consumption (kg/t)		Gold Dissolution (% Au)		% Preg-Robbing
	With Carbon	No Carbon	With Carbon	No Carbon	With Carbon	No Carbon	
1	0.02	0.02	0.78	0.78	69.81	54.07	15.74
2	0.02	0.02	0.78	0.78	84.14	66.93	17.21
4	0.06	0.07	0.78	0.78	88.14	78.72	9.42
8	0.18	0.18	0.78	0.78	89.15	82.39	6.75
16	0.33	0.34	0.78	0.78	91.42	86.23	5.19
24	0.36	0.36	0.78	0.78	91.42	85.32	6.10
32	0.37	0.37	0.78	0.78	91.42	86.23	5.19
36	0.38	0.38	0.78	0.78	91.42	87.15	4.27
48	0.39	0.39	0.78	0.78	91.42	86.23	5.19

The results indicated that the preg-robbers were active within the first 2 h of the leach with approximately 15.74 % gold lost to the preg-robbers within the first hour. This implies that there cannot be preleaching ahead of the CIL circuit due to the possibility of losing gold to the preg-robbers. Lime for pH control will, therefore, be added ahead of milling since there is no preconditioning tank.

From the rate tests, the optimum residence time of 12 h to 16 h with carbon addition was selected for the design.

The following optimum parameters were established for maximum gold dissolution:

- Grind: 80 % passing 150 µm
- Cyanide addition: 0.7 kg/t
- pH: 10.5
- Sparging: Air sparging required
- Leach process: CIL due to presence of preg-robbers
- Leach residence time: 24 h (laboratory), 16 h (design) and 12 h (minimum)

13.1.11 Bulk Gravity and Leach Test

A bulk leach was conducted at the optimum conditions established as described above. The bulk slurry produced was used for further test work, namely

- Thickening and rheology
- Cyanide detoxification
- Geochemical test work
- Arsenic balance test work

Table 13.15 shows a summary of the bulk leach results.

Table 13.15: Bulk Leach Test Work Results

Leach Time (h)	Reagent Addition		Pregnant Solution					Gravity Middlings and Tailings		Residue	Carbon	Reagent Consumption		Dissolution (Assay)		Dissolution (Calculated)		Account-ability
	NaCN	CaO	NaCN	CaO	pH	pH	Au	Assayed	Calculated	Au	Au	NaCN	CaO	Solution + Carbon	Solid	Sol + Carbon	Solid	Au
	g/t	kg/t	ppm	%	Initial	Final	ppm	Au (g/t)	Au (g/t)	g/t	g/t	kg/t	kg/t	Au %	Au %	Au %	Au %	%
24	700	0.95	130	0.007	7.7	10.5	0.01	1.23	1.20	0.08	29	0.45	0.82	91.14	93.5	93.34	93.34	97.64

13.1.12 Gravity and CIL Variability Test Work

Gravity and CIL variability test work was conducted under the following optimised test conditions:

- Gravity: Scalp off gravity gold and leach gravity middlings and tails
- Gravity concentrate: Intensive leach
- Gravity tails: Subject to CIL recovery tests
- CIL grind: 80 % passing 150 µm
- CIL cyanide addition: 0.7 kg/t
- Sparging during leach: Air sparging
- CIL residence time: 24 h

Table 13.16 shows a summary of the variability test work results.

Table 13.16: Variability Test Work Results Summary

Samples	Sample ID	Ore Type	Calculated Head (g/t Au)	Gravity Recovery	ILR	Leach on Middlings and Tailings				Overall Recovery (%)
				% Au	% Au	Au Dissolution (%)	Leach Residue (g/t Au)	Cyanide Consumption (kg/t)	Lime Consumption (kg/t)	
Initial samples	North	Laterite and Saprolite	0.52	61.84	97.11	89.53	0.02	0.45	0.47	96.13
	Central	Saprolite	0.37	38.98	92.65	91.27	0.02	0.49	0.82	94.62
	South	Laterite and Saprolite	0.33	40.82	98.87	85.88	0.03	0.47	0.65	90.97
Additional samples	North Com	Saprolite	0.66	61.28	99.09	89.17	0.03	0.31	0.21	95.47
	South Com	Saprolite	1.24	61.49	98.81	88.98	0.04	0.37	0.47	96.78
	Central Com	Saprolite	1.24	84.92	96.04	90.12	0.02	0.23	0.45	98.39
	Waste Com	Saprolite and Transition	0.86	71.32	96.08	88.37	0.03	0.23	0.35	96.49
	Heap Leach	Saprolite and Laterite	1.30	70.53	92.86	85.75	0.06	0.19	0.35	95.38
	CIL Composite sample	Saprolite and Laterite	3.13	61.03	98.92	93.34	0.08	0.45	0.82	97.45
Variability Results summary	Maximum		3.13	84.92	99.09	93.34	0.08	0.49	0.82	98.39
	Minimum		0.33	38.98	92.65	85.75	0.02	0.19	0.21	90.97
	Average		1.07	61.36	96.71	89.16	0.04	0.35	0.51	95.74
	85th Percentile		1.29	71.17	98.91	91.04	0.06	0.46	0.78	97.31

The results indicated the following:

- Head grades ranged from 0.33 g/t Au to 3.13 g/t Au with an average of 1.07 g/t Au.
- Gravity gold recoveries ranged from 38.98 % to 84.98 % in line with findings from the composite sample.
- CIL gold dissolutions ranged from 85.75 % to 93.34 %.
- Regardless of the head grade variations, the recoveries were high, ranging from 90.97 % to 98.39 % in line with the optimised recovery of 97.45 %.
- Cyanide consumptions ranged from 0.19 kg/t to 0.49 kg/t, which were consistent with the optimised composite sample consumption of 0.45 kg/t.
- Lime consumptions ranged from 0.21 kg/t to 0.82 kg/t, which were consistent with the optimised composite sample consumption of 0.82 kg/t.

The variability test work thus confirmed that the optimised process conditions selected can be used for the Kobada ore (saprolite and laterite).

13.1.13 Oxygen Uptake

Oxygen uptake tests were conducted over 6 h, and the results are shown in Table 13.17.

Table 13.17: Oxygen Uptake Results Summary

Time (h)	Oxygen Uptake (mg/L/min)
1	0.038
2	0.035
3	0.033
4	0.032
5	0.031
6	0.032
Average	**0.0335**

The results indicated an oxygen uptake of 0.034 mg/L/min, which is considered a minimal oxygen demand. An oxygen demand of less than 0.15 mg/L/min indicates that aeration will be sufficient to provide the oxygen required for cyanidation. This outcome ties in well with the decision from the effect of air and oxygen sparging tests, which showed that air sparging was sufficient to provide the oxygen required to leach the Kobada ore (saprolite and laterite).

13.1.14 Viscosity

Viscosity tests were conducted on the ROM material at various percentages of solids. Table 13.18 shows a summary of the results.

Table 13.18: Viscosity Results Summary

Shear Rate	Viscosity at Various Solids Percentages			
s^{-1}	30 % w/w	40 % w/w	50 % w/w	60 % w/w
2.2	13.2	97.8	756	4 950
2.64	14	83.5	660	4 789
4.4	11.1	55.5	447	3 569
6.6	10.2	41.6	337	3 011
11	8	29.5	231	2 203
13.2	7.8	25.9	206.5	1 926
22	8.34	19.02	148.2	

The results indicated that the ore became viscous at 60 % solids. This indicates that pumping will need to be conducted between 50 % and 60 % solids, with 55 % solids used in the mass balances.

13.1.15 Thickening and Rheology

Thickening and rheology test work was performed on the slurry from the bulk leach test. Table 13.19 shows a summary of the thickening and rheology results.

Table 13.19: Thickening and Rheology Results

Test Work	Unit	Value
Solids Flux Rate	$t/h/m^2$	0.2
Flocculant Selected		Flomin 934VHM
Underflow Solids Density	% m/m	49

The thickening test work indicated the following:

- No flocculant polymer was able to produce clear supernatant liquor without coagulation. After coagulation with inorganic and organic coagulants, Flomin 934VHM gave the fastest settling rates and near-to-best supernatant liquors.
- The optimum feed well density was 5 % w/w.
- The pilot thickener achieved an underflow density of 49 %.
- The solids flux rate was 0.2 $t/h/m^2$.

13.1.16 Cyanide Detoxification

Cyanide detoxification test work was conducted targeting a residual WAD cyanide, after detoxification, of < 5 ppm. Batch detoxification test work was conducted first to determine a detoxification process that would be applicable to the Kobada ore. The batch tests were followed by continuous cyanide detoxification test work on the slurry to confirm the reagent consumption.

During the batch detoxification tests, SMBS-copper and hydrogen peroxide methods were investigated on the slurry, and it was concluded that the best results were obtained using SMBS-copper addition.

Therefore, a continuous cyanide detoxification was performed on the leached slurry using the SMBS-copper detoxification method.

Table 13.20 shows the results of the detoxification feed and discharge characterisation.

Table 13.20: Detoxification Feed and Discharge Characterisation

Element	Unit	Feed	Discharge
Free Cyanide	ppm	113.0	0.0
WAD Cyanide	ppm	115.0	0.002
S(CN)	ppm	1.0	0.459
Al	ppm	11.7	1
Ca	ppm	89.0	125
Fe	ppm	< 2.0	< 2.0
Cu	ppm	3.4	< 2.0
Ni	ppm	< 2.0	< 2.0
Co	ppm	< 2.0	< 2.0
As	ppm	< 2.0	< 2.0
Pb	ppm	< 2.0	< 2.0
Mg	ppm	3.5	11.4

The continuous cyanide detoxification test work indicated the following:

- A residence time of 1 h was sufficient to destroy the WAD cyanide to the required levels of less than 5 ppm.
- The SMBS-copper process yielded a 99 % WAD cyanide destruction.
- SMBS consumption was 5.48 g of SMBS per gram of WAD cyanide destroyed.
- The target copper concentration of 50 ppm was sufficient for the reactions to take place.

13.1.17 Arsenic Balance

An arsenic balance was performed to assess the distribution of arsenic in the products from the recovery test work and to determine if arsenic reports in the final tailings discharged into the environment.

Figure 13.7 shows a summary of the arsenic balance, which indicates that 0.18 % of the arsenic (1.3 ppm) reports to the filtrate post-detoxification. This warrants an arsenic precipitation process to be included in the circuit to remove any arsenic prior to discharge into the environment.

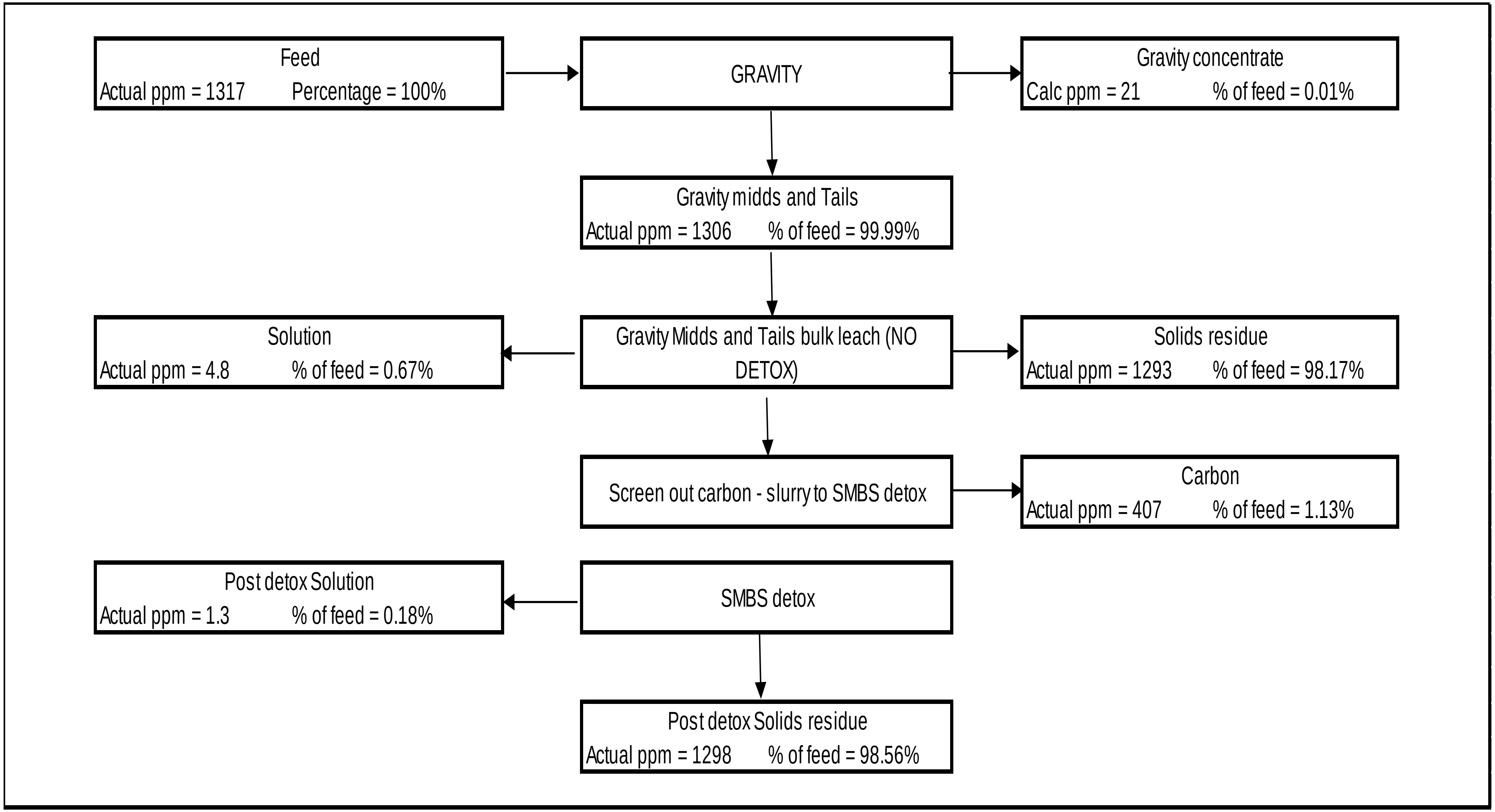


Figure 13.7: Arsenic Balance

13.2 SULPHIDE TESTWORK

13.2.1 Introduction

SENET was requested by African Gold Group (AGG) to conduct a bankable feasibility study (BFS) on the Kobada sulphides Gold Project. SENET developed a metallurgical test work programme whose results were used to develop the process flowsheet and determine values to size and design the process plant.

No historic test work was performed on the Kobada sulphide ore. The starting point for metallurgical test work on the sulphide ore were the following on a composite sample (this was Phase 1 of the test work programme)

1. Comminution test work
2. Multi head Assays.
3. Diagnostic leach
4. Investigate response of the sulphide ore to the process route already established for the Kobada oxide ore. The Kobada oxide ore responded well to gravity followed by cyanidation.
5. Optimisation test work on the selected process route to establish optimum conditions for maximum gold recovery.

The Phase 1 test work results indicated that the Kobada sulphide ore is free milling and responds well to gravity followed by cyanidation giving 96% overall gold recovery, fast leach kinetics and low reagents consumptions (cyanide and lime) on gravity middlings and tailings leach.

Phase 2 of the test work programme looked at variability comminution and variability recovery test work using the optimum conditions as established to confirm comminution parameters and gold recovery across the body.

13.2.2 Sample Selection

Samples for metallurgical test work were selected from the north zone, central zone and south zone domains to cover the entire deposit. Figure 13.8 shows a diagrammatic representation of where all the sulphide samples were taken from.

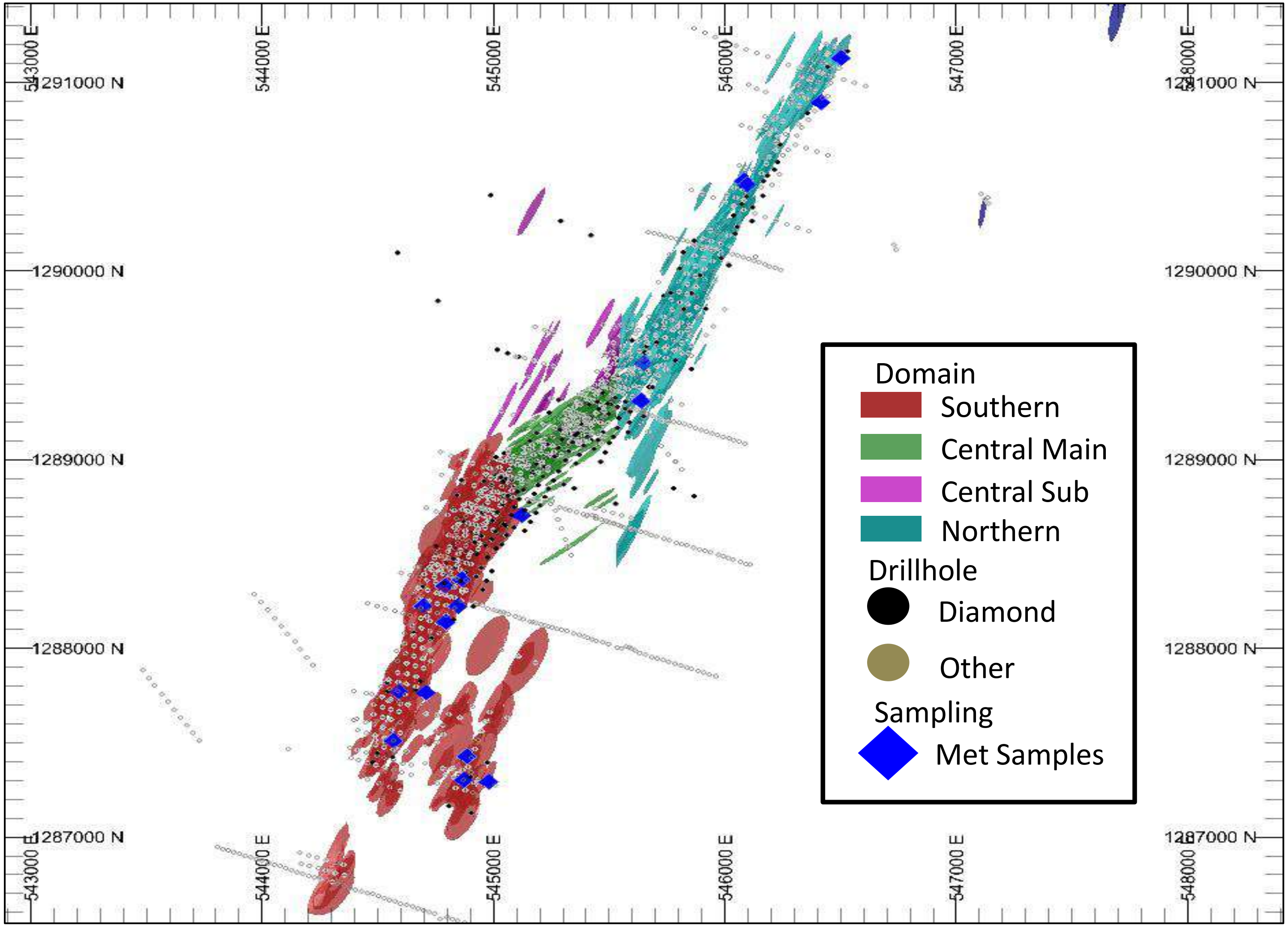


Figure 13.8: Sulphide Sample Selection

13.2.3 Summary of Results and Design Values

Table 13.21 shows a summary of the Phase 1 test work results.

Table 13.21: Summary of the Phase 1 Test Work Results

Test Work	Outcome
Comminution	Comminution test work on the sulphide ore indicated the following values: • BBWi ranged from 9.0 kWh/t to 15.7 kWh/t • Ai ranged from 0.035 to 0.284 The Kobada sulphide ore has soft to medium comminution indices (Bond ball work index (BBWi)and abrasion index (Ai) therefore, the energy requirements for crushing and milling are not expected to be excessively high. Liner wear and media wear are also not expected to be high.
Head Assays	Multi-head assays did not show the presence of deleterious elements except for arsenic, whose value were around 1 830 ppm Arsenic was, therefore monitored in the test work and an arsenic balance was performed. Copper levels were low 42 ppm indicating that cyanide consumption is not expected to be high due to complexing with copper. Carbonate levels are high at 4.9% CO_3^{2-}. Lime requirements during cyanidation are not expected to be high Silver levels were low at 1.6 g/t. Therefore no issues due to co-loading are expected in elution.
Gravity Recovery	Batch laboratory gravity recovery gave 61.85% gold recovery which was an indication of indicated a high gravity recoverable gold.
Leach Optimisation	Leach optimisation tests were conducted on the gravity middlings and tailings. The following optimum conditions were determined for maximum gold dissolution: • Grind to leach: 80 % passing 75 µm • Cyanide addition: 0.35 kg/t NaCN • Leach process: carbon-in-leach (CIL) due to the presence of preg-robbers • Air sparging: required during leach. • pH: 10.5 (with a lime consumption of 0.30 kg/t) • Residence time: min. 12 h and design of 16 h
Variability Test work	Variability test work confirmed the high gold recovery and consistently low cyanide and lime consumption. The variability results showed the following: • Gold head grades ranged from 0.39 g/t Au to 1.34 g/t Au • Cyanide consumption ranged from 0.14 kg/t to 0.31 kg/t in line with optimised cyanide consumption of 0.35 kg/t. • Lime consumptions ranged from 0.23 kg/t to 0.31 kg/t in line with the optimised lime consumption of 0.30 kg/t. • Overall gold recovery ranged from 77 % to 95 % with low value corresponding to low head grade
Predicted Plant Tails	$T_p = 0.06 * [(H / 1.11)^{0.5}]$ where T_p = Predicted Tails (g/t) H = Head grade g/t
Arsenic Balance	An arsenic balance showed that arsenic dissolved into solution of 3.63 ppm arsenic.

Following completion of the BFS test work, design values were selected to size the process plant. Table 13.22 shows the design values selected.

Table 13.22: Recommended Values for Design Purposes (Sulphide Ore)

Test Work		Unit	Sulphide	Notes
Comminution	BBWi	kWh/t	14.8	
	Ai	g	0.1716	
	BRWi	kWh/t	17.76	Assumed 1.2 × BBWi
	CWi	kWh/t	15	Assumed OMC
	SG	-		
Recovery	Gravity Recovery	%Au	62.98	
	Intensive Leach on Gravity conc	%Au	96.10	
	Cyanidation on gravity Midds & Tails			
	Optimum grind to leach		80%-75µm	
	Leach process	-	Carbon in leach (CIL)	
	Residence time	h	Min. 12 h and design of 16 hours	Test work indicated an optimum of 24 h but leach kinetics were fast.
	Percent solids in leach	% w/w	50	
	Oxygenation during leach		Air sparging	The oxide circuit already designed has air blowers.
	Optimum cyanide addition	kg/t	1.0	
	Gold dissolution	%Au	94.16	
	Cyanide Consumption	kg/t	0.35	
	Lime consumption	kg/t	0.30	
	Overall gold dissolution	%Au	95.38	Overall gold dissolution refers to gold going into solution and does not include other losses incurred in the plant during operations
	Arsenic in solution post leach	ppm As	3.6	Arsenic levels above the standard of 0.1ppm. Therefore, an arsenic precipitation plant will be required. Oxide circuit design already catered for an arsenic precipitation plant
	WAD Cyanide in solution post leach	ppm WAD CN		

13.2.4 Test Work Results

The Kobada test work was split into two phases namely Phase 1 and Phase 2.

- Phase 1 – Test work to establish the best process route for treating the Kobada sulphide ore followed by optimisation test work to establish optimum parameters for maximum gold extraction.
- Phase 2 – Variability comminution and variability recovery test work.

Detailed results and test work procedures are contained in the following reports

- MSA, "Kobada Sulphides", 23 June 2021, Report Number 20-205 Rev1.
- MSA, "Kobada Sulphides Variability Testwork", 24 June 2021, Report Number 21-054 Rev 2.

13.2.4.1 Phase 1 Test Work

13.2.4.2 Head Assays

Representative sub samples were submitted for head assays at Superlabs and SGS. Table 13.23 shows a summary of the gold head assays.

Table 13.23: Gold Head Grades

Lab	Element	Unit	Head
Superlabs	Au	g/t	0.44
	Au (duplicate)	g/t	0.48
SGS	Au 1	g/t	0.51
	Au 2	g/t	0.58
	Au 3	g/t	0.54
Average			**0.51**

Initial duplicate head assays were done at Superlabs and the results showed an average head grade of 0.46 g/t Au. This was lower than the expected composited head grade of 1.36 g/t from the geologist. Confirmatory head assays were performed at SGS. The confirmatory head assays at SGS confirmed that the head grade is low at around 0.5 g/t Au. The average of all the head assays was 0.51 g/t Au.

Table 13.24 shows the multi head assays.

Table 13.24: Multi Head Grade Analysis

Au (g/t)	0.51
S (%)	0.50
ELEM_S (%)	0.73
S^{2-} (%)	0.40
SO_4^{2-} (%)	<0.4
C (%)	1.17
CO_3^{2-} (%)	4.9
GRAP_C (%)	0.22
ORG C (%)	<0.05

Al (%)	7.94
Ca (%)	1.46
Fe (%)	4.44
K (%)	2.04
Mg (%)	1.12
Mn (%)	0.05
P (%)	0.07
Si (%)	>25
Ti (%)	0.32

Ag (g/t)	1.6
As (ppm)	1830
Ba (ppm)	536
Be (ppm)	<5
Bi (ppm)	9.8
Cd (ppm)	0.4
Ce (ppm)	53.1
Co (ppm)	13.6
Cr (ppm)	87

Cs (ppm)	4.4
Cu (ppm)	42
Dy (ppm)	3.29
Er (ppm)	1.52
Eu (ppm)	0.99
Ga (ppm)	18.1
Gd (ppm)	4.2
Ge (ppm)	2.1
Hg (ppm)	0.1

Ho (ppm)	0.62
In (ppm)	0.9
La (ppm)	29.5
Li (ppm)	57
Lu (ppm)	0.36
Mo (ppm)	4.1
Nb (ppm)	8.2
Nd (ppm)	24.9
Ni (ppm)	66

Pb (ppm)	16.9
Pr.(ppm)	6.95
Rb (ppm)	88.2
Sb (ppm)	2.3
Sc (ppm)	12.7
Sm.(ppm)	4.4
Sn (ppm)	4.8
Sr (ppm)	323
Ta (ppm)	1.8

Tb (ppm)	0.43
Th (ppm)	5.8
Tl (ppm)	<0.5
Tm (ppm)	0.28
U (ppm)	2.77
W (ppm)	21.4
Y (ppm)	15.3
Yb (ppm)	1.6
V (ppm)	101
Zn (ppm)	36

The results indicate the following:

- Arsenic levels are high at 1,830 ppm As. These levels are in line with what was obtained for saprolite/laterite which ranged from 552 ppm to 2,330 ppm. The additional tests recommended include an arsenic balance which will show if arsenic levels in the final tails solution are high. If arsenic levels in the tails are higher than the environmental guidelines, an arsenic precipitation circuit is already part of the oxide design and will treat the arsenic to within an acceptable environmental level.
- Carbonate levels are high at 4.9 % CO_3^{2-}. Lime requirements during cyanidation are not expected to be high.
- Mercury is low at 0.1 ppm Hg. No special mercury handling equipment will be required in the gold room to handle mercury.
- Copper levels are low at 42 ppm. Cyanide consumption due to complexing with copper is not expected to be high.

13.2.4.3 Gravity Recovery

Gravity recovery tests performed via a single pass through a centrifugal gravity concentrator showed a high gravity gold recovery of 41.46 % which is consistent with what was obtained for saprolite/laterite. Table 13.25 shows a summary of the gravity recovery results.

Table 13.25: Gravity Recovery Test Work Results

Fraction	Mass		Gold		Cumulative	
			Grade	Recovery	Mass	Recovery
	g	%	g/t	%	%	%
Gravity conc	101	2.52	8.24	41.46	2.52	41.46
Gravity Midds and Tails	3 899	97.49	0.30	58.54	100.00	100.00
Total	4 000	100.00	0.50	100.00		

13.2.4.4 Gravity Concentrate Intensive cyanidation

Intensive cyanidation tests were performed on the gravity concentrate. Table 13.26 and Figure 13.9 show the intensive cyanidation results.

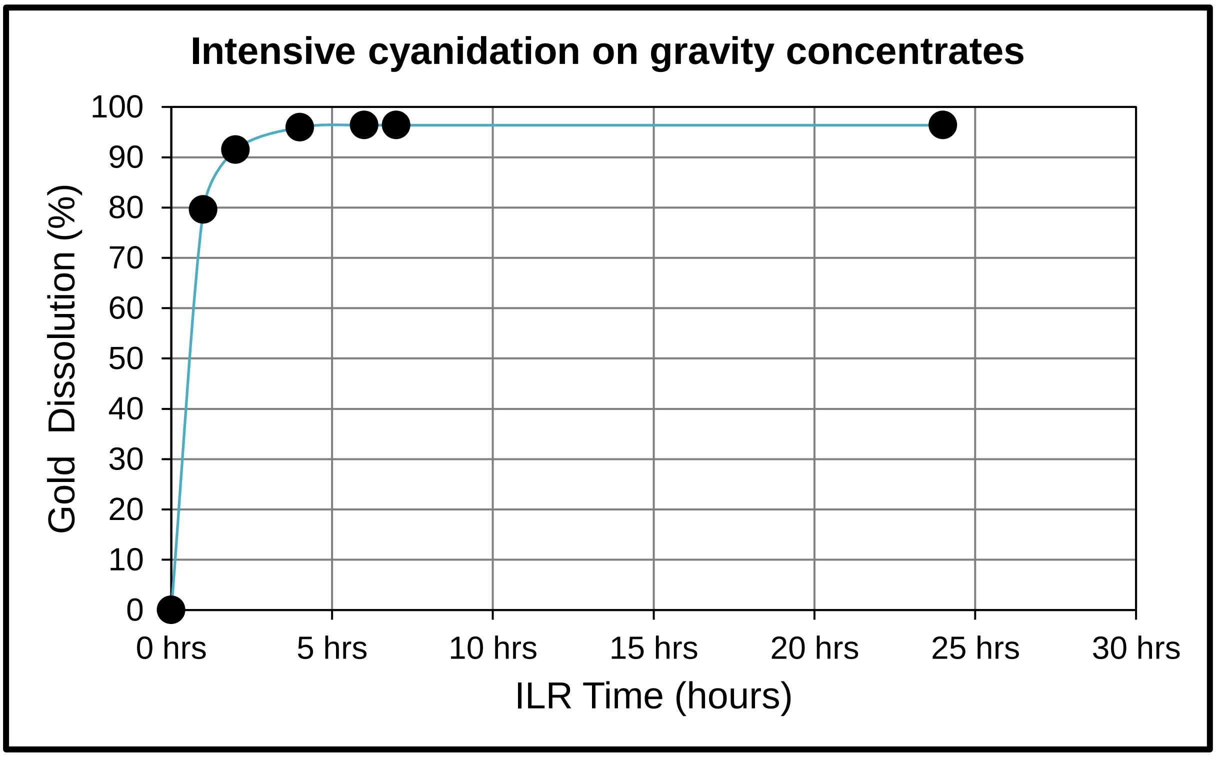


Figure 13.9: Intensive Cyanidation on Gravity Concentrates

Table 13.26: Intensive Cyanidation Results

Leach Time	NaCN	Recovery	pH	Sample	leached	Dissolved Oxygen measured
		Au		Au	Au	DO
	ppm	%		mg	mg	mg/L
0 h	20000	0	11.64	0	0	15.93
1 h	20000	79.61	11.31	0.656	0.656	30.18
2 h	20000	91.48	11.16	0.753	0.098	22.49
4 h	20000	95.95	10.74	0.79	0.037	26.15
6 h	20000	96.37	10.47	0.794	0.003	21.71
7 h	20000	96.37	10.66	0.794	0	26.69
24 h	20000	96.37	10.6	0.794	0	

The ILR tests indicate that a gold dissolution of 96.37 % over 6 h is achieved and this is consistent with what was obtained for saprolite and laterite.

The current gravity recovery equipment selection is adequately sized to recover gravity gold from the sulphides.

13.2.4.5 Effect of Grind

Leach tests were conducted on sub samples of the gravity middling's and tailings milled to varied grind sizes. Table 13.27 shows a summary of the results.

Table 13.27: Summary Results – Effect of Grind

Grind	Gravity Rec. (%)	Au Diss. (%)	Cyanide cons. (kg/t)	Lime cons. (kg/t)	ILR	Overall Diss* (%)
80%-212µm	41.46	50.65	1.20	0.22	96.37	69.60
80%-150µm		66.52	1.18	0.21		78.90
80%-106µm		89.14	1.44	0.20		92.14
80%-75µm		93.55	2.38	0.22		94.72

** Overall gold dissolution refers to gold going into solution and does not include other losses incurred in the plant during operations.*

The results indicate the following:

- Gold dissolution increases with finer grind.
- An optimum grind of 80%-75µm was selected,
- Cyanide consumptions are high ranging from 1.18 kg/t to 2.38kg/t. Cyanide consumptions are high because the tests were done with excess initial cyanide addition of 5kg/t. Cyanide consumption spikes from 1.44kg/t (at 106µm) to 2.38 kg/t (at 75µm). Cyanide optimisation tests were later performed at the optimum grind.

13.2.4.6 Effect of Cyanide Addition

The effect of cyanide addition tests was performed at following varied cyanide additions 0.1,0.3, 0.5,0.7, 1 and 2 kg/t. Table 13.28 shows a summary of the effect of cyanide addition tests.

Table 13.28: Summary Results – Effect of Cyanide Addition

Cyanide Addition (kg/t)	Gravity Rec. (%)	Au Diss. (%)	Cyanide cons. (kg/t)	Lime cons. (kg/t)	ILR	Overall Diss* (%)
0.1	41.46	57.12	0.40	0.09	96.37	73.40
0.3		59.07	0.40	0.27		74.54
0.5		71.66	0.39	0.40		81.91
0.7		76.11	0.39	0.55		84.51
1		93.38	0.38	0.75		94.62
2		93.92	0.36	1.44		94.94
** Overall gold dissolution refers to gold going into solution and does not include other losses incurred in the plant during operations.*						

From 0.7 kg/t to 1 kg/t, overall gold dissolution increases from 84.51 % to 94.62 %, respectively. Increasing cyanide addition from 1 kg/t to 2 kg/t does not significantly increase gold extraction. An optimum cyanide addition of 1 kg/t was therefore selected as optimum.

13.2.4.7 Effect of Residence Time

Leach rate tests were performed on the gravity middlings and tailings at a grind of P_{80} passing 75 µm. Table 13.29 shows a summary of the results and Figure 13.10 shows the leach rate curve.

Table 13.29: Leach Rate Tests

Time	Reagent addition		Head		Residue	Carbon	Reagent consumption	Dissolution (Calc)		Account
	NaCN	CaO	Assayed	Calc.	Au	Au	NaCN	Sol + Carbon	Solid	Au
	g/t	kg/t	Au (g/t)	Au (g/t)	g/t	g/t	kg/t	Au %	Au %	%
2 h	5000	0.24	0.30	0.31	0.25	*2*	0.94	20.48	20.48	104.80
4 h		0.24	0.30	0.29	0.14	*5*	0.94	51.05	51.05	95.33
6 h		0.24	0.30	0.31	*0.02*	10	1.99	93.52	93.52	102.93
8 h		0.24	0.30	0.33	*0.02*	11	1.84	93.92	93.92	109.73
12 h		0.24	0.30	0.33	*0.02*	11	2.13	93.92	93.92	109.73
16 h		0.24	0.30	0.30	*0.02*	10	2.54	93.38	93.38	100.67
18 h		0.24	0.30	0.31	*0.02*	10	2.43	93.61	93.61	104.29
24 h		0.24	0.30	0.31	*0.02*	10	2.38	93.55	93.55	103.39
36 h		0.24	0.30	0.32	*0.02*	11	2.42	93.66	93.66	105.20
48 h		0.24	0.30	0.32	*0.02*	11	2.59	93.66	93.66	105.20

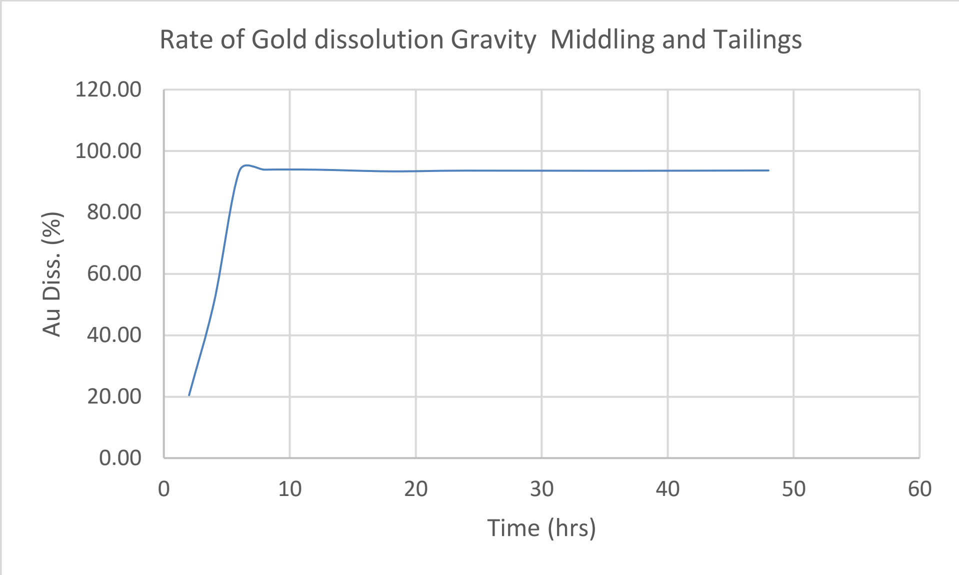


Figure 13.10: Gravity Middlings and Tailings Leach Rate Tests

The following is noted regarding the results:

- Rate of gold dissolution is high and the rate curve "plateaued" after 8 h. Current leach design residence time of 24 h will be more than adequate when treating sulphides.
- Lime requirements are low at 0.24 kg/t; this is in line with expectations as sulphides generally contain carbonates which will result in low lime demand.
- Cyanide consumption at 2.38 kg/t is considered to be high but this is because the test was done at excess cyanide addition in order to avoid cyanide starvation. Cyanide optimisation tests will be conducted to determine cyanide consumptions for design and operating cost estimates.

13.2.5 Phase 2 Test Work

13.2.5.1 Variability Comminution Test Work

The plan was to conduct variability comminution on five samples from the north, central and south zones. But on the samples received, there was insufficient sample on the central zone sample. Therefore, variability comminution test work was conducted on only. Table 13.30 shows a summary of the variability comminution results.

Table 13.30: Variability Comminution Results Summary

Sample ID	BBWi (kWh/t)	Ai (g)
North	9.0	0.0345
South 1	13.9	0.0761
South 2	13.0	0.0913
South 3	14.5	0.0772
South 4	15.7	0.1339

The results indicate the following.

- Bond Ball Work index (BBWi) ranged from 9.0 kWh/t to 15.7 kWh/t with an average of 13.2 kWh/t showing that the sulphide ore is soft to medium hard in terms of ball milling. The composite sample showed a BBWi value of 13.5 kWh/t which compares well to the average of the variability of 13.2 kWh/t.
- Abrasion index (Ai) values ranged from 0.0345 g to 0.1339 g with an average of 0.086 g showing that the sulphide ore is low abrasive. The composite sample showed an Ai value of 0.284 g which shows that the ore is medium abrasive. Liner and media wear is not expected to be high.

13.2.5.2 Variability Recovery Test Work

Upon completion of recovery optimisation test work on a composite sample, the optimum conditions selected were tested on variability samples to determine if the selected process route and optimum conditions selected are applicable across the entire orebody. Six variability samples were selected from the North, South and Central zones.

The following optimum leach conditions were selected for the sulphide ore:

- Grind: 80 %-75 µm.
- Leach process: Carbon in leach (CIL).
- Cyanide addition – 1 kg/t.
- Lab residence time – 12 to 24 hours
- pH – 10.5.
- Percent solids – 50 % w/w

Table 13.31 shows a summary of the variability recovery results.

Table 13.31: Variability Recovery Results Summary

Sample ID	Head Grade	Gravity Rec.	ILR	Leach on Midds & Tails	Cyanide Cons	Lime Cons	Leach Tails	Overall Gold Dissolution*
Composite	g/t Au	%Au	%Au	% Au	g/t	g/t	g/t Au	%Au
Central	0.39	41.82	84.15	94.05	0.14	0.26	0.02	89.91
North	0.49	49.56	87.08	66.67	0.46	0.27	0.07	76.78
South 1	0.47	40.40	76.55	95.16	0.29	0.29	0.02	87.64
South 2	0.48	57.63	94.61	91.85	0.26	0.23	0.03	93.44
South 3	1.34	73.18	96.07	92.67	0.27	0.31	0.02	95.16
South 4	1.03	46.24	81.15	89.90	0.29	0.23	0.06	85.85
85th percentile	1.11	61.52	94.97	94.33	0.34	0.30	0.06	93.87
Minimum	0.39	40.40	76.55	66.67	0.14	0.23	0.02	76.78
Maximum	1.34	73.18	96.07	95.16	0.46	0.31	0.07	95.16
* Overall gold dissolution refers to gold going into solution and does not include other losses incurred in the plant during operations								

The results indicate the following:

- Gravity gold recovery vary from 40.40 % to 73.18 % with an 85th percentile of 61.52 %. Theseresults indicate that the Kobada sulphide ore is amenable to gravity recovery.
- The 85th percentile overall gold dissolution is 93.87% which confirm initial findings that overallgold dissolution above 90% can be achieved on the sulphide ore. The ore leaches well to give tails of 0.03 g/t for low and high grades tested with the exception of North and South 4 samples. Low overall recoveries shown are consistent with low grades tested- but tail gradesare generally less than 0.03 g/t Au. This ties in with findings from the diagnostic leach test which indicated that 0.02 g/t of the gold is locked in quartz or gangue.
- Cyanide consumption was low varying from 0.14 kg/t to 0.46 kg/t with an 85th percentile of 0.34kg/t. These results confirm initial findings that the sulphide ore is a low cyanide consumer.
- Lime consumption was low varying from 0.23 kg/t to 0.31 kg/t with an 85th percentile of 0.30 kg/t.These results confirm initial findings that the sulphide ore is a low lime consumer.

13.2.5.3 Predicted Gold Recovery (White's Rule)

Using White's rule, the predicted tails for the Kobada sulphides is as follows:

$$T_p = 0.06 * [(H / 1.11)^{0.5}]$$

Where T_p = Predicted Tails (g/t)

H = Head grade to be evaluated for predicted gold recovery (g/t)

Using a LOM head grade of 1.3 g/t Au gives a predicted tail of 0.065 g/t which translates to a LOM overall gold recovery of 95 %.

14 MINERAL RESOURCE ESTIMATES

14.1 ASSUMPTIONS, PARAMETERS AND METHODS USED FOR RESOURCE ESTIMATES

This section describes the Mineral Resource estimation process utilised by Minxcon and summarises the key assumptions considered in the estimation. The Mineral Resource has been estimated in conformity to the accepted Canadian Institute of Mining, Metallurgy and Petroleum (CIM) *Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines* (2019) and is reported in accordance with the Canadian Securities Administrators' NI 43-101. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources may be converted into Mineral Reserves.

It is the QP's opinion that the database used in the estimate is of suitable reliability to interpret the geological boundaries and of suitable assay quality to estimate the Mineral Resources for the Project. Measured, Indicated and Inferred Mineral Resources have been declared by Minxcon in this 2021 Mineral Resource estimation.

Leapfrog Geo software was used to construct the geological wireframes/grade shells, while Datamine Studio RM software was used to conduct statistical and geostatistical analyses, conduct variography, and estimate the grade into a block model.

The Mineral Resource estimation methodology involved the following procedures:

- Database compilation
- Geological modelling (discussed in 8.2)
- Domaining
- Statistical analysis
- Data conditioning (compositing and capping)
- Geostatistical analysis and variography
- Bulk density determination
- Block modelling and grade interpolation
- Mineral Resource classification and validation
- Assessment of reasonable prospects for eventual economic extraction (RPEEE)
- Selection of appropriate cut-off grades
- Preparation of the Mineral Resource statement

14.1.1 Database Compilation

The drillhole database consisted of a total of 1,609 drillholes: 218 DD drillholes, 981 RC drillholes, and 410 air core drillholes or auger drillholes. The air core and auger drillholes were excluded from the Mineral Resource database because they were not balanced samples. Furthermore, only drillholes intersecting the mineralised grade shell were used in the Mineral Resource estimation, resulting in a net 865 drillholes used for the estimation (201 DD and 664 RC drillholes). In Figure 14.1, the full drillhole database is shown with the geological model. The wireframe model is colour-coded by domains as determined by the geological modelling.

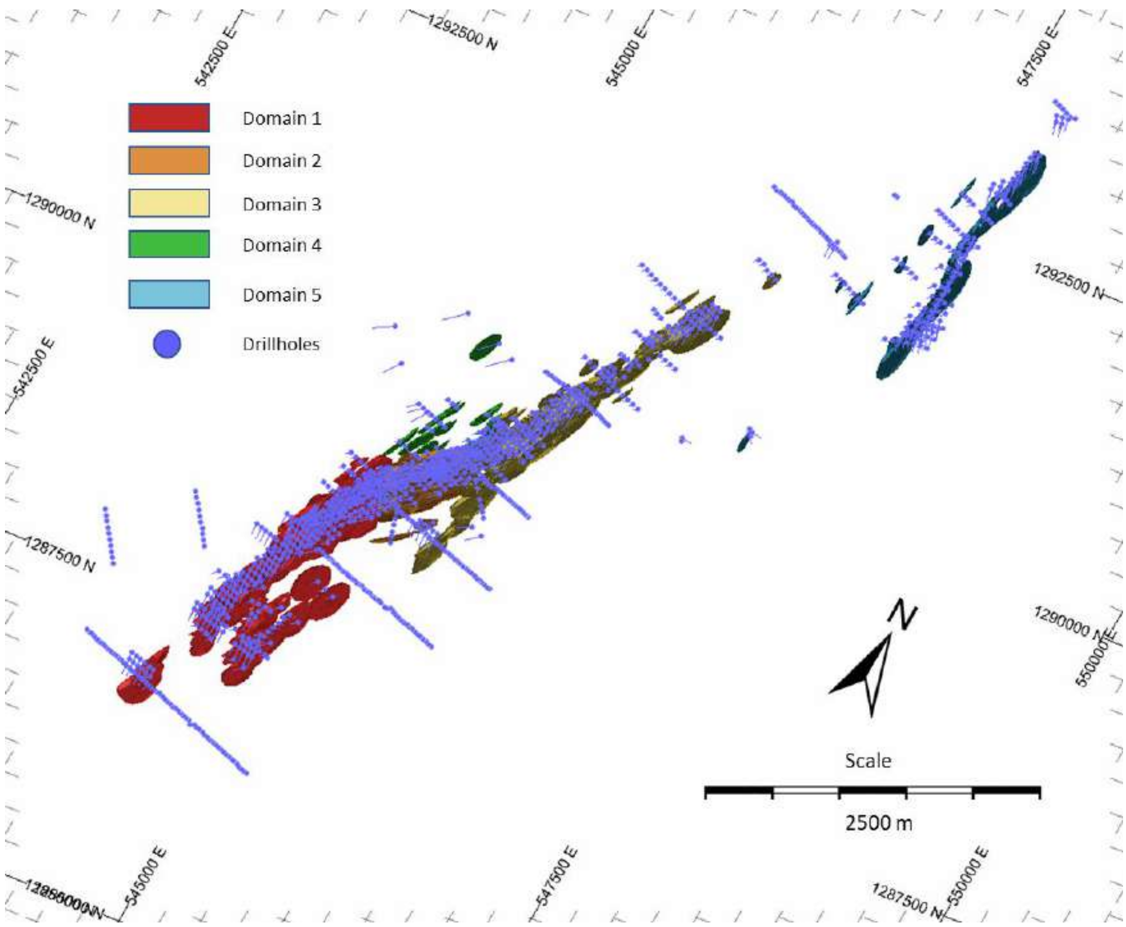


Figure 14.1: Orthogonal View of Structural Domains and Drillhole Database

14.1.2 Geological Model

The geological model is discussed in detail in Section 8.2.

14.1.3 Domaining

Domaining of the orebody was done based on structural boundaries and included five structural and grade-constrained domains. These domains were further split into weathering zones of laterite, saprolite, transition and sulphide. Figure 14.2 shows the main structural domains of the model while Figure 14.3 shows a section through Domain 2 with the weathering profile.

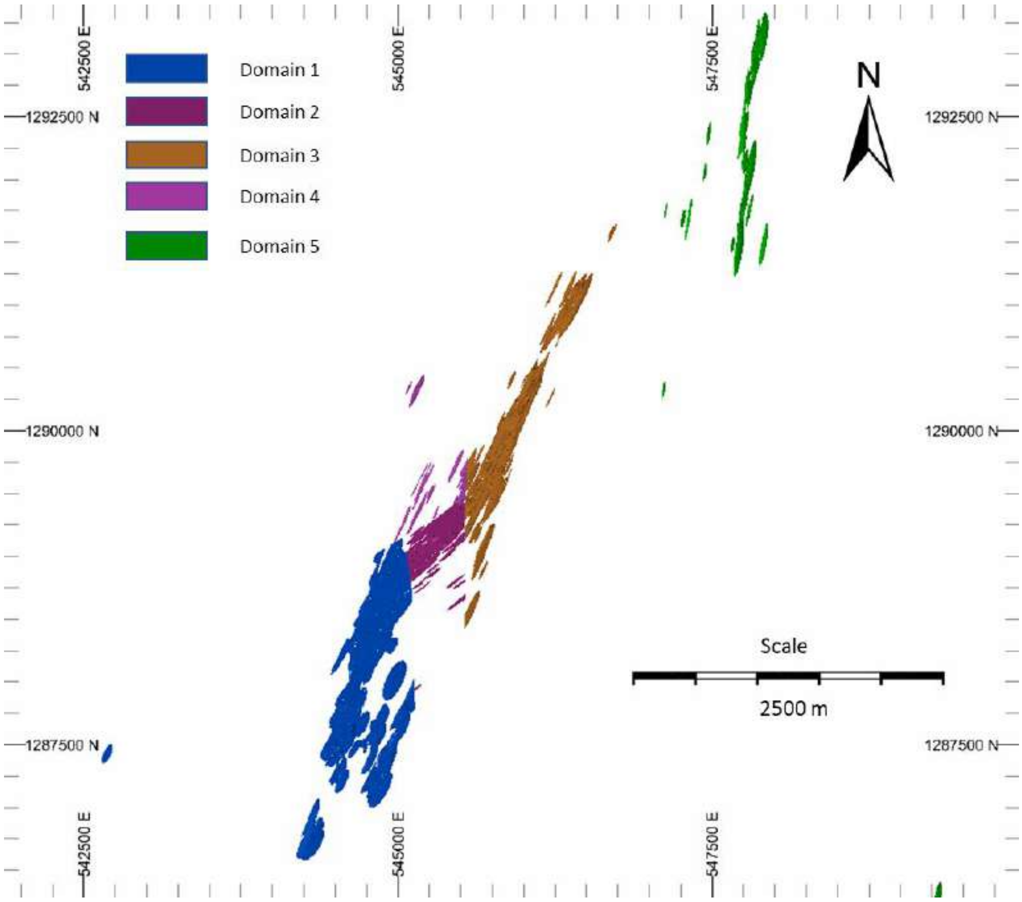


Figure 14.2: Plan View of Structural Domains

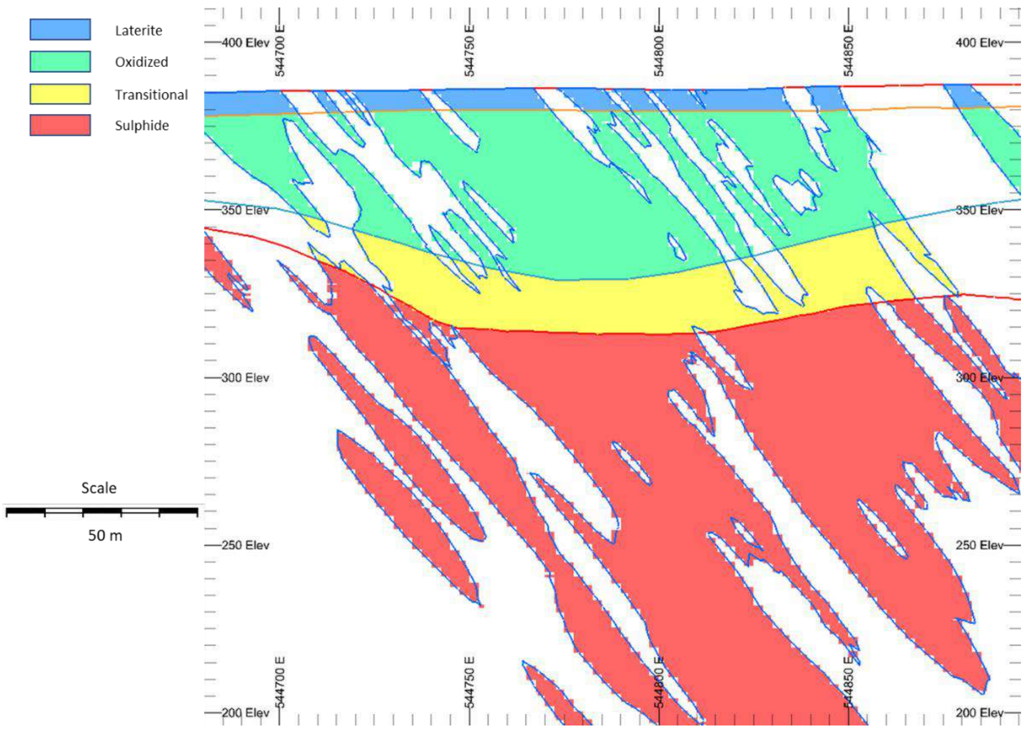


Figure 14.3: General Section View of Weathering Zones in Structural Domain 2 across the Kobada Main Shear looking North

14.1.4 Statistical Analysis

A statistical analysis of the drillholes falling within the mineralised shells was conducted on the Au grade (g/t) and the sample length (m). Table 14.1 represents the uncapped data for the Au values per domain, The mean of the sample lengths shows an average sample length of 1 m, which was then used as the composite length.

Table 14.1: Classic Statistics of the Uncapped Au Grades and Sample Lengths for the Uncomposited Resource Drillholes per Domain

Field	Domain	No. of Samples	Units	Minimum	Maximum	Range	Mean	Variance	STDev
Au	1	15,178	g/t	0.01	145.55	145.54	0.73	9.31	3.05
Au	2	10,869	g/t	0.01	65.92	65.91	0.76	4.07	2.02
Au	3	9,941	g/t	0.01	125.50	125.49	0.75	11.56	3.40
Au	4	623	g/t	0.01	61.75	61.74	0.73	13.90	3.73
Au	5	1,863	g/t	0.01	79.43	79.42	0.67	10.13	3.18
Length	1	15,183	m	0.02	37.50	37.48	1.06	0.33	0.58
Length	2	10,869	m	0.10	39.00	38.90	1.10	0.46	0.68
Length	3	9,947	m	0.01	7.70	7.69	1.02	0.05	0.22
Length	4	623	m	0.10	3.00	2.90	1.04	0.05	0.22
Length	5	1,863	m	1.00	1.00	0.00	1.00	-	-

The 2019 Mineral Resource estimation used subdomaining for the weathering profile. This was reinvestigated, and the mean Au grades per weathering profile show a natural decrease with depth, as shown in Table 14.2. The log probability plots of the total composites show a good correlation and do not indicate that the orebody should be split into subdomains. The log probability plots are shown in Figure 14.4 to Figure 14.6 for the three main domains (1, 2 and 3).

Table 14.2: Comparison of the Mean, Variance and Standard Deviation of the Au Grades per Weathering Profile

Weathering Profile	Mean Grade	Variance	Standard Deviation
	g/t		g/t
Laterite	0.81	10.98	3.31
Oxide	0.74	8.76	2.96
Transitional	0.67	3.15	1.78
Sulphide	0.67	5.78	2.40

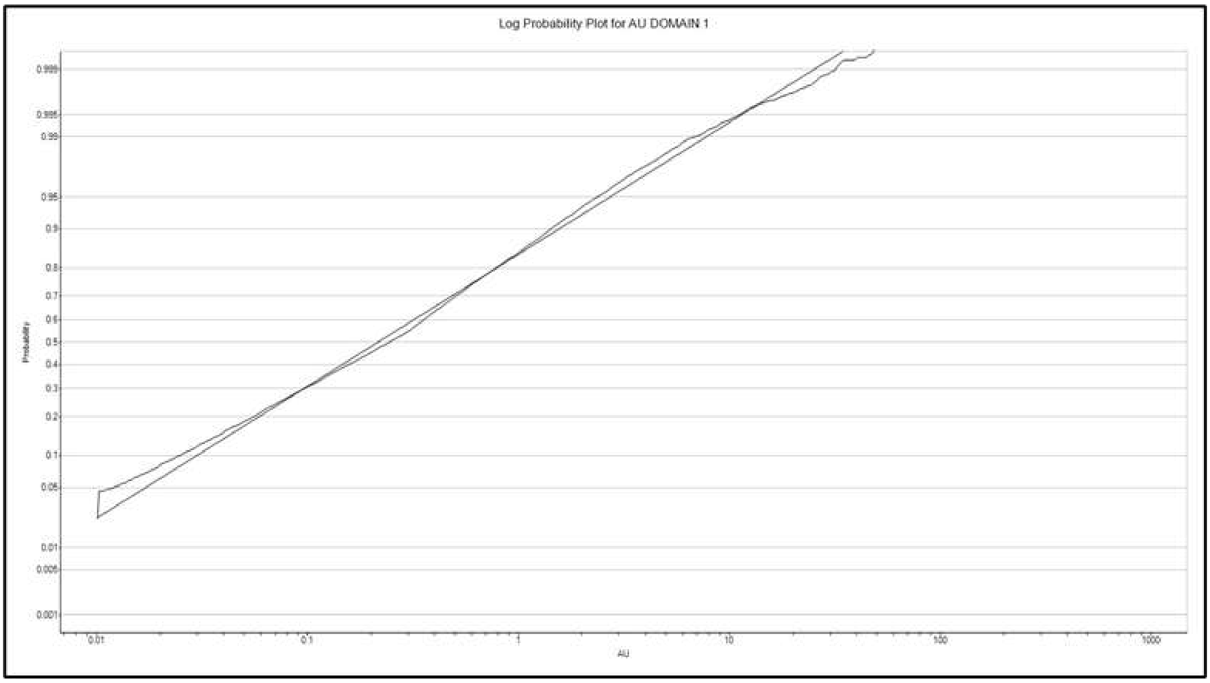


Figure 14.4: Log Probability Plot of the Composited Au Grade – Domain 1

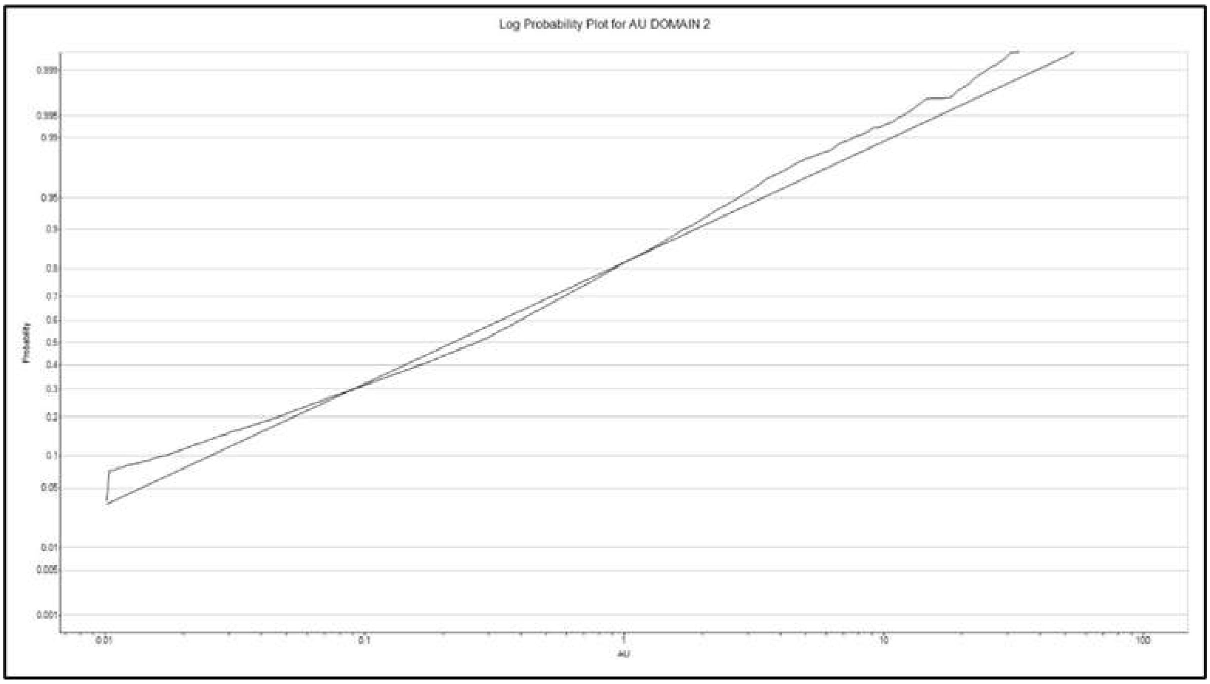


Figure 14.5: Log Probability Plot of the Composited Au Grade – Domain 2

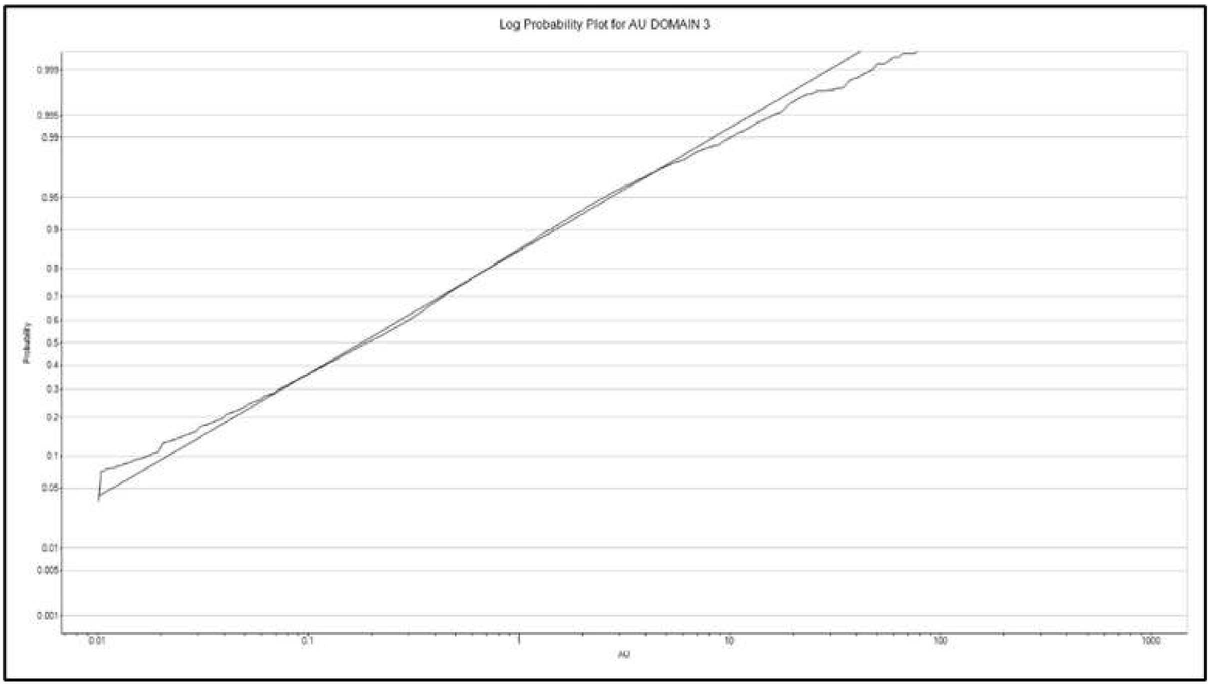


Figure 14.6: Log Probability Plot of the Composited Au Grade – Domain 3

The following histograms show the grade distribution for all the domains and RC and DD drilling types. The data shows a strong log-normal distribution. Figure 14.7 shows the log-normal distribution of the Au grades for the combined RC and DD drilling, while Figure 14.8 and Figure 14.9 show the distribution for the DD drilling and the RC drilling separately.

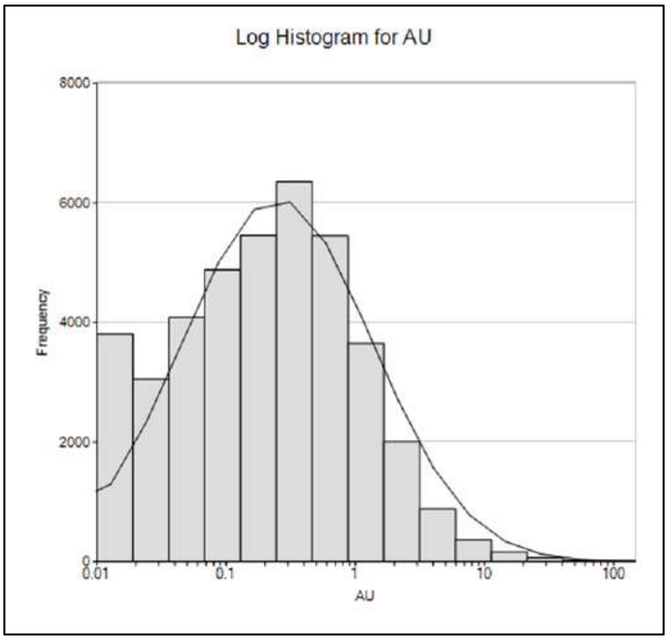


Figure 14.7: Log Histogram of Au Grades for All Domains

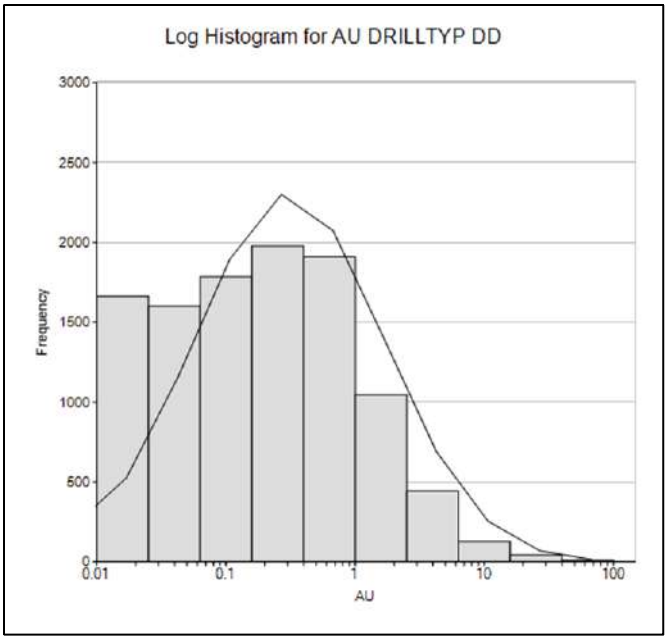


Figure 14.8: Log Histogram of the Au Grades for All Domains – Diamond Drilling

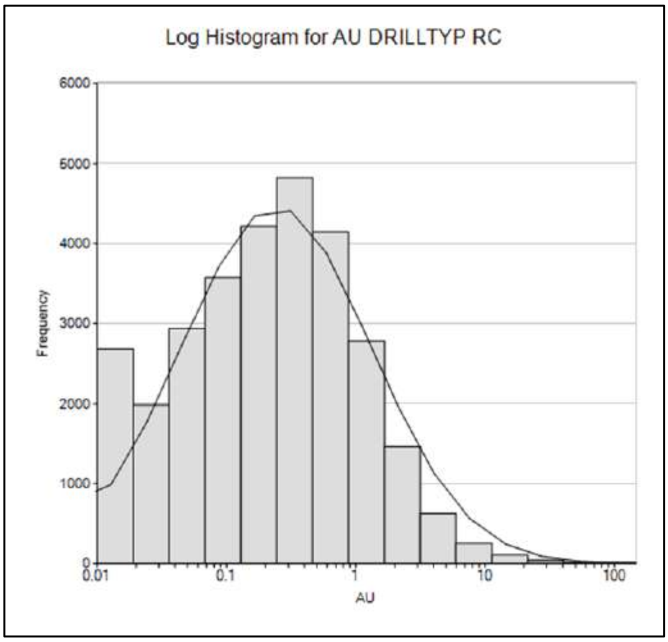


Figure 14.9: Log Histogram of the Au Grades for All Domains – RC Drilling

14.1.5 Data Conditioning (Compositing and Capping)

An investigation of the sample length indicated that the majority of the sampling was conducted on a 1 m sample interval. Drillholes were composited to a 1 m length to standardise the sample support size. A capping analysis of the 1 m composited data was conducted per domain to identify any outliers in that data set. The data was capped per domain, and the capping represents the 99th or 98th percentile. Table 14.3 shows the capping applied, and it should be noted that Domain 4 was capped back to the mean of the 98th percentile.

Table 14.3: Capping of the Composited Resource Sampling

Domain	Maximum	Capping	Mean 99th Percentile	Mean 98th Percentile
	Au g/t	Au g/t	Au g/t	Au g/t
1	145.55	98.00	17.99	5.42
2	65.92	90.00	15.05	5.70
3	125.50	48.00	22.00	6.49
4	61.75	18.00	34.74	8.70
5	79.43	33.00	16.78	3.76

Figure 14.10 to Figure 14.14 show the cumulative coefficients of variance plots for Domains 1, 2, 3, 4 and 5, respectively. These plots show the capping point where the deviation from the norm occurs, which is the basis for capping the Au values.

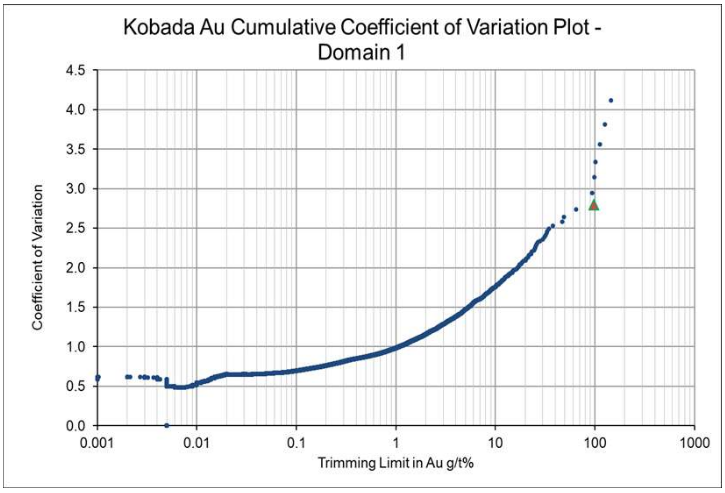


Figure 14.10: Domain 1 – Cumulative Coefficient of Variation Plot Showing a Capping of 98 g/t

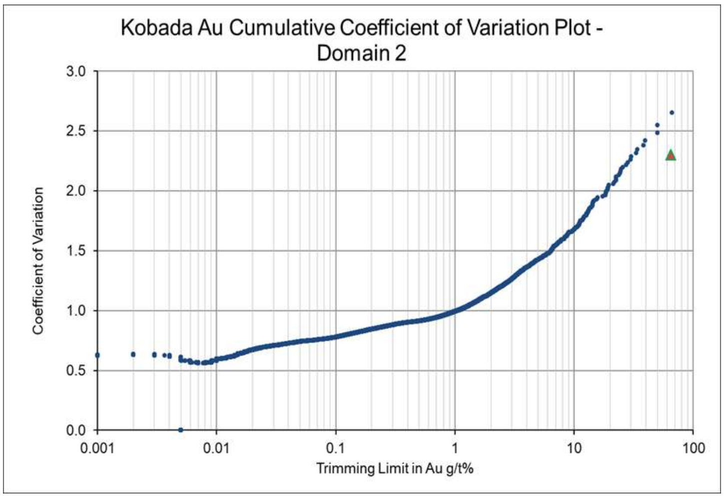


Figure 14.11: Domain 2 – Cumulative Coefficient of Variation Plot showing a Capping of 90 g/t

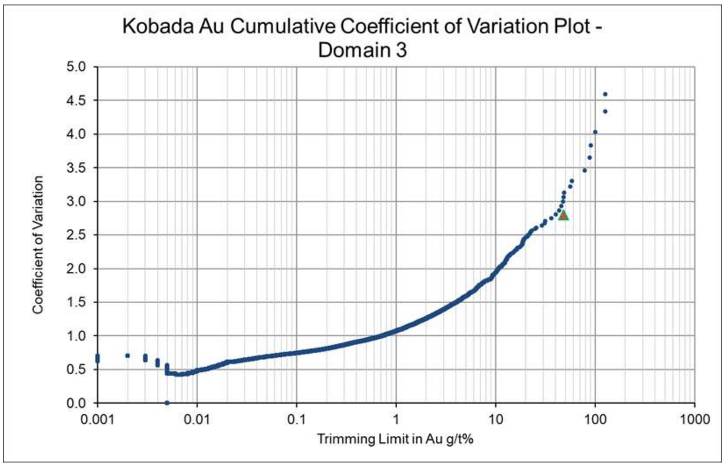


Figure 14.12: Domain 3 – Cumulative Coefficient of Variation Plot showing a Capping of 48 g/t

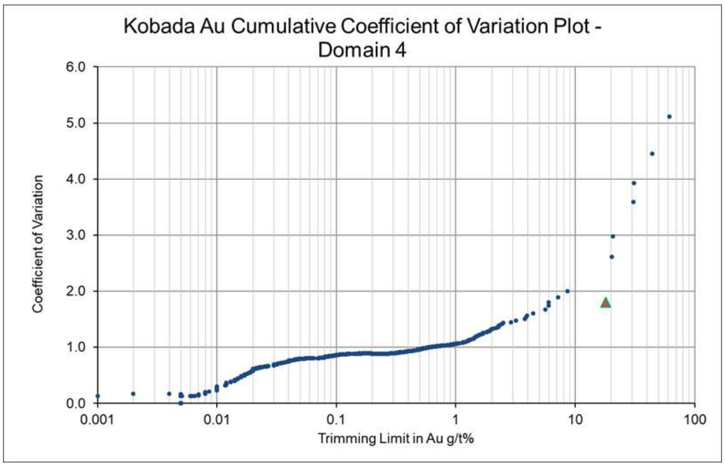


Figure 14.13: Domain 4 – Cumulative Coefficient of Variation Plot showing a Capping of 18 g/t

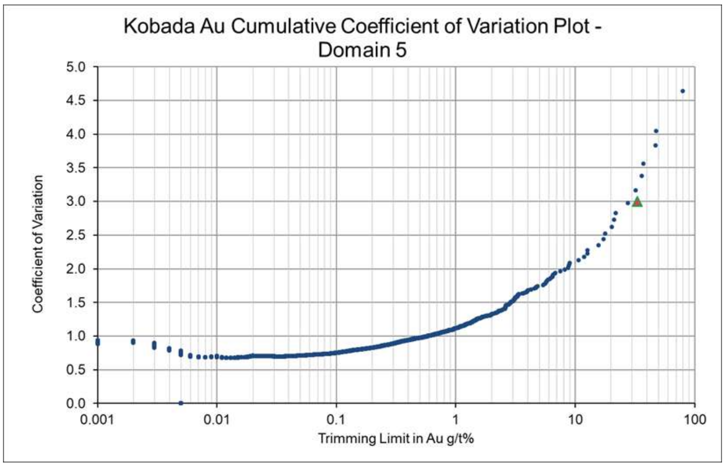


Figure 14.14: Domain 5 – Cumulative Coefficient of Variation Plot showing a Capping of 33 g/t

Table 14.4 shows the statistical analysis of the composited and capped data set used in the estimation.

Table 14.4: Statistical Analysis of the Composited and Capped Data

Field	Domain	No. of Samples	Units	Minimum	Maximum	Range	Mean	Variance	STDev
Au	1	16,143	g/t	0.01	98.00	97.99	0.69	7.04	2.65
Au	2	12,123	g/t	0.01	65.92	65.91	0.74	3.79	1.95
Au	3	11,789	g/t	0.01	48.00	47.99	0.73	6.50	2.55
Au	4	676	g/t	0.01	18.00	17.99	0.54	3.33	1.83
Au	5	1,885	g/t	0.01	33.00	32.99	0.64	6.38	2.53
Au	6	396	g/t	0.01	11.82	11.81	0.53	1.38	1.17
Length	1	16,152	m	0.50	1.20	0.70	1.00	0.00	0.02
Length	2	12,123	m	0.70	1.20	0.50	1.00	0.00	0.02
Length	3	11,805	m	0.75	1.17	0.42	1.00	0.00	0.02
Length	4	676	m	0.95	1.03	0.08	1.00	0.00	0.01
Length	5	1,885	m	1.00	1.00	0.00	1.00	-	-
Length	6	396	m	0.98	1.30	0.32	1.00	0.00	0.02

14.1.6 Variography

The composited and capped data sets were used for experimental variography and variogram modelling for the domains. The search ellipses for the domains were set from the strike and dip directions obtained from the orebody wireframe generation. Variograms were generated for all the domains, and a summary of these variograms is given in Table 14.5. Figure 14.15 to Figure 14.18 show the variograms for Domains 1, 2, 3 and 5. Domain 4 used the variogram of Domain 1 as it is a possible extension north of Domain 1.

Table 14.5: Summary of Variograms

Domain	Parameter	Nugget	Structure 1		Structure 2		Structure 3	
			Range (m)	Variance	Range (m)	Variance	Range (m)	Variance
Domain 1	Au	4.4	3	1.3	20	2.4	39	0.3
Domain 2	Au	1.9	4	0.4	13	1.1	43	0.4
Domain 3	Au	7.0	3	1.4	22	1.3	68	1.2
Domain 4	Au	4.4	3	1.3	20	2.4	39	0.3
Domain 5	Au	7.2	3	1.5	29	0.5	70	1.0

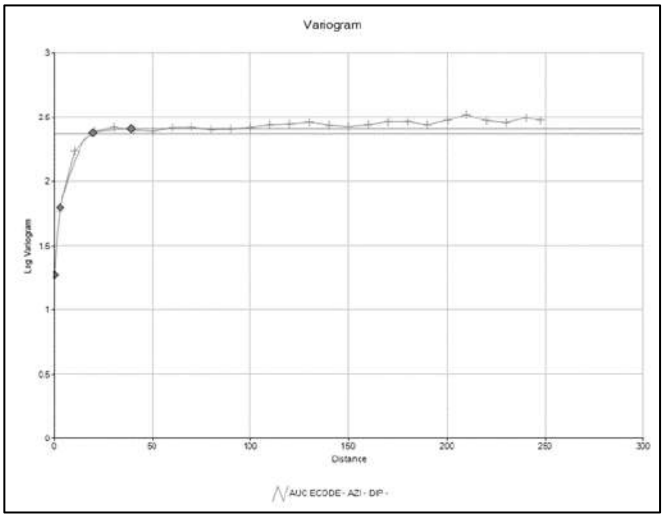


Figure 14.15: Au Variogram of Domain 1

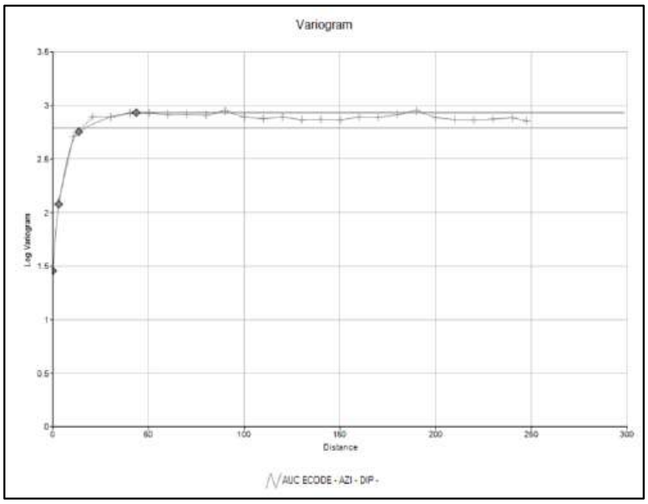


Figure 14.16: Au Variogram of Domain 2

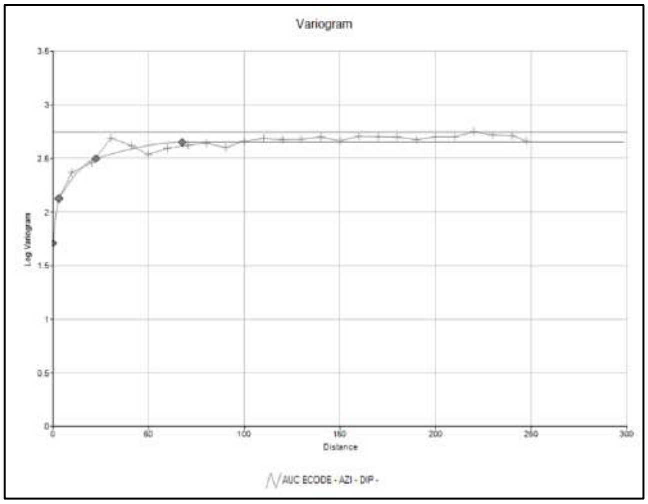


Figure 14.17: Au Variogram of Domain 3

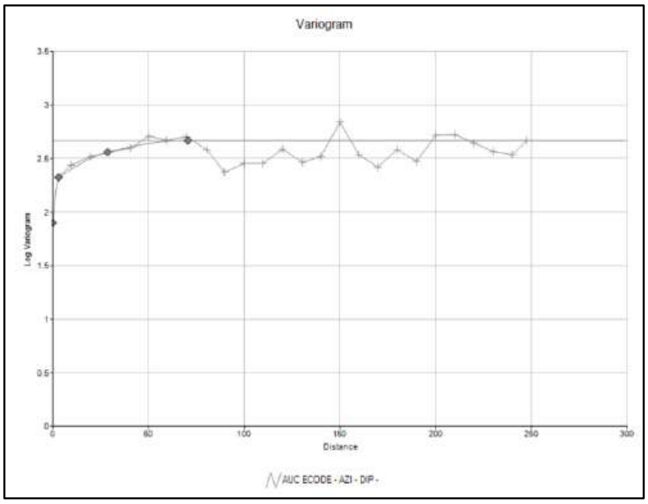


Figure 14.18: Au Variogram of Domain 5

14.1.7 Kriging Neighbourhood Analysis

A Kriging Neighbourhood Analysis (KNA) was conducted on the data to test the optimal block size for the estimation and to determine the minimum and maximum number of samples that would be appropriate to achieve the best estimate based on the variogram. The block size analysis suggested that the appropriate block size for the orebody would be a 10 m block (see Figure 14.19). For this block size, a minimum of six samples and a maximum of 20 samples are needed for the estimate (see Figure 14.20).

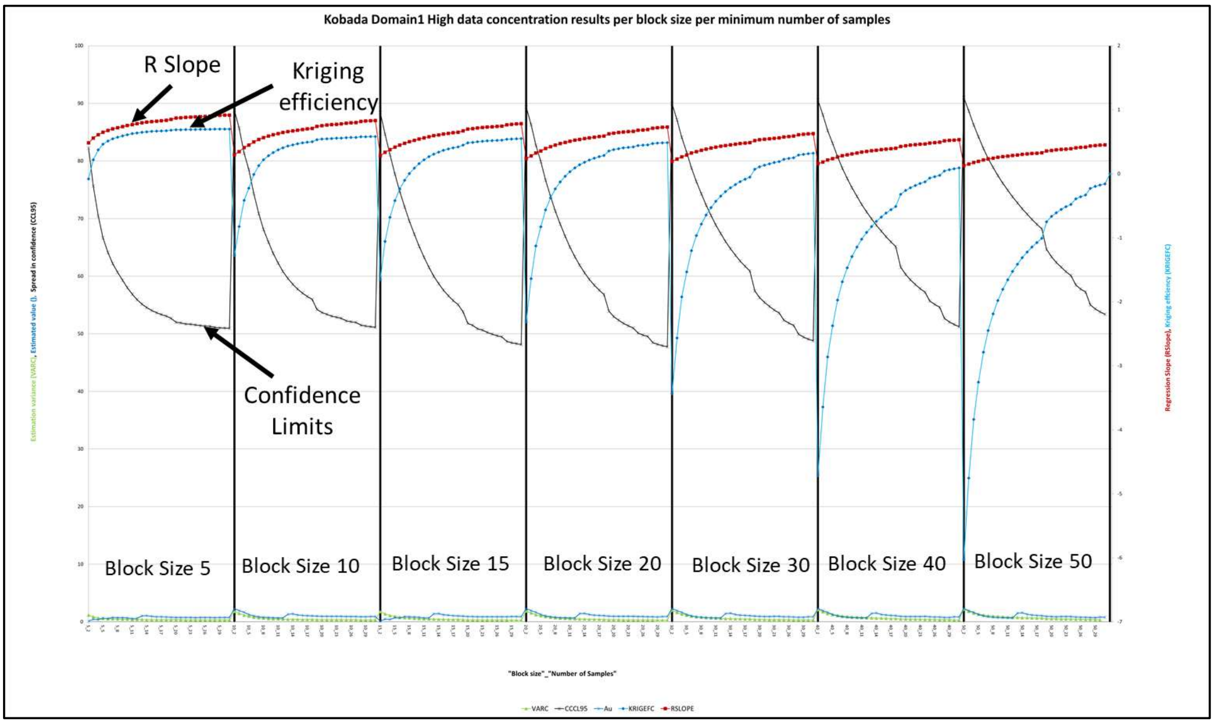


Figure 14.19: Block Size Analysis

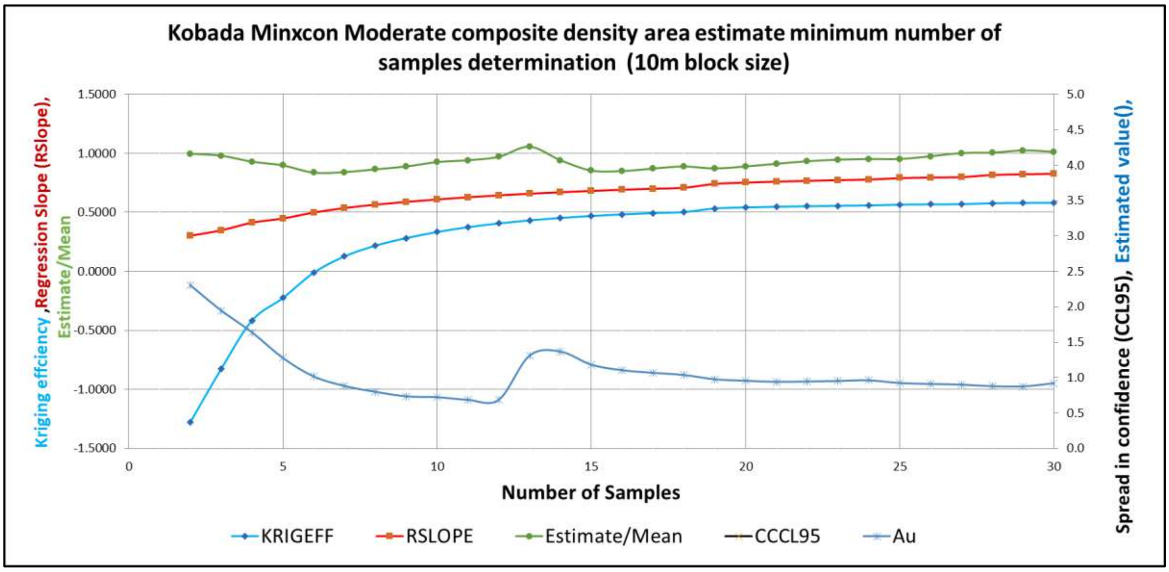


Figure 14.20: Minimum and Maximum Number of Samples in the Estimate

14.1.8 Bulk Densities

AGG has reported a total of 1,907 bulk density records by means of the Archimedes submersion method, comprising 1,795 samples taken from fresh rock, 24 from transition, 24 from oxide, and 7 from the laterite material. Only 104 density records were available to Minxcon in excel format; however, a check of the densities showed that they corresponded well with the average densities applied to the previous estimation. Minxcon applied the following base densities, which were the average densities reported by AGG:

- Laterite: 2.02 t/m^3
- Oxide: 1.85 t/m^3
- Transition: 2.1 t/m^3
- Fresh: 2.65 t/m^3

More density measurements are planned for the laterite, oxide, and transition zones; therefore, the focus of the density sample exercise should be these zones.

14.1.9 Block Model Creation

A Mineral Resource block model, which included all the domains and weathering codes, was created in Datamine Studio RM. The block model was based on the KNA and had a parent estimation cell of 5 m × 10 m × 10 m in the x, y, and z directions. The origin of the mining model was set at the southeast to ensure that it could be included in the model boundary. Table 14.6 shows the block model extent and origin.

Table 14.6: Block Model Extent and Origin

Direction	Origin	Maximum	Cell Size	No. of Cells
	m	m	m	
X	543,000	549,060	5	1,212
Y	1,286,000	1,294,040	10	804
Z	−10	500	10	51

14.1.10 Ordinary Kriging

The grade estimation was conducted using Ordinary Kriging (OK) based on the variograms and the grade estimation was done per structural domain. Additional estimations of Nearest Neighbour and Inverse Distance Squared were used as a check for the OK estimate. Only points falling within the grade halo were considered for the estimation.

Sample search parameters were based on the variograms per domain, and the minimum and maximum number of samples were selected based on the KNA. The second search range was set to 1.5 times the range, and the third search to 3 times the range. An additional parameter on the search was the use of MAXKEY, which limits the number of samples per drillhole to three samples. Limiting the samples per drillhole was done to limit the "smearing" of grade.

The resulting estimation is shown in Figure 14.21 which is an orthographic view of the resource model filtered at 0.35 g/t. Figure 14.22 shows the Far North orebody, also filtered at 0.35 g/t.

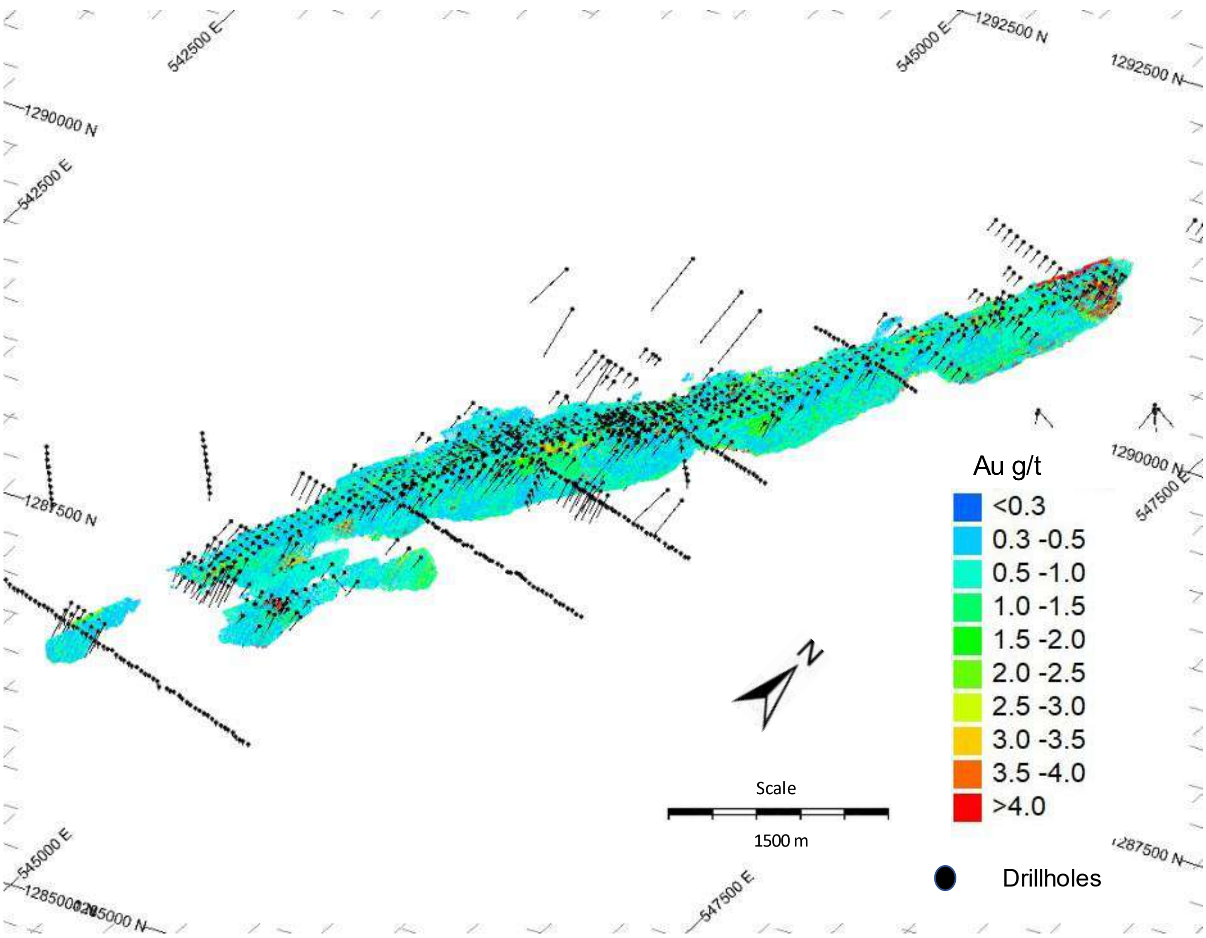


Figure 14.21: Orthographic View of the Mineral Resource Estimation of the Main Orebody (Domains 1 to 4)

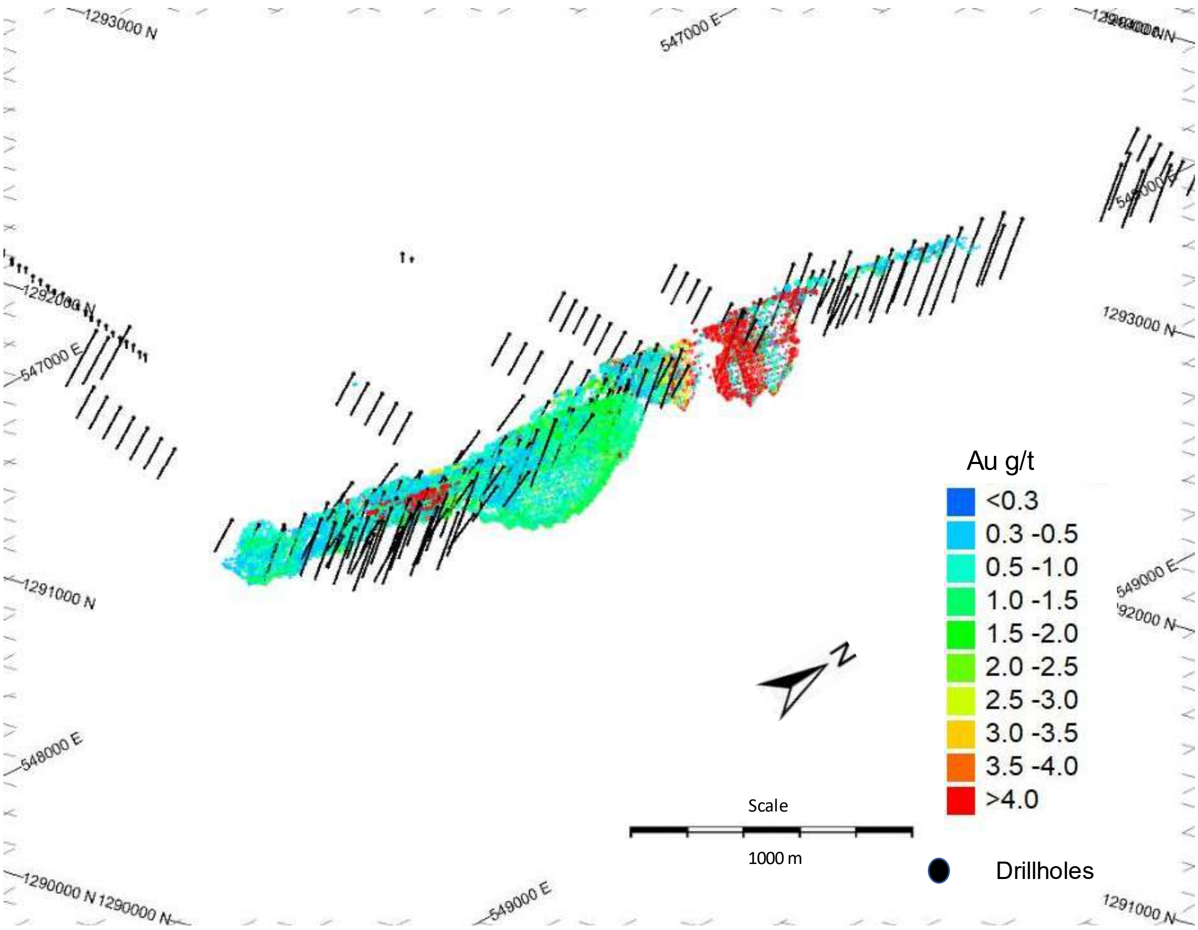


Figure 14.22: Orthographic View of the Mineral Resource Estimation of the Far North Orebody (Domain 5)

14.1.11 Sections

The following sections show the grade distribution, at a 0.35 g/t cut-off, across the orebody. The Mineral Resource is declared only within the Mineral Resource pit although the estimation extends below the Mineral Resource pit floor. Figure 14.23 shows Sections A, B and C, which are in Domains 1, 2 and 3, respectively (see Figure 14.24, Figure 14.25 and Figure 14.26).

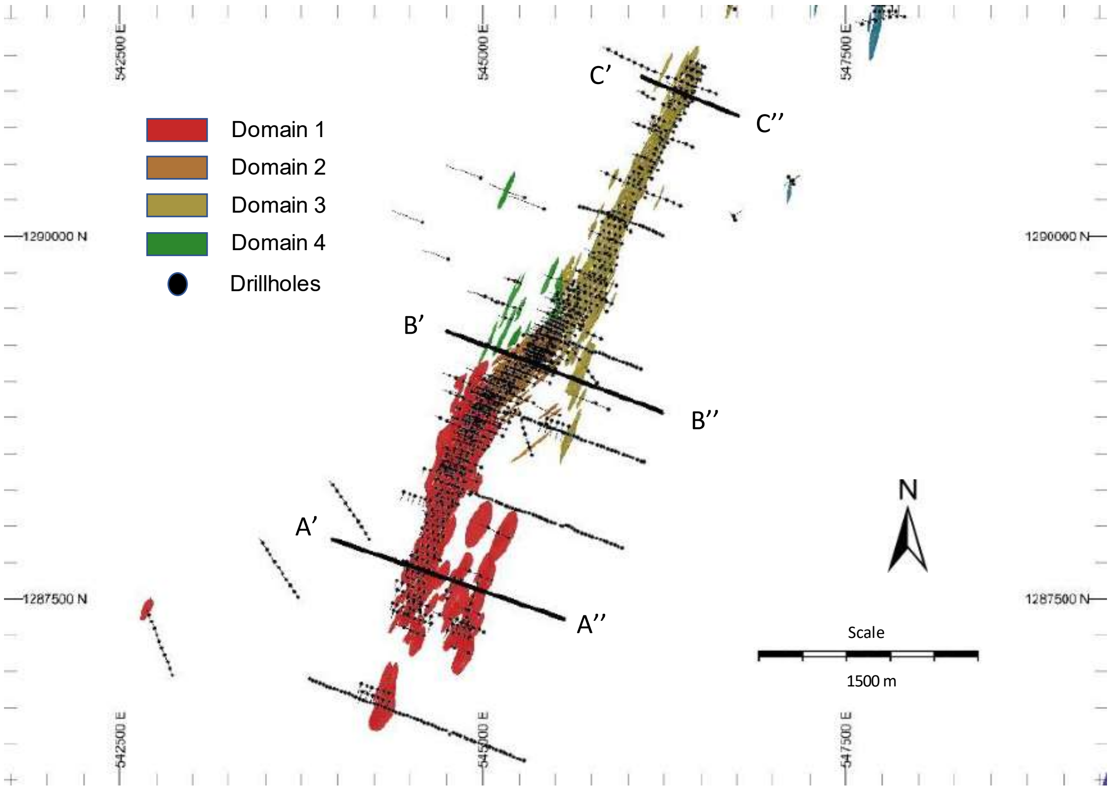


Figure 14.23: Plan of the Sections through Domains 1, 2 and 3

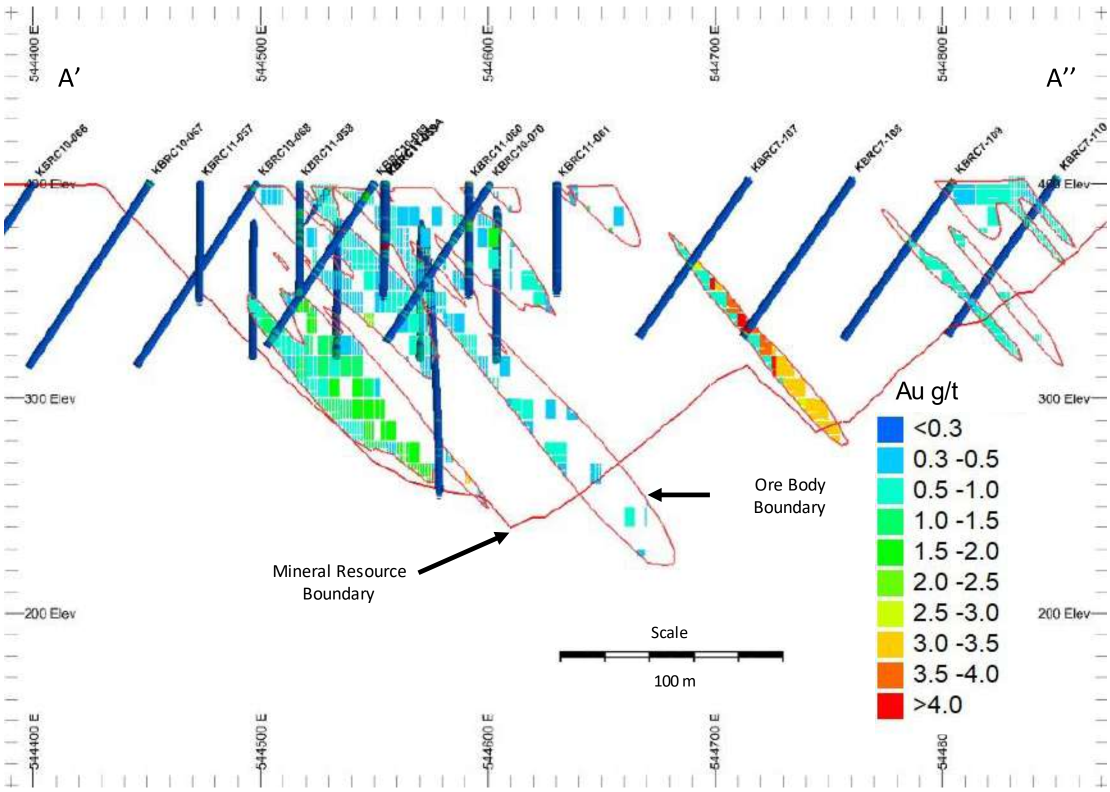


Figure 14.24: Section A through Domain 1

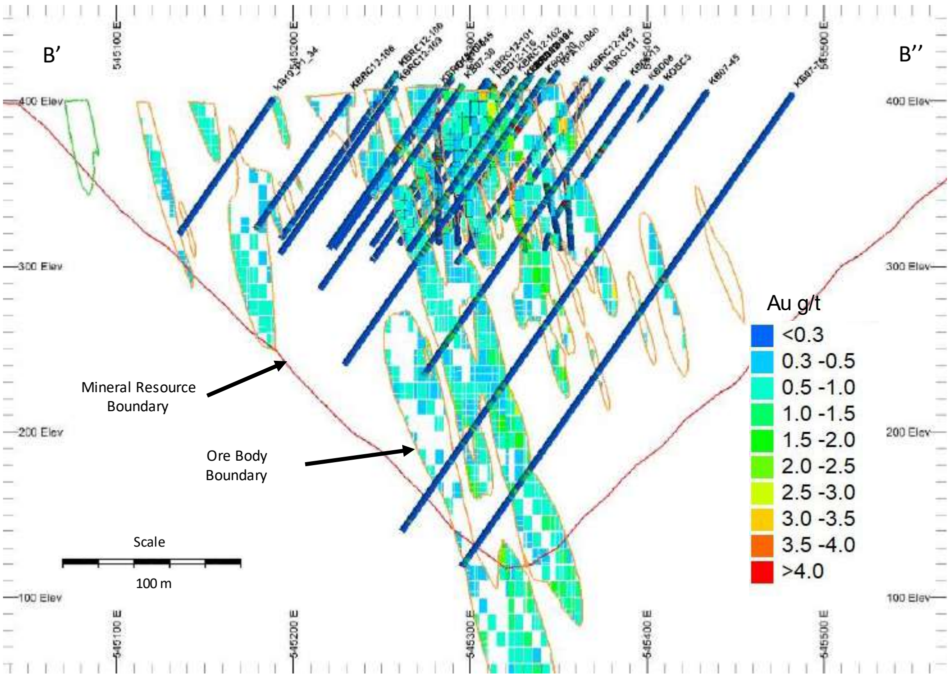


Figure 14.25: Section B through Domain 2

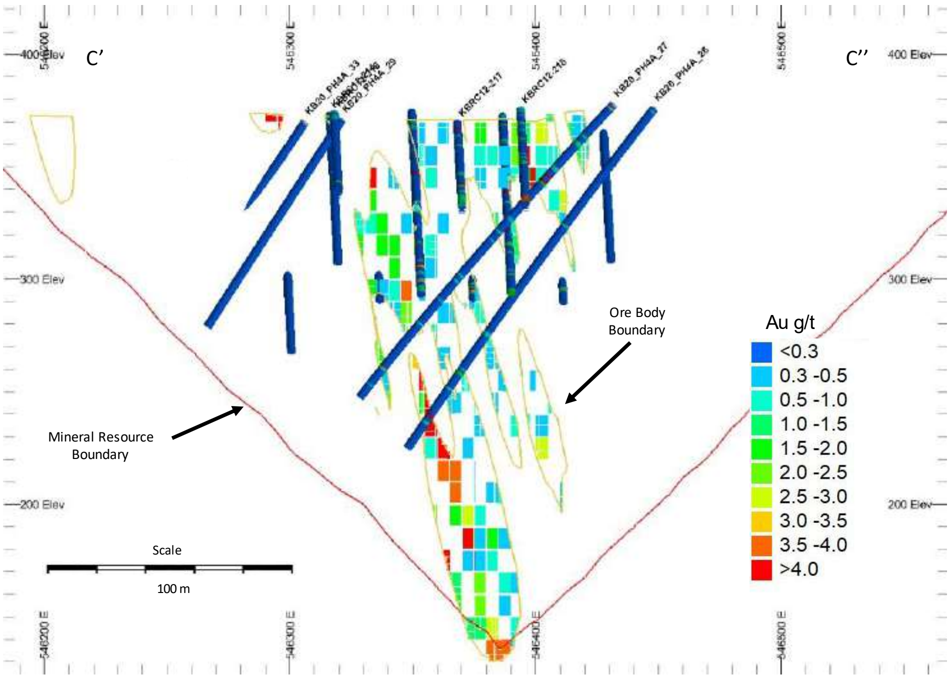


Figure 14.26: Section C through Domain 3

14.1.12 Block Model Validation

The block model estimates were validated by several checks. A visual check was conducted on the model on a section-by-section basis by reviewing the estimated block model grade versus the drillholes. Swath plot analyses were conducted from west to east and south to north, comparing the estimated grade of the model per swath with the drillhole assay value per swath. In general, the swath analyses show a good agreement of the estimated grades and the drillhole grades. The swath slices and swath analyses for Domains 1 to 3 are shown in Figure 14.27 to Figure 14.32.

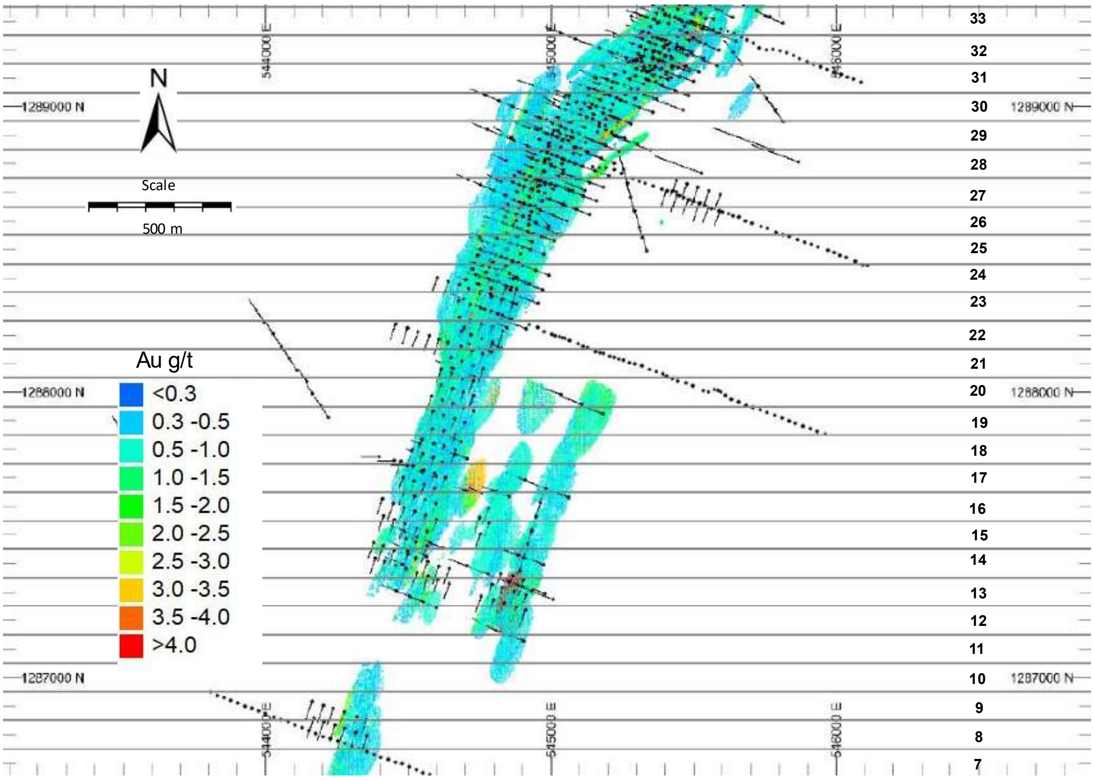


Figure 14.27: 100 m Swath Slices from South to North for Domain 1

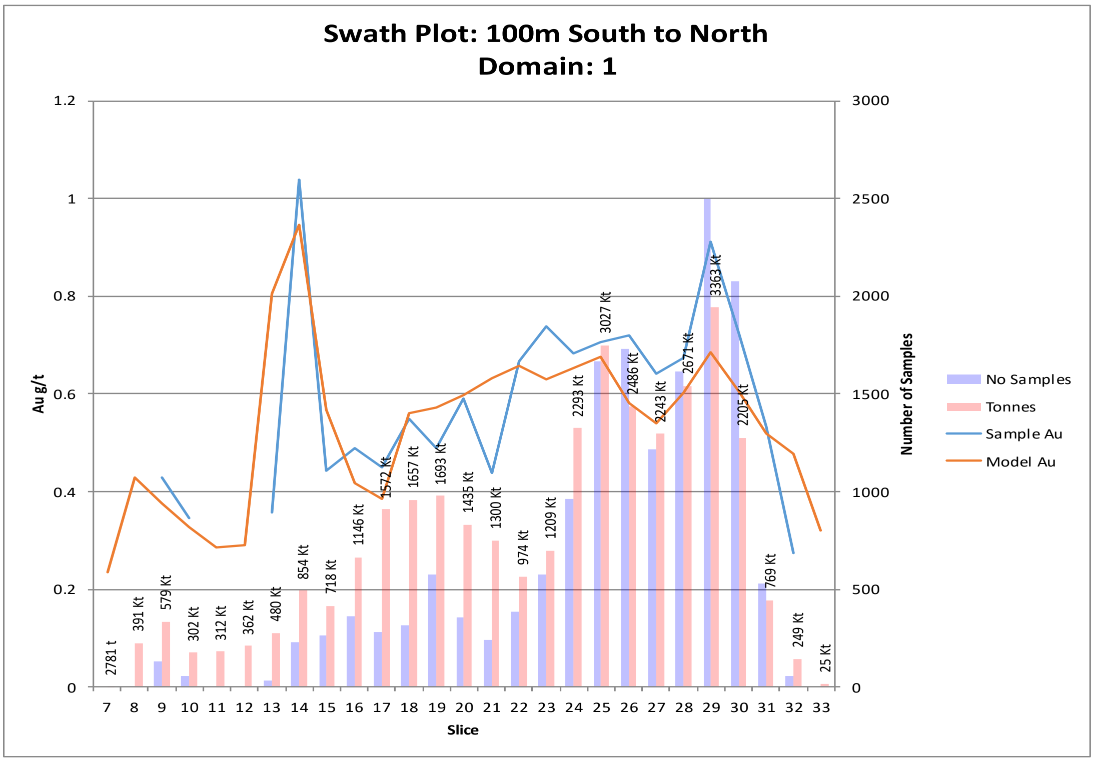


Figure 14.28: Swath Analysis from South to North at 100 m Bin Size Domain 1

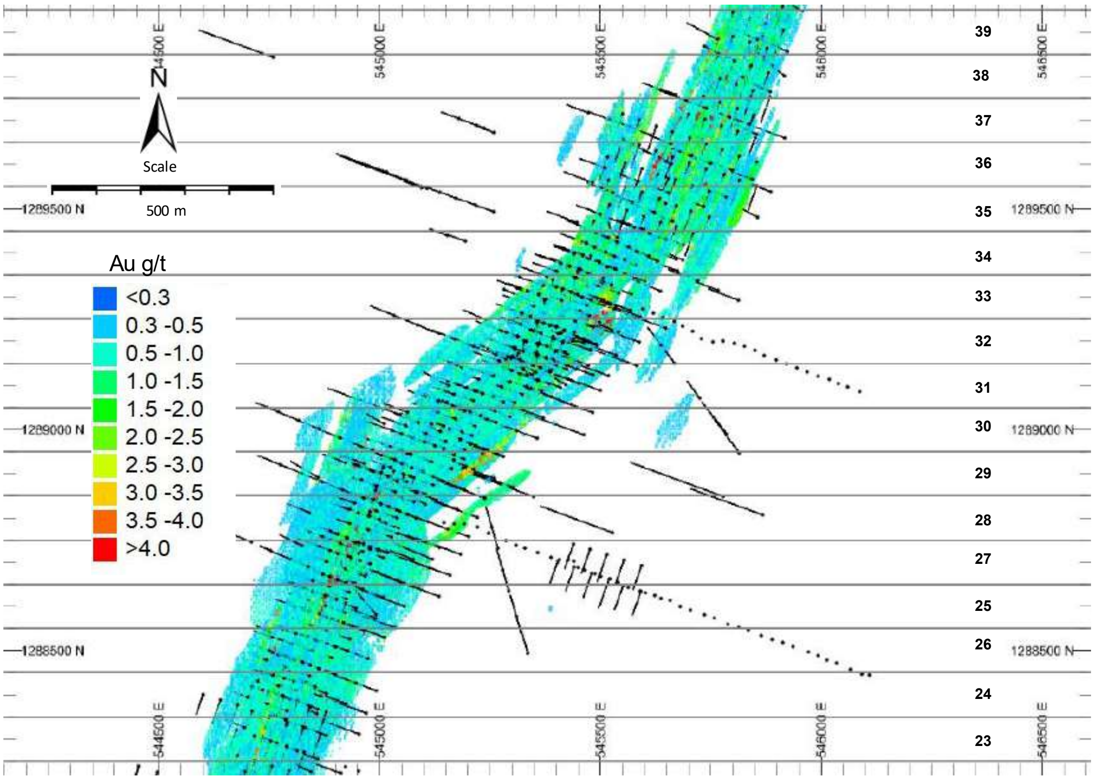


Figure 14.29: 100 m Swath Slices from South to North for Domain 2

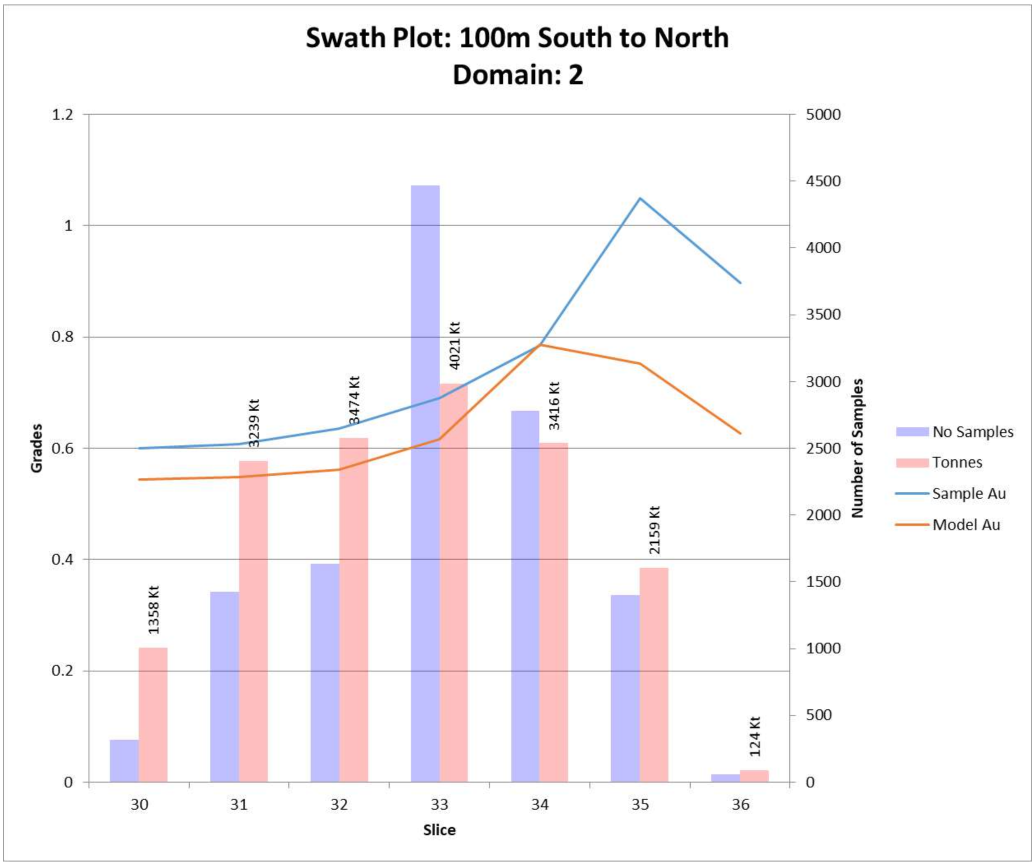


Figure 14.30: Swath Analysis from South to North at 100 m Bin Size for Domain 2

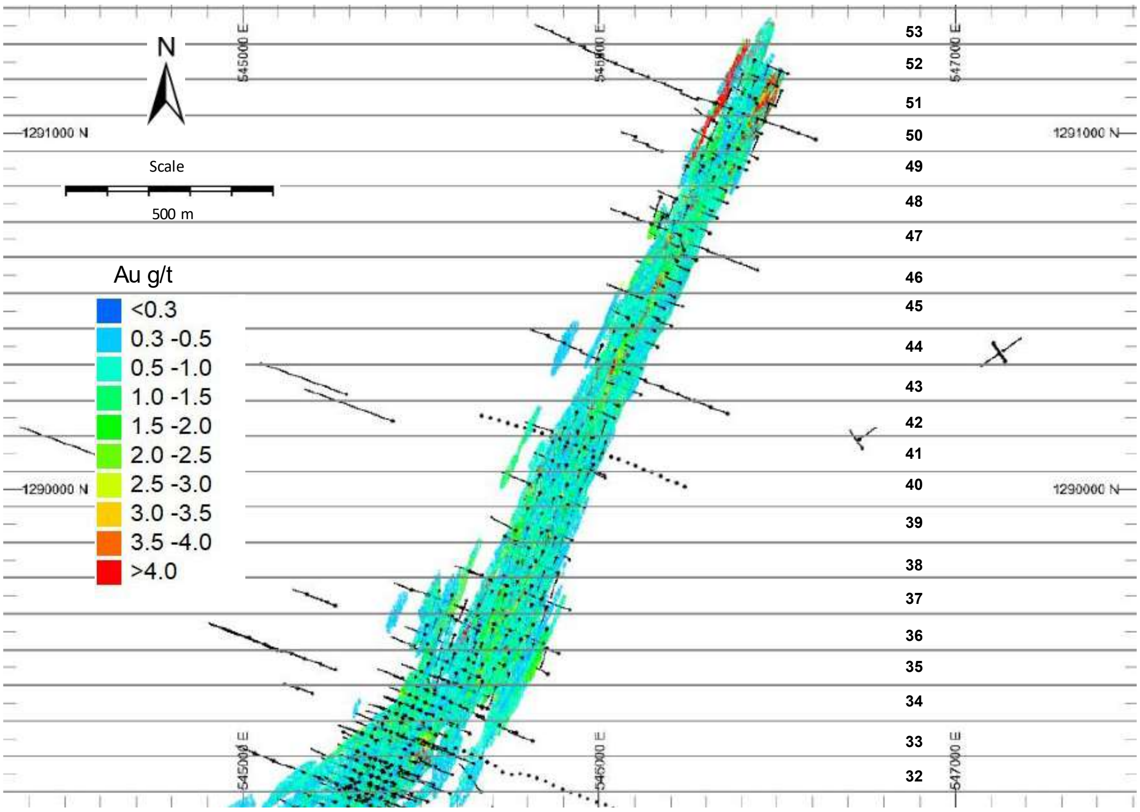


Figure 14.31 : 100 m Swath Slices from South to North for Domain 3

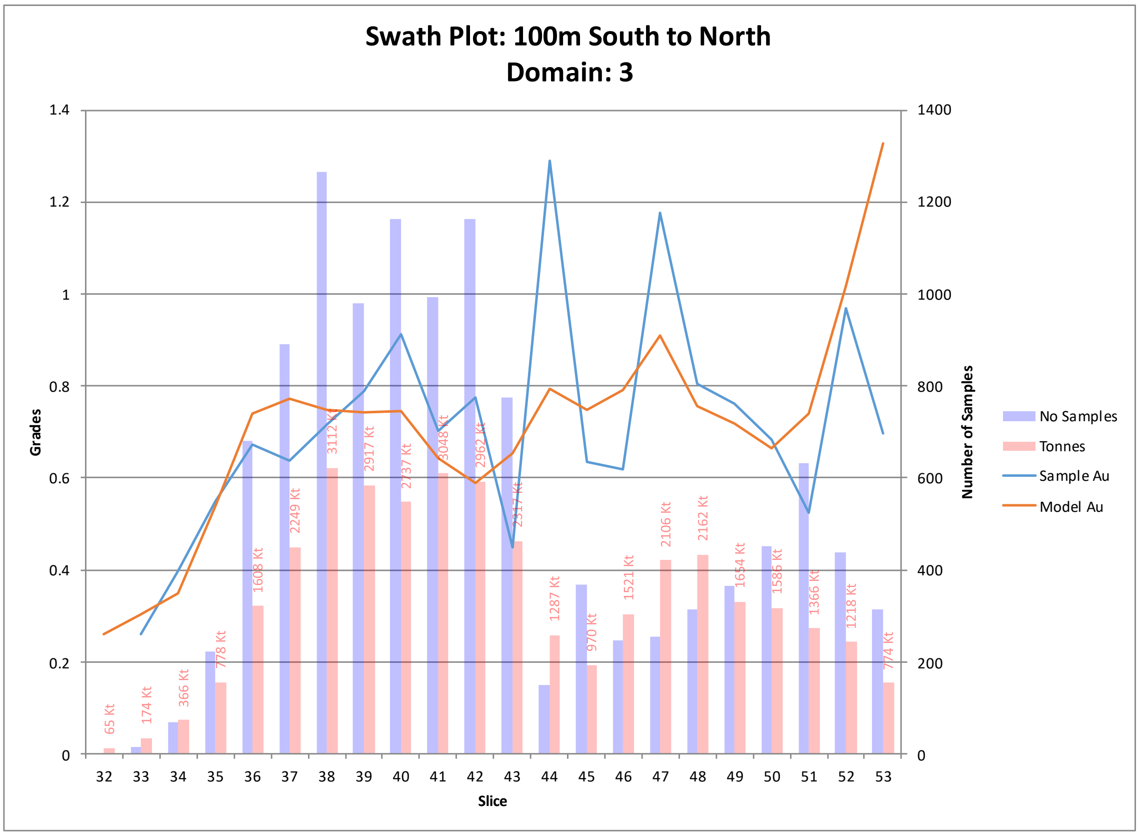


Figure 14.32: Swath Analysis from South to North at 100 m Bin Size for Domain 3

14.1.13 Mineral Resource Classification

The Mineral Resource classification uses the number of samples, number of drillholes, and the range of the variograms to categorise the Mineral Resource. The Measured Mineral Resource is based on the standard variogram range and a minimum of 19 samples from a minimum of 9 drillholes. The Indicated Mineral Resource is based on 1.5 times the variogram range and a minimum of 13 samples from at least 6 drillholes. The Inferred Mineral Resource is based on a maximum of 3 times the variogram range and a minimum of 4 samples from at least 2 drillholes. All the estimates not classified were kept as exploration target values only. Figure 14.33 shows the classification of the Main Orebody, which includes Domains 1 to 4. Figure 14.34 highlights the classification in sections. Domain 5 (see Figure 14.35) was categorised as Inferred as only RC drilling and no DD drilling has been undertaken in this area, and the geology in this domain is uncertain.

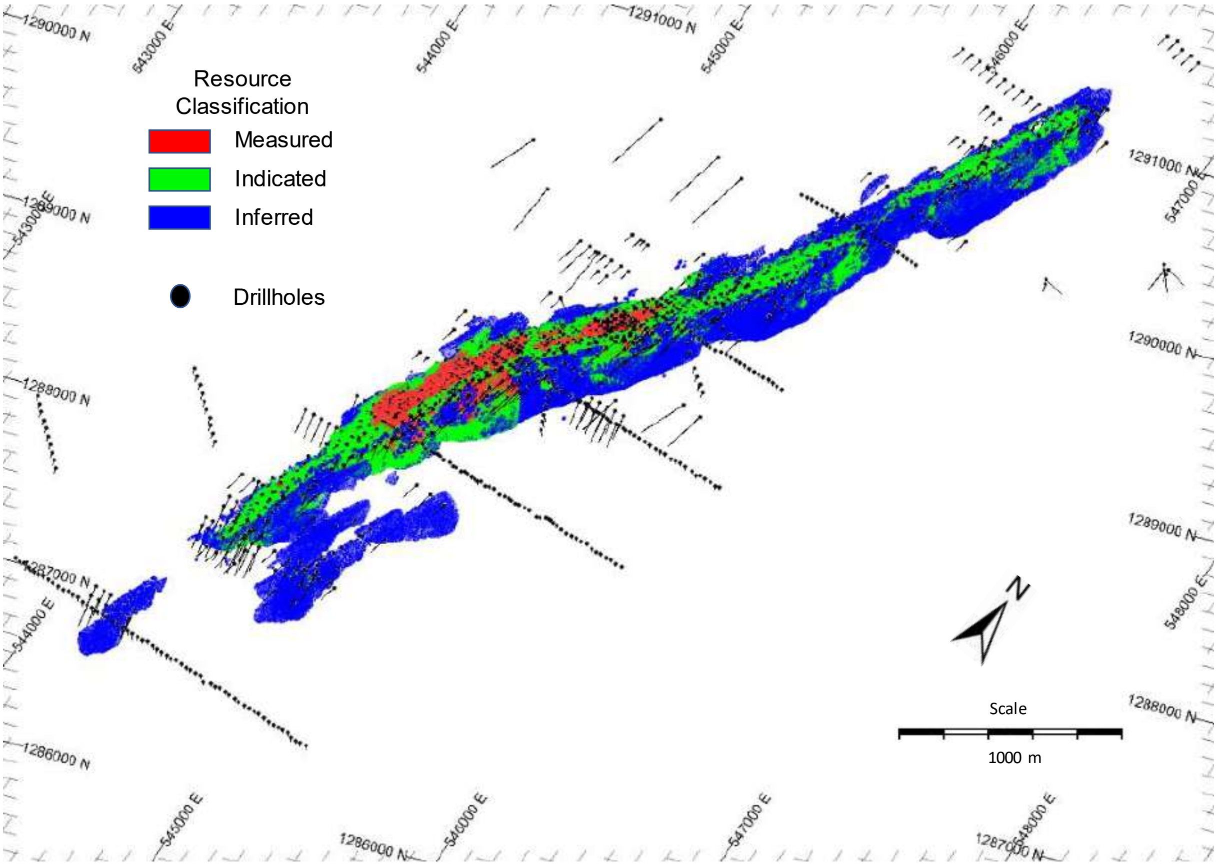


Figure 14.33: Mineral Resource Classification of the Main Orebody (Domains 1 to 4)

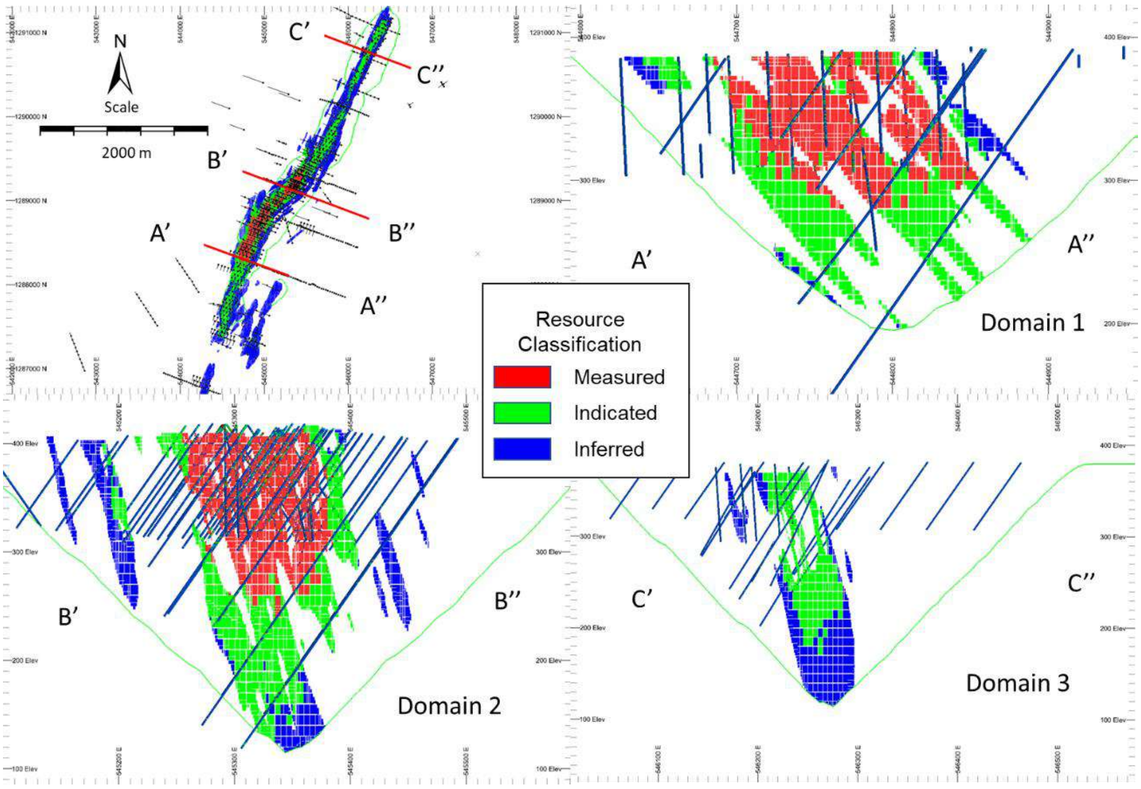


Figure 14.34: Section Views of the Mineral Resource Classification of the Main Orebody (Domains 1 to 3)

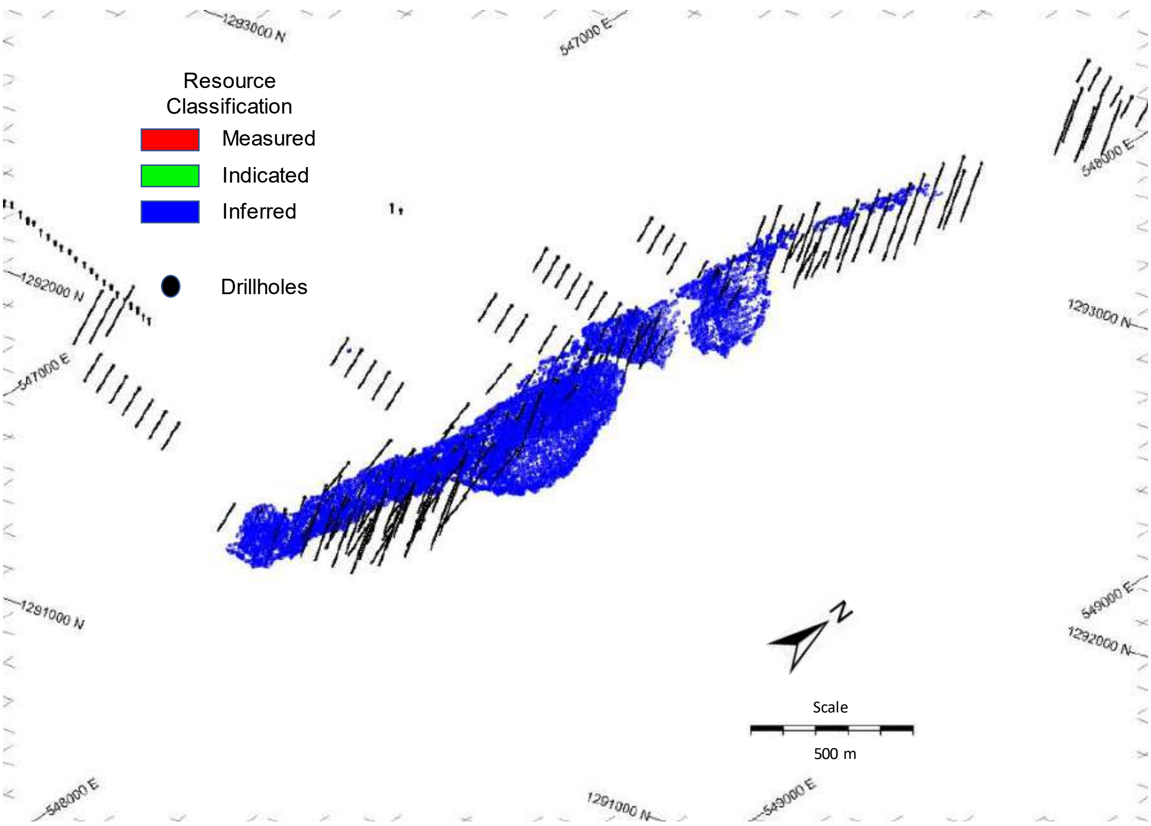


Figure 14.35: Mineral Resource Classification of the Far North Orebody (Domain 5)

14.1.14 Reasonable Prospects of Eventual Economic Extraction (RPEEE)

Two parameters for RPEEE have been applied to the Kobada Gold Project. The first is to test which blocks could be reasonably extracted by means of open-pit mining within a period of approximately 10 to 15 years utilising optimistic but realistic parameters. The second is the application of a cut-off grade based on the same cost and gold price parameters utilised in the Mineral Resource pit.

14.1.14.1 Pit Optimisation

The Mineral Resource models were investigated using NPV Scheduler to determine the Mineral Resource pit based on the criteria in Table 14.7. The final declared Mineral Resource consists only of estimated blocks within the optimised Mineral Resource pit. The Mineral Resource pit and limits are shown in Figure 14.36.

Table 14.7: Parameters Utilised in the Determination of the Mineral Resource Pit and Cut-off Grade

Costing	Unit	Assumption	Comment
Au Price	US$	1,800	90th percentile since 1980 = US$1,650/oz
Processing Cost (Laterite and Saprolite)	US$	8.9	Based on 2020 SENET Study, 9.92 less 10 % = 8.93
Processing Cost (Transition)	US$	11.0	Based on 2020 SENET Study, 12.30 less 10 % = 11.07
Ore Mining (Laterite and Saprolite)	US$	2.25	Based on 2020 SENET, 2.5 less 10 % = 2.25
Ore Mining (Transition and Sulphide)	US$	2.7	Based on 2020 SENET, 3 less 10 % = 2.7
Mining Recovery	%	95	Based on 2020 DRA study
Processing Recovery (Laterite and Saprolite)	%	97	Based on 2020 DRA study
Processing Recovery (Transition)	%	96	New SENET test work on the sulphides
Tonnes per Annum	Mt/a	1.6	Based on 2020 SENET
Slope Angle	Degrees	45	Reserve is 40° but this is increased as sulphides would be more competent ground
Mining Dilution	%	1	Based on DRA study

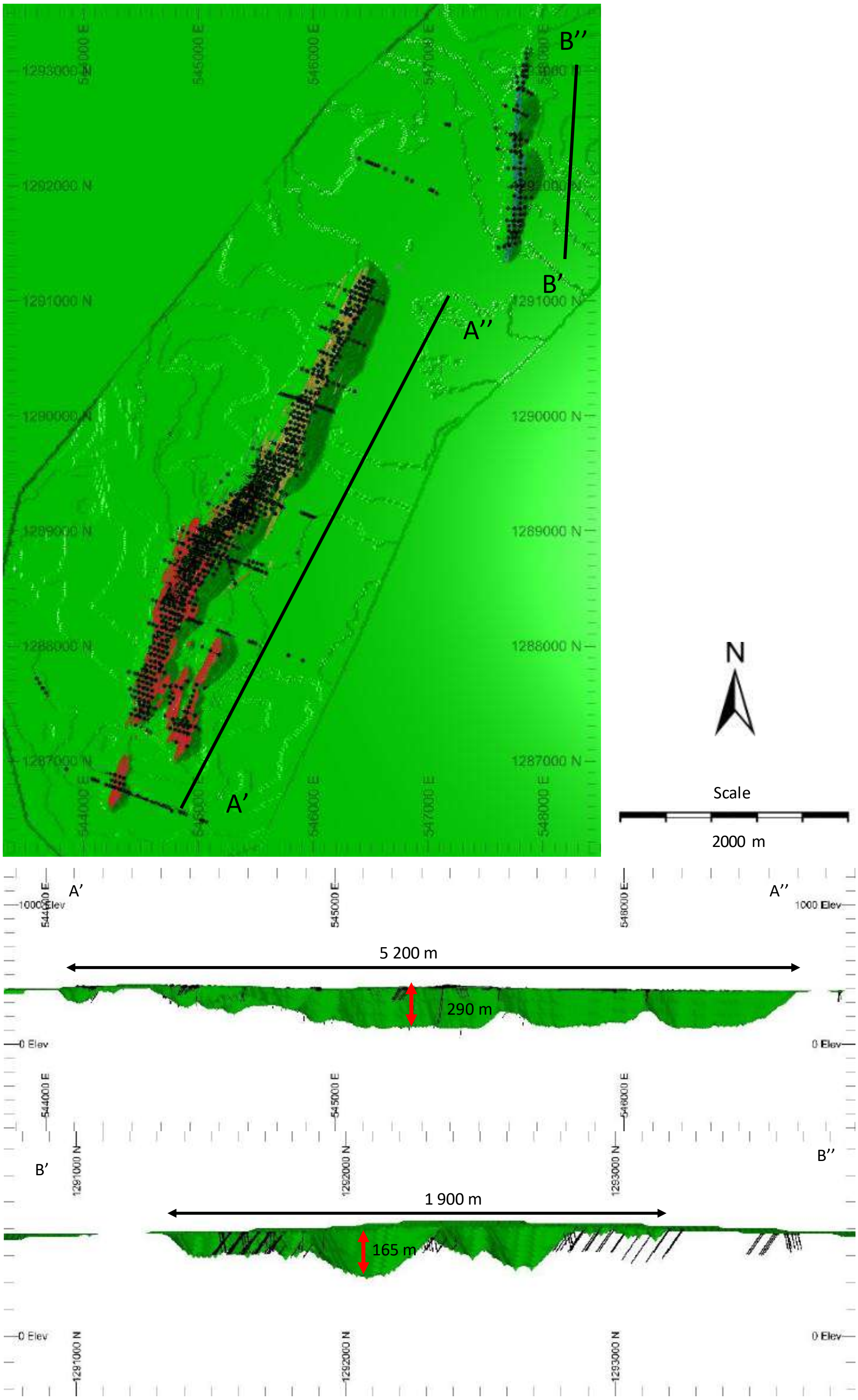


Figure 14.36: Mineral Resource Pit

14.1.14.2 Cut-Off Parameters

The Mineral Resource cut-off grades for the open-pit Mineral Resources at Kobada were based upon reasonable mining considerations. The Mineral Resource cut-off grades should not be considered in terms of Mineral Reserves, but as a long-term view based on realistic operational and processing costs, as well as long-term projected commodity prices.

The parameters used for the declaration and cut-off calculation are gold price, mine call factor, plant recovery factor, and mining and plant cost. The gold price of US$1,650/oz, which is the 90th percentile (red dotted line) of the historical real-term commodity prices since 1980 (see Figure 14.37), was deemed too low based on the recent gold price consistently being above US$1,800/oz. Hence, a gold price of US$1,800/oz was used. The costs and plant recovery factor are based on the current and previous feasibility studies. A summary of the parameters utilised in the calculation of the Mineral Resource cut-off is presented in Table 14.7.

The Mineral Resource pit based on the above parameters calculated an economic resource cut-off grade of 0.25 g/t. It was however decided, based on discussions with AGG, to keep the cut-off grade unchanged at 0.35 g/t.

Figure 14.37: Historical Real Term Gold Price Since 1980

14.1.15 Mineral Resource Statement

The QP for the resource estimate is Mr U Engelmann, BSc (Zoo. & Bot.), BSc Hons (Geol.), Pr.Sci.Nat. and an employee of Minxcon (Pty) Ltd. The open-pit mining cost and design assumptions were developed by the QP and Diane Beverley, BSc Hons (Geol.), Pr.Sci.Nat., MGSSA, also an employee of Minxcon (Pty) Ltd.

According to the *CIM Definition Standards for Mineral Resources and Mineral Reserves* (10 May 2014):

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource. A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

<u>Inferred Mineral Resource</u>

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.

<u>Indicated Mineral Resource</u>

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.

Measured Mineral Resource

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.

The "reasonable prospects for economic extraction" requirement implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off grade considering extraction scenarios and processing recoveries. Minxcon considers that the Kobada Project Mineral Resource meets this requirement because it is suitable for open pit mining and has applied suitable constraints accordingly.

Table 14.8 presents the estimated Mineral Resource at a cut-off grade of 0.35 g/t Au within the Optimised Mineral Resource pit as at 1July 2021.

Table 14.8: Kobada Project Mineral Resources as at 1 July 2021

Mineral Resource Classification	Tonnes	Tonnes Less Geological Loss	Au	Au	Au
	Mt	Mt	g/t	kg	koz
Measured	22.65	21.40	0.83	17,784	572
Indicated	44.81	40.15	0.88	35,425	1,139
Measured and Indicated Total	**67.46**	**61.54**	**0.86**	**53,209**	**1,711**
Inferred Total	**49.60**	**42.03**	**1.06**	**44,564**	**1,433**

NOTES:
1. A Mineral Resource cut-off grade of 0.35 g/t Au was applied.
2. A gold price of US$1,800/oz was used for ultimate optimisation.
3. Columns might not add up due to rounding.
4. Mineral Resources are stated as inclusive of Mineral Reserves.
5. Mineral Resources are reported as total Mineral Resources and are not attributed.
6. Geological losses have been applied.

The Mineral Resource is summarised in Table 14.9, split into oxides and sulphide. The Inferred Mineral Resource is included to give an indication of all the Mineral Resources currently available. This is also shown in the grade tonnage curves in Figure 14.38 and Figure 14.39. Figure 14.38 shows the tonnes (geological losses applied) and grade for all weathering codes between 0 g/t and 3 g/t cut-off to illustrate the sensitivity of the estimated material at low grades. Figure 14.39 shows the tonnages (geological losses applied) and grade between the cut-off of 0 g/t and 3 g/t, excluding the sulphides.

Table 14.9: Kobada Mineral Resources per Weathering Zone as at 1 July 2021

Rock Type	Mineral Resource Classification	Tonnes	Tonnes Less Geological Losses	Au	Au	Au
		Mt	Mt	g/t	kg	koz
Laterite	Measured	0.34	0.33	0.79	258	8
	Indicated	1.31	1.18	0.90	1,062	34
	Measured & Indicated Total	**1.66**	**1.51**	**0.87**	**1,320**	**42**
	Inferred	**1.53**	**1.30**	**1.01**	**1,308**	**42**
Oxide	Measured	12.48	11.73	0.88	10,308	331
	Indicated	18.16	16.16	0.94	15,113	486
	Measured & Indicated Total	**30.64**	**27.89**	**0.91**	**25,421**	**817**
	Inferred	**12.89**	**10.83**	**1.14**	**12,373**	**398**
Transitional	Measured	1.99	1.89	0.84	1,595	51
	Indicated	4.93	4.43	0.89	3,936	127
	Measured & Indicated Total	**6.92**	**6.33**	**0.87**	**5,531**	**178**
	Inferred	**5.41**	**4.60**	**0.95**	**4,345**	**140**
Total Excluding Sulphides	Measured	14.82	13.95	0.87	12,161	391
	Indicated	24.40	21.78	0.92	20,110	647
	Measured & Indicated Total	**39.22**	**35.73**	**0.90**	**32,271**	**1,038**
	Inferred	**19.82**	**16.72**	**1.08**	**18,027**	**580**
Sulphide	Measured	7.84	7.45	0.76	5,623	181
	Indicated	20.41	18.37	0.83	15,315	492
	Measured & Indicated Total	**28.25**	**25.81**	**0.81**	**20,938**	**673**
	Inferred	**29.78**	**25.31**	**1.05**	**26,537**	**853**
Total Including Sulphides	Measured	22.65	21.40	0.83	17,784	572
	Indicated	44.81	40.15	0.88	35,425	1,139
	Measured & Indicated Total	**67.46**	**61.54**	**0.86**	**53,209**	**1,711**
	Inferred	**49.60**	**42.03**	**1.06**	**44,564**	**1,433**

NOTES:
1. Mineral Resource cut-off of 0.35 g/t Au applied.
2. A gold price of USD1,800/oz was used for ultimate optimisation.
3. Columns may not add up due to rounding.
4. Mineral Resources are stated as inclusive of Mineral Reserves.
5. Mineral Resources are reported as total Mineral Resources and are not attributed.
6. Geological losses have been applied.

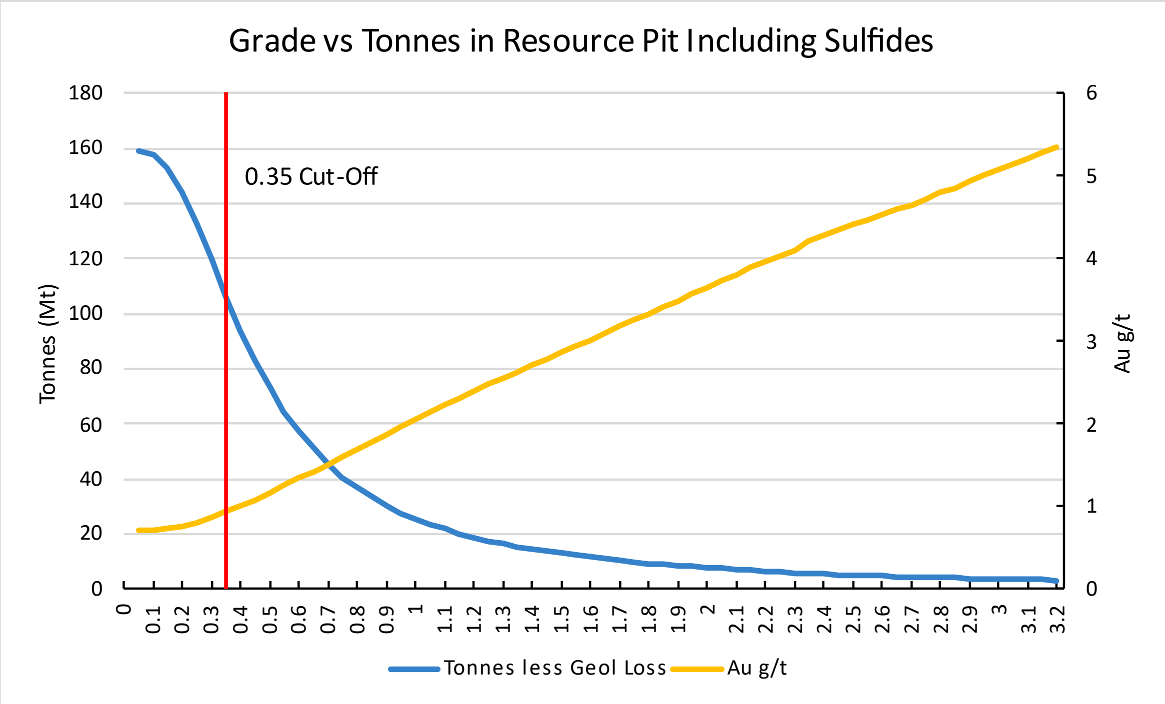


Figure 14.38: Grade Tonnage Curve All Domains between 0 g/t and 3 g/t Cut-off Including Sulphides

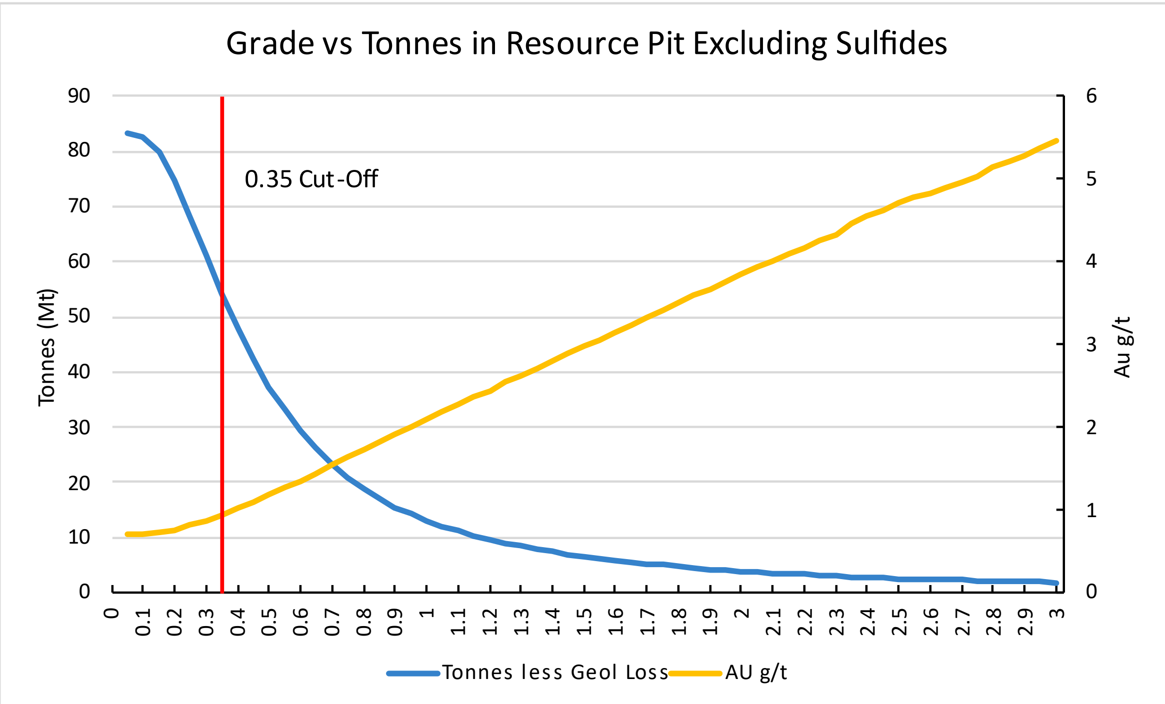


Figure 14.39: Grade Tonnage Curve All Domains between 0 g/t and 3 g/t Cut-off Excluding Sulphides

14.2 DISCLOSURE REQUIREMENTS FOR RESOURCES

All the Mineral Resources have been categorised and reported in compliance with the definitions given in the *CIM Definition Standards for Mineral Resources and Mineral Reserves* (2014). According to the CIM standards, Mineral Resources have been reported in the Measured, Indicated and Inferred Mineral Resource categories. The Inferred Mineral Resources have been reported separately and have not been incorporated with the Measured and Indicated Mineral Resources.

14.3 INDIVIDUAL GRADE OF METALS

The Mineral Resources for gold have been estimated for the Kobada Gold Project. No other metals or minerals have been estimated for the Project.

14.4 FACTORS AFFECTING MINERAL RESOURCE ESTIMATES

No socio-economic, legal, or political modifying factors have been applied in the estimation of the Mineral Resources for the Kobada Gold Project. Minxcon is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, and political or other factors that will materially affect the Mineral Resource estimates.

RPEEE have been applied to the Mineral Resource as detailed in Section 14.1.

15 MINERAL RESERVE ESTIMATES

15.1 INTRODUCTION

Mineral Reserves are based on the Measured and Indicated Resources presented in Section 14 and use FS level engineering designs for the pit and associated operating parameters. This Section presents the parameters and steps used to estimate the Mineral Reserves for the Project.

The terminology used to classify the reserves in this Report is in accordance with National Instrument (NI) 43-101 and the *Canadian Institute of Mining, Metallurgy, and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves* (2014) as well as following the *CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines* (2019). The terminology is summarised below.

15.1.1 Mineral Reserves as Defined by Ni 43-101

Mineral Reserves are sub-divided in order of increasing confidence into Probable Mineral Reserves and Proven Mineral Reserves. A Probable Mineral Reserve has a lower level of confidence than a Proven Mineral Reserve.

A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource demonstrated by at least a Pre-Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.

Mineral Reserves are those parts of Mineral Resources which, after the application of all mining factors, result in an estimated tonnage and grade which, in the opinion of the Qualified Person(s) making the estimates, is the basis of an economically viable Project after taking account of all relevant processing, metallurgical, economic, marketing, legal, environment, socio-economic and governmental factors. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the Mineral Reserves and delivered to the treatment plant or equivalent facility. The term 'Mineral Reserve' need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

Probable Mineral Reserve

A Probable Mineral Reserve is the economically mineable part of an Indicated and, in some circumstances, a Measured Mineral Resource. The confidence applied in the Modifying Factors applying to a Probably Mineral Reserve is lower than that applying to a Proven Mineral Reserve.

The Qualified Person(s) may elect to convert Measured Mineral Resources to Probably Mineral Reserves if the confidence in the Modifying Factors is lower than that applied to a Proven Mineral Reserve. Probable Mineral Reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a Pore-Feasibility Study.

Proven Mineral Reserve

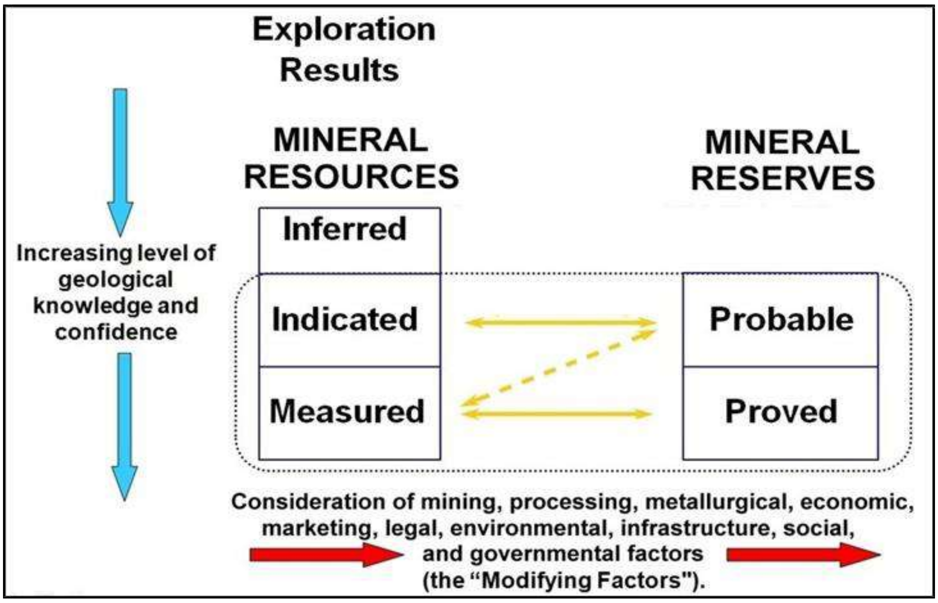
A Proven Mineral Reserve is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors.

Application of the Proven Mineral Reserve category implies that the Qualified Person has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect the potential economic viability of the deposit. Proven Mineral Reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a Pre-Feasibility Study. Within the CIM Definition standards the term Proved Mineral Reserve is an equivalent term to a Proven Mineral Reserve

Modifying Factors

Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.

Figure 15.1 shows the relationship between the Mineral Resource and Mineral Reserve categories.



Source: CRIRSCO International Reporting Template, October 2019

Figure 15.1:Relationship between Mineral reserves and Mineral Resources

15.2 PIT OPTIMISATION

Open-pit optimisation was conducted on the deposit to determine the economic limits of the pit. The optimisation was performed during the initial stage of the Project using initial cost, product (gold) sales price, and pit and plant operating parameters.

The pit optimisation was conducted using GEOVIA Whittle™ software to provide guidance as to the economic pit limit of the Kobada deposit. The optimisation software operates on a net value calculation for all the blocks in the model (i.e., revenue from the sales of products minus the operating cost). For an NPV estimation, the following principal aspects are considered:

$$\boldsymbol{Product\ Tonnage} = \text{Mineralised Tonnage} \times \text{Recovery} \times \text{Feed Grade (Au Grade)}$$

$$\boldsymbol{Revenue} = \text{Product Tonnage} \times \text{Sales Price}$$

$$\boldsymbol{Net\ Value} = \text{Revenue} - (Mining\ Cost + Processing\ Cost + \text{Transportation Cost} + G\&A\ Cost)$$

The optimiser software establishes the pit limits or ultimate pit shell where the revenue from the ore and the waste stripping costs are equal. The result is that the sum of all the blocks contained within this optimal pit shell will report the optimal economic value (if the pit is smaller, some value is left in the ground, and if the pit is bigger, some value is "destroyed" due to the additional stripping).

Once this pit shell is generated, it is used as a guide to design the engineered pit incorporating berms, benches, and haulage roads. This engineered pit will be used for the development of the Mine Schedule to determine the ore quantity and grades mined for each period of the LOM.

Table 15.1 presents the economic parameters used for the Kobada pit optimisation.

Table 15.1: Pit Optimisation Parameters

Description	Unit	Value
Mining Cost (Laterite and Saprolite)	US$/t	2.50
Mining Cost (Transition and Sulphide)	US$/t	3.00
Processing Cost (Laterite and Saprolite) *	US$/t	9.41
Processing Cost (Transition)*	US$/t	12.64
Processing Cost (Sulphide)*	US$/t	15.08
Mining Recovery	%	95.0
Mining Dilution	%	5.0
Process Recovery (Laterite and Saprolite)	%	96.5
Process Recovery (Transition)	%	90.5
Process Recovery (Sulphide)	%	95.4
Gold Price	US$/oz	1,610
Refining Cost	US$/oz	7.59
Royalties	%	3.0
Overall Pit Slope	Degrees	40
* This cost includes the G&A and tailings operating costs.		

15.2.1 Mining Recovery and Dilution

In every mining operation, it is impossible to perfectly separate the mineralised material and the waste material because of the large scale of the mining equipment and the use of drilling and blasting equipment. To account for this, DRA assumes a mining recovery of 95 % and a mining dilution of 5 %.

The original sub-blocked model has been converted to a model containing 5.0 m × 5.0 m × 5.0 m block, with sub-blocks of 5.0 m x 5.0 m x 2.5 m, in the X, Y and Z directions, respectively.

This conversion process introduces an external dilution because a portion of each converted block contained some waste material at the edges of the mineralised zones. The total dilution introduced by the conversion process has been estimated at 2.3 % for ore (see Table 15.2). Therefore, an additional 3% dilution factor has been applied to the reserves to achieve the 5% dilution target.

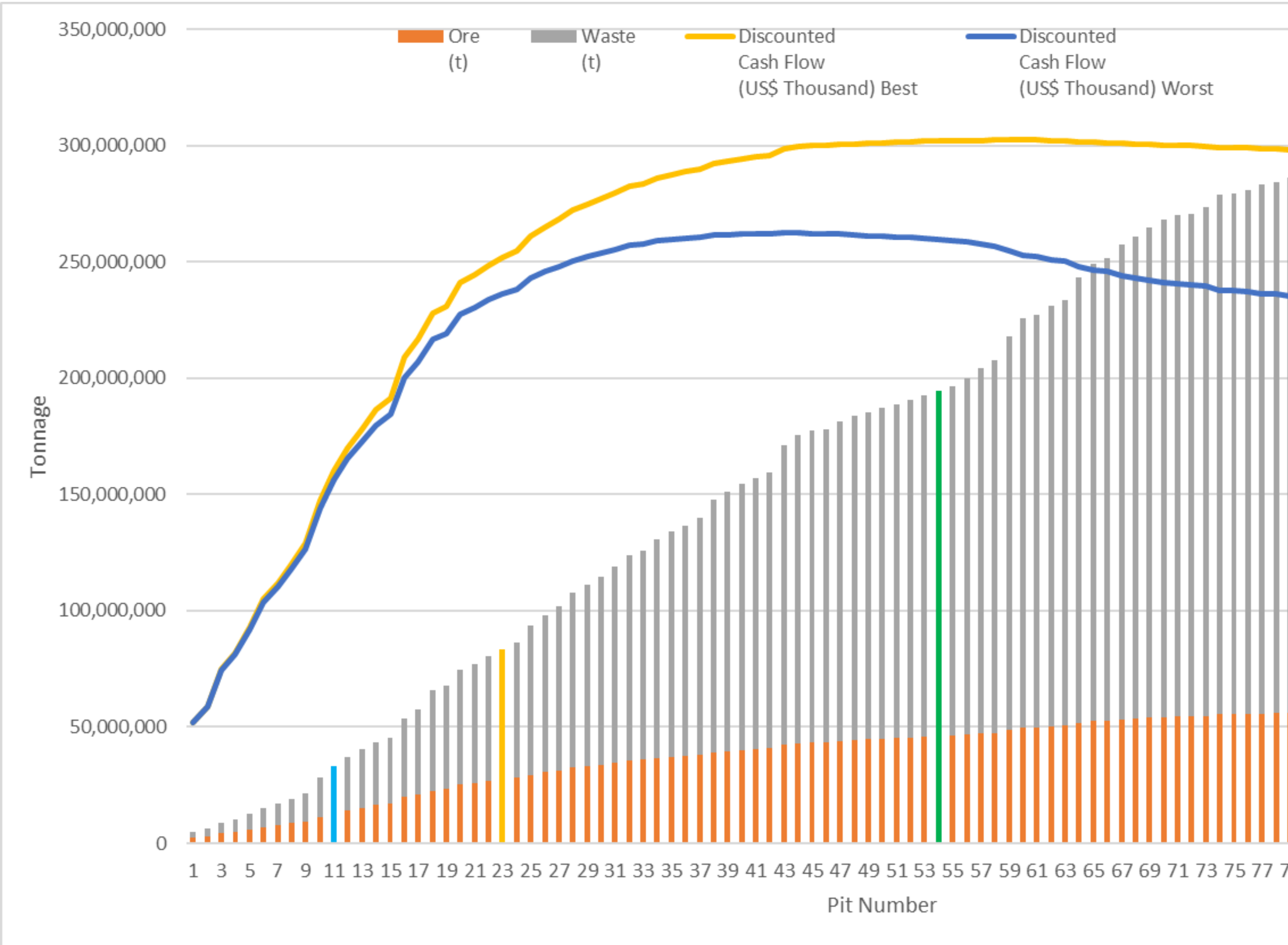
Table 15.2: Dilution Induced by Regularisation

Block Model	Total Ore		Gold
	Million Tonnes	Grade	oz
Original Sub-Blocked Model	67.46	0.86	1 875 779
Converted Model	69.01	0.84	1 857 077
Difference	**+2.3 %**	**−2.3 %**	**−1.0 %**

15.2.2 Pit Optimisation Results

15.2.2.1 Final Pit

The optimal open-pit mining limits were established using Whittle, which uses the Lersch-Grossmann algorithm for pit optimisation. The results of the pit optimisation evaluation conducted on the deposit for varying revenue factors are summarised in Table 15.3 and shown in



Source: DRA 2021

Figure 15.2.

Mining optimisation was conducted over the geological model produced by Minxcon geologists. The NPV sensitivity analysis was used as the main criterion to select the optimal pit in the optimisation software, together with a range of nested pit shells generated so that the most economically viable pit be selected.

Note that the NPV in this optimisation summary does not consider capital costs and is used only as a guide for shell selection and the determination of the mining shapes. The actual NPV of the Project is summarised in the economic analysis section (see Section 22).

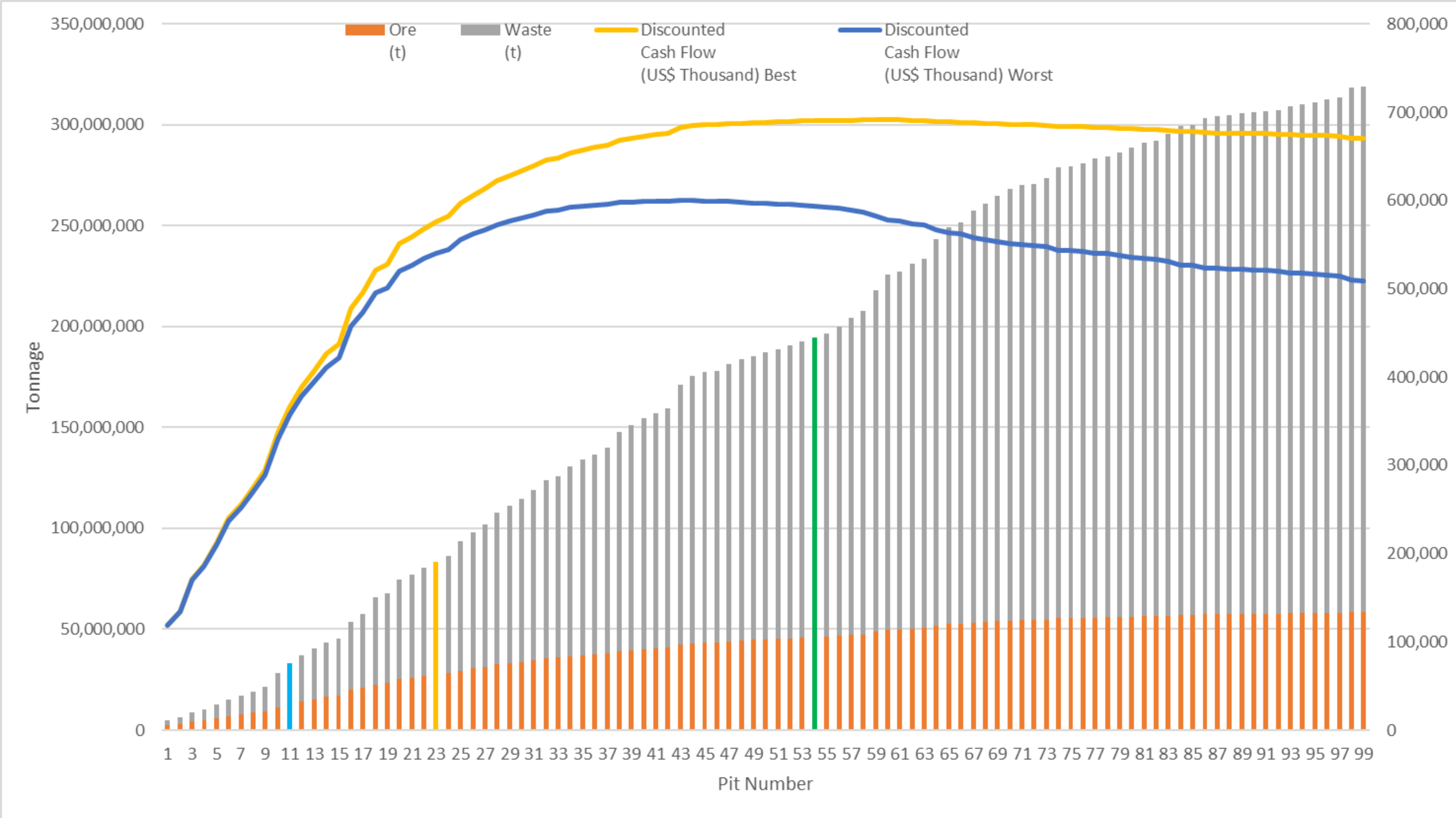
As part of the optimisation process, a stockpiling strategy has been defined in order to feed the process plant with the higher-grade material at the start of the operation and stockpile the lower-grade material to be processed in the later years. The goal, therefore, was to optimise the pit for the high-grade ore and process the low-grade ore contained inside the pit at the end of the LOM.

Table 15.3: MPPE Optimisation Results

Pit Shell	Revenue Factor	Discounted Cash Flow (US$ Thousand) Best	Discounted Cash Flow (US$ Thousand) Worst	Ore (t)	Waste (t)	Grade (g/t)	Pit Shell	Revenue Factor	Discounted Cash Flow (US$ Thousand)	Discounted Cash Flow (US$ Thousand)	Ore (t)	Waste (t)	Grade (g/t)	Pit Shell	Revenue Factor	Discounted Cash Flow (US$ Thousand) Best	Discounted Cash Flow (US$ Thousand) Worst	Ore (t)	Waste (t)	Grade (g/t)
1	0.30	118,586	118,586	2,540,426	2,347,726	1.32	34	0.70	653,976	591,924	36,676,981	94,142,763	0.89	67	1.09	687,991	558,268	53,274,795	204,374,319	0.85
2	0.31	134,526	134,523	3,074,341	3,018,618	1.26	35	0.71	656,944	593,443	37,101,242	96,869,906	0.89	68	1.10	687,528	555,759	53,544,612	207,540,432	0.85
3	0.32	170,739	170,115	4,327,912	4,601,901	1.18	36	0.72	659,936	594,637	37,707,350	99,005,709	0.89	69	1.12	686,910	553,176	53,956,931	210,717,380	0.84
4	0.34	187,232	186,135	4,959,302	5,314,179	1.14	37	0.73	662,681	595,342	38,248,010	101,500,359	0.88	70	1.13	686,279	551,003	54,309,736	213,893,912	0.84
5	0.35	212,401	210,524	5,921,845	6,849,423	1.11	38	0.74	668,503	597,601	39,206,047	108,701,304	0.89	71	1.14	685,929	549,713	54,481,947	215,606,943	0.84
6	0.36	239,790	237,117	7,053,963	8,209,514	1.08	39	0.76	670,857	598,380	39,605,756	111,742,183	0.89	72	1.15	685,796	549,324	54,555,755	216,106,854	0.84
7	0.37	255,396	252,069	7,711,795	9,200,518	1.06	40	0.77	673,163	598,936	40,077,851	114,733,895	0.89	73	1.16	685,209	547,407	54,821,219	218,536,627	0.84
8	0.38	274,705	270,323	8,609,410	10,555,086	1.04	41	0.78	674,980	599,135	40,585,765	116,528,000	0.88	74	1.18	683,964	543,832	55,368,481	223,337,888	0.84
9	0.40	294,293	289,059	9,493,896	12,194,098	1.03	42	0.79	676,268	599,171	40,896,590	118,627,611	0.88	75	1.19	683,743	543,186	55,479,951	224,006,261	0.84
10	0.41	337,102	329,881	11,357,179	17,080,594	1.02	43	0.80	682,338	599,800	42,428,501	128,781,637	0.88	76	1.20	683,362	542,012	55,625,834	225,405,657	0.84
11	0.42	365,725	356,910	12,766,768	20,497,715	1.01	44	0.82	684,453	599,948	43,054,662	132,285,159	0.88	77	1.21	682,794	540,080	55,791,686	227,713,815	0.84
12	0.43	388,844	378,098	14,074,263	22,914,220	0.99	45	0.83	685,516	599,383	43,470,774	134,062,798	0.88	78	1.22	682,624	539,549	55,848,316	228,301,175	0.84
13	0.44	407,856	395,063	15,176,790	25,275,290	0.98	46	0.84	685,786	599,233	43,576,922	134,597,765	0.88	79	1.24	681,987	537,683	56,062,338	230,332,950	0.84
14	0.46	426,037	411,189	16,401,230	26,782,588	0.96	47	0.85	686,943	598,594	44,015,862	137,359,558	0.88	80	1.25	681,363	535,984	56,255,608	232,327,487	0.84
15	0.47	437,877	421,409	17,175,829	28,192,238	0.96	48	0.86	687,697	598,033	44,338,281	139,273,443	0.87	81	1.26	680,558	534,027	56,499,772	234,732,336	0.84
16	0.48	477,415	457,616	19,753,970	33,856,959	0.94	49	0.88	688,193	597,405	44,620,005	140,575,683	0.87	82	1.27	680,291	533,294	56,569,954	235,604,300	0.84
17	0.49	495,690	473,588	21,074,408	36,688,817	0.94	50	0.89	688,781	596,466	44,979,286	142,197,056	0.87	83	1.28	679,330	530,690	56,809,949	238,673,540	0.84
18	0.50	520,986	495,508	22,657,427	42,984,200	0.94	51	0.90	689,198	595,987	45,248,696	143,432,764	0.87	84	1.30	678,071	526,940	57,147,988	242,274,936	0.84
19	0.52	528,317	501,450	23,255,361	44,363,780	0.94	52	0.91	689,594	595,265	45,483,902	145,250,637	0.87	85	1.31	677,851	526,364	57,196,702	242,927,819	0.84
20	0.53	551,220	520,448	25,174,433	49,448,358	0.93	53	0.92	689,988	594,256	45,809,197	146,980,405	0.87	86	1.32	676,721	523,715	57,488,359	245,828,299	0.83
21	0.54	558,710	526,579	25,837,330	51,153,938	0.92	54	0.94	690,274	593,485	46,075,597	148,703,252	0.87	87	1.33	676,462	523,040	57,543,998	246,550,163	0.83
22	0.55	568,057	533,932	26,679,838	54,020,566	0.92	55	0.95	690,460	592,749	46,266,835	150,195,134	0.87	88	1.34	676,199	522,147	57,576,958	247,454,888	0.83
23	0.56	575,399	539,585	27,349,846	56,253,216	0.92	56	0.96	690,736	591,416	46,586,695	153,124,474	0.87	89	1.36	675,974	521,703	57,639,169	247,925,964	0.83
24	0.58	582,443	544,839	28,055,640	58,427,139	0.91	57	0.97	691,002	588,933	47,146,973	157,257,594	0.86	90	1.37	675,758	521,092	57,683,355	248,487,636	0.83
25	0.59	596,421	555,368	29,371,211	64,056,323	0.91	58	0.98	691,124	586,991	47,532,627	160,103,764	0.86	91	1.38	675,589	520,581	57,722,912	248,898,427	0.83
26	0.60	606,244	561,925	30,493,568	67,501,519	0.91	59	1.00	691,251	582,856	48,553,655	169,348,233	0.86	92	1.39	675,319	519,832	57,776,078	249,577,854	0.83
27	0.61	613,305	566,850	31,274,733	70,688,735	0.90	60	1.01	691,139	577,402	49,741,873	175,905,256	0.86	93	1.40	674,611	518,096	57,897,548	251,432,944	0.83
28	0.62	622,679	572,729	32,436,634	75,107,755	0.90	61	1.02	691,089	576,483	49,901,554	177,376,409	0.86	94	1.42	674,347	517,282	57,947,379	252,113,343	0.83
29	0.64	628,597	576,803	33,169,176	78,018,814	0.90	62	1.03	690,858	574,030	50,342,401	180,957,320	0.85	95	1.43	673,949	516,432	58,025,698	253,002,301	0.83
30	0.65	633,723	580,327	33,801,764	80,696,088	0.90	63	1.04	690,655	572,255	50,670,469	182,978,394	0.85	96	1.44	673,352	515,079	58,141,340	254,344,787	0.83
31	0.66	639,045	583,414	34,430,088	84,466,267	0.90	64	1.06	689,649	566,582	51,912,687	191,616,249	0.85	97	1.45	672,937	513,986	58,231,201	255,199,410	0.83
32	0.67	646,097	587,700	35,454,746	88,569,338	0.89	65	1.07	689,051	563,509	52,477,379	196,862,973	0.85	98	1.46	670,781	509,599	58,612,026	259,981,198	0.83
33	0.68	648,675	589,068	35,894,499	90,042,383	0.89	66	1.08	688,764	562,016	52,739,981	198,910,074	0.85	99	1.48	670,555	509,112	58,648,472	260,506,962	0.83

(blue)	Pit shell for Phase 1 pit
(orange)	Pit shell for Phase 2 pit
(green)	Pit shell for ultimate pit limit

Source: DRA 2021



Source: DRA 2021

Figure 15.2: MPPE Optimisation Results

15.2.2.2 Sensitivities

During the optimisation process, additional sensitivity runs were examined to determine the sensitivity of the resource to certain parameters. The following main parameters were changed as part of the sensitivity runs:

- Gold price: from 1,450 USD/oz to 1,750 USD/oz
- Mining Cost: ±10%
- Processing Cost: ±10%

Table 15.4 shows the results of the sensitivity runs.

Table 15.4: Kobada Sensitivity Results

Description	NPV (%)	Tonnes of Ore (%)	Grade (%)	Contained Gold (%)
Selling Price				
US$1,450/oz	83	96	103	100
US$1,500/oz	91	98	101	100
US$1,550/oz	94	99	101	100
US$1,610/oz (base case)	**100**	**100**	**100**	**100**
US$1,650/oz	104	101	99	100
US$1,700/oz	110	102	99	100
US$1,750/oz	115	102	98	100
Mine OPEX				
Mine OPEX −10 %	103	100	100	100
Mine OPEX (base case)	**100**	**100**	**100**	**100**
Mine OPEX +10 %	97	100	100	100
Process OPEX				
Process OPEX −10 %	104	103	98	101
Process OPEX (base case)	**100**	**100**	**100**	**100**
Process OPEX +10 %	96	97	102	99

These results show that the Kobada reserves are relatively robust to withstand changes in the optimisation parameters; therefore, even if the commodity price fluctuates or cost parameters change, the defined engineered pit will not vary drastically.

15.2.3 Cut-Off Grade

The COG is used to determine whether the material being mined will generate a profit after the mining, processing, and administrative costs have been paid. Material that is mined below the COG is sent to the waste dump. The COG has been calculated according to the following formula:

$$COG = \frac{Mining\ Cost + Processing\ Cost + G\&A\ cost}{Sales\ Price \times Mill\ Recovery}$$

The marginal COG has been defined as the total ore cost excluding mining costs (but including a rehandling cost) divided by the net recovered gold price. There is a total of 19.8Mt of marginal ore, defined as the material with a grade higher than the marginal COG but lower than the COG. The material has an average grade of 0.46 g/t. The marginal material is stockpiled in the low-grade stockpiles in order to be processed once the high-grade ore has been depleted.

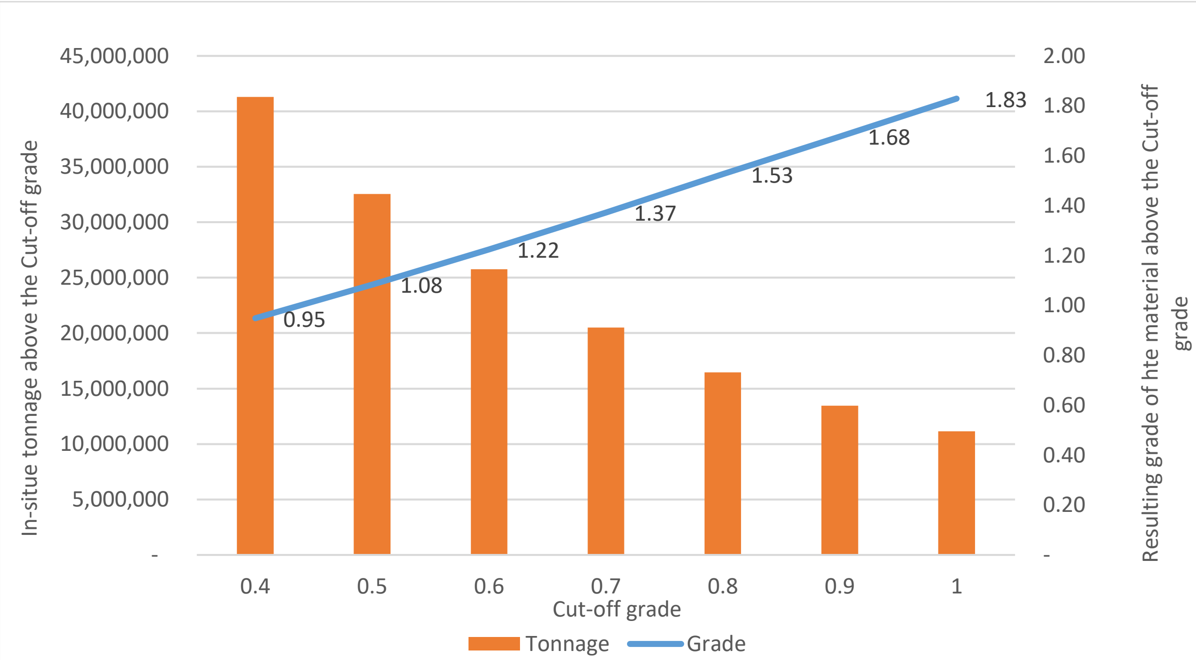
The COG for the Kobada Project is given in Table 15.5.

Table 15.5: COG for the Kobada Project

Material	COG (g/t of Au)	
	Direct Feed to Mill	Marginal COG
Laterite	0.60	0.35
Saprolite	0.60	0.35
Transition	0.60	0.35
Sulphide	0.60	0.35

The COG used to determine if the material will be fed directly to the mill has been calculated to maximise the gold production for the first ten years of the LOM and reach an average of 100 koz/a of gold production, given the mill throughput constraint of 3 Mt/a of material.

Figure 15.3 shows the grade tonnage curve of the in-pit material (prior to applying any dilution and mining recovery parameters).

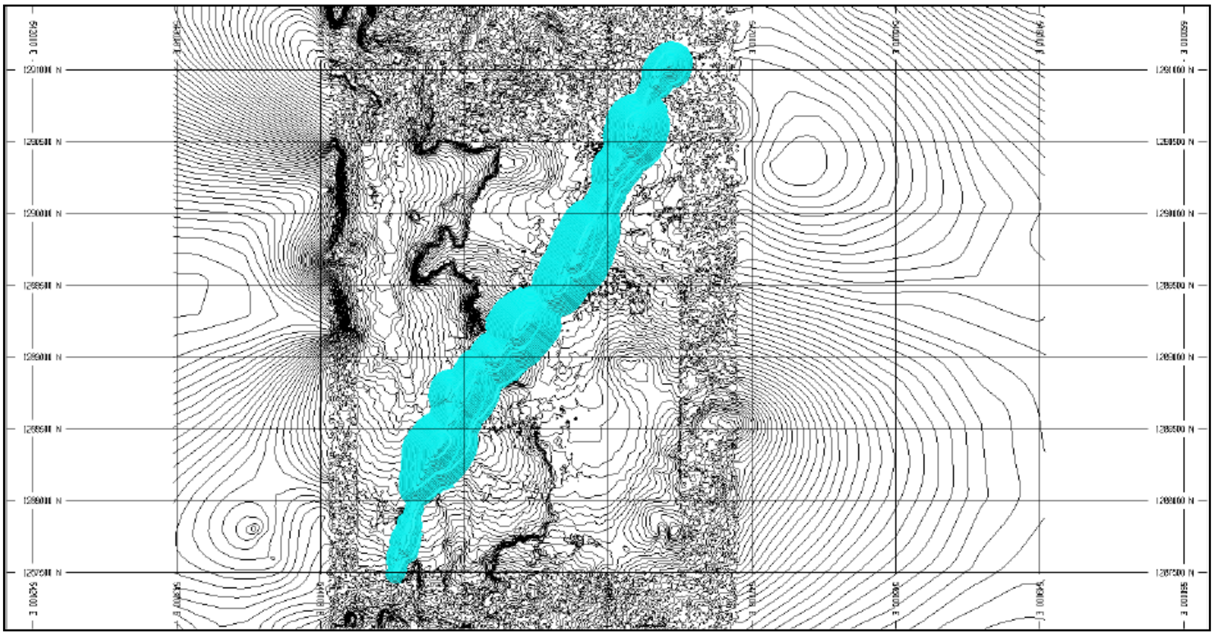


Source: DRA 2021

Figure 15.3: Grade Tonnage Curve

15.2.4 Open-Pit Design Results

After determining the ultimate pit limit, an operational pit must be designed. This pit forms the basis of the production plan and the mineral reserve estimate. The operational pit uses the selected ultimate pit limit as a guide and smooths the pit walls, adds ramps to access the pit bottom, adds catch benches, and ensures that the pit can mined with the selected equipment. The operational pit is presented Figure 15.4.



Source: DRA 2021

Figure 15.4: Kobada Pit Design

15.3 MINERAL RESERVE STATEMENT

The Mineral Reserves are estimated at 45.0 Mt of Proven and Probable Mineral Reserves at a grade of 0.87 g/t Au based on the marginal COG of 0.35 g/t. To access these reserves, 157.9 Mt of waste rock will need to be removed. This results in a stripping ratio of 3.5 to 1 (waste to ore). Table 15.6 presents the open-pit Mineral Reserves estimate for the Project.

Table 15.6: Kobada Mineral Reserve Estimate

Category	Material	Ore (kt)	Grade (g/t)	In Situ Ounces (koz)	Recovered Ounces (koz)
Proven	Laterite	336	0.87	8.3	8.0
	Saprolite	12,276	0.90	331.8	320.1
	Transition	1,776	0.90	47.4	42.9
	Sulphide	5,870	0.94	140.4	133.9
Subtotal		**20,259**	**0.91**	**527.8**	**504.9**
Probable	Laterite	1,288	0.85	36.1	34.8
	Saprolite	16,015	0.88	465.6	449.3
	Transition	1,975	0.87	57.2	51.8
	Sulphide	5,492	0.84	165.9	158.2
Subtotal		**24,770**	**0.87**	**724.7**	**694.1**
Total Proven and Probable	Laterite	1,624	0.85	44.4	42.8
	Saprolite	28,291	0.88	797.3	769.4
	Transition	3,752	0.87	104.6	94.6
	Sulphide	11,362	0.84	306.3	292.2
Total		**45,029**	**0.87**	**1,252.5**	**1,199.0**

Notes:
1. The effective date of the Mineral Reserve Estimate is 24 September 2021.
2. Mineral Reserves are reported in accordance with the CIM guidelines.
3. A marginal COG of 0.35 g/t Au for all material is applied.
4. Mineral Reserves were estimated at a gold price of US$1,610/oz and include modifying factors related to mining costs, dilution and recovery, process recoveries and costs, G&A costs, and royalties.
5. Figures have been rounded to an appropriate level of precision for the reporting of Mineral Reserves.
6. Due to rounding, some columns or rows might not add up exactly as shown.
7. The Mineral Reserves are stated as dry tonnes processed at the crusher. All figures are in metric tonnes.
8. The in situ and recovered ounces are in troy ounces.

15.4 IN-PIT INFERRED RESOURCES

Inferred Resources are not used to determine Mineral Reserves. Note that the *CIM Definition Standards for Mineral Resources and Mineral Reserves* (2014) define inferred resources as follows:

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.

Table 15.7 shows the total amount of inferred resources inside the ultimate pit limit.

Table 15.7: Kobada Inferred Resources Included Inside the Ultimate Pit Limit

Category	Material	Ore (kt)	Grade (g/t)	In Situ Ounces (koz)
Inferred	Laterite	1,109	0.97	34.6
	Saprolite	5,914	0.97	184.9
	Transition	544	0.93	16.3
	Sulphide	1,492	0.94	45.0
Total		**9,059**	**0.96**	**280.9**

Notes:
1. Inferred Resources are excluded from the Mineral Reserves estimate.
2. A marginal COG of 0.35 g/t Au for all material is applied.
3. Figures have been rounded to an appropriate level of precision.
4. Due to rounding, some columns or rows might not add up exactly as shown.
5. Inferred resources within the ultimate pit limit shown above are in-situ tonnes with no mining recovery and dilution applied.
6. In situ ounces are in troy ounces.

There is approximately 9.1 Mt of inferred resources that can be drilled by an in-fill drilling program, similar to the Phase 4 drill programme completed in January 2021. It is highly probable that this will increase the size of the measured and indicated resource, and therefore the reserves within the optimised pit shell. It is recommended that this drilling be prioritised prior to the start of production in order to increase the ounces available for processing.

16 MINING METHODS

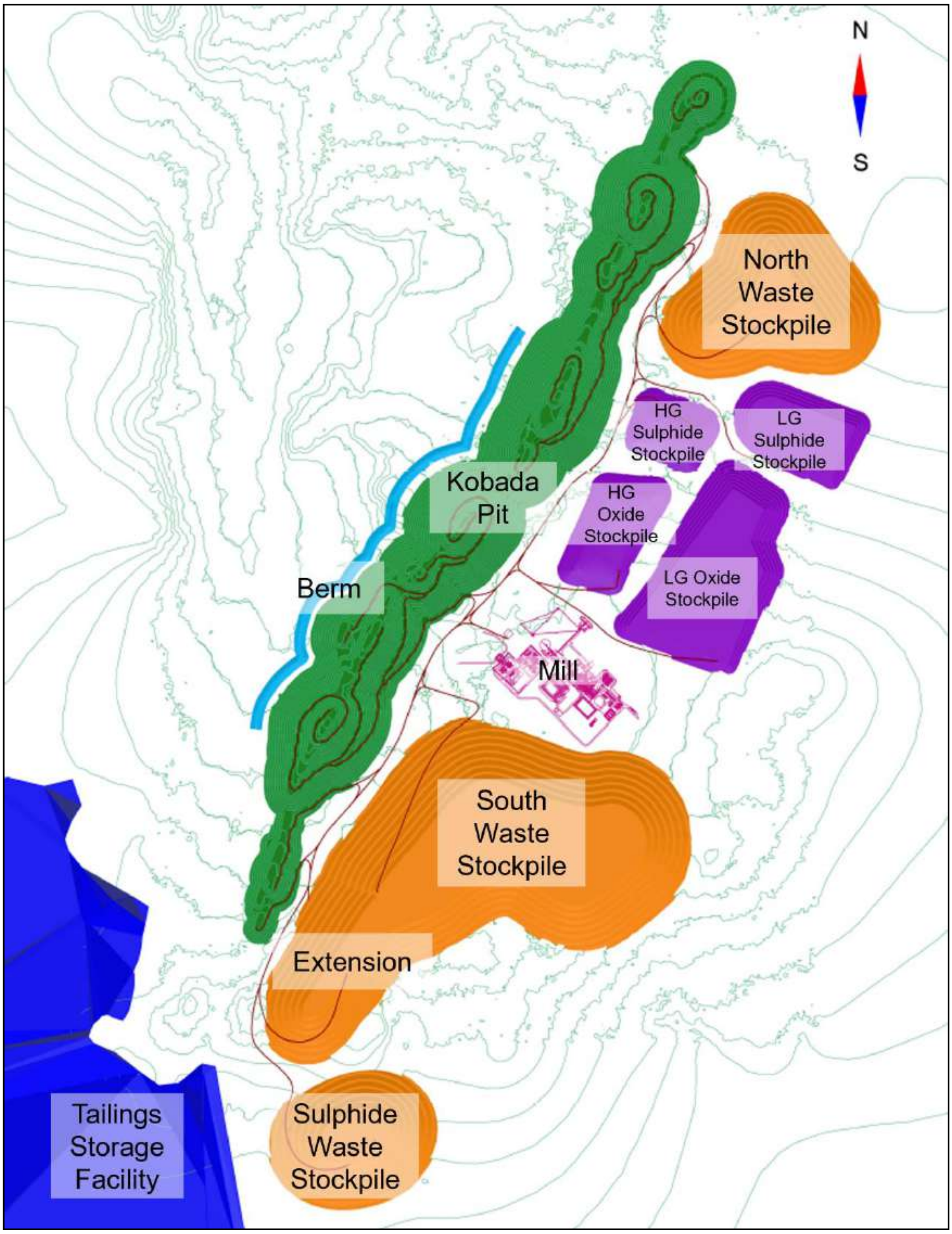
16.1 INTRODUCTION

The Kobada Open-Pit Project is an approximately 16-year LOM operation including the pre-production period. The mine will need to support a processing plant with a nominal mill feed of 3 Mt/a of ROM.

The mineralised material contained within the final pit design has been determined based on a 0.35 g/t gold COG. All Inferred Resource materials are treated as waste in the pit optimisation. The mill feed is made up entirely of Measured and Indicated Resource materials (laterite, saprolite, transition and sulphide) above the calculated COG.

This section describes the mine design and mine engineering for the Project, including pit optimisation, open-pit phasing and design, ore and waste stockpile design, annual mine production plans, and a description of the planned open-pit operations.

Figure 16.1 shows the locations of the active mining area, mill, ore stockpiles, waste stockpiles, and other significant infrastructure features discussed in this section.



Source: DRA 2021

Figure 16.1: General Site Layout

16.2 MINING METHODOLOGY

The Kobada Gold Mine lends itself to a standard open-pit mining method using articulated trucks and a hydraulic loader (hydraulic shovel or excavator).

Approximately 66% of all the material to be mined is contained in the saprolite and laterite. Although this type of material can be extracted directly with excavators, without drilling and blasting, it has been estimated that approximately 5 % of this material will require some drilling and blasting. All transition and sulphide material must be drilled and blasted.

The first six months will be exclusively dedicated to pre-production without feeding the process plant.

From Year 1 to Year 8, the process plant will be fed mainly with high-grade oxides from the pit and supplemented by the ROM stockpile.

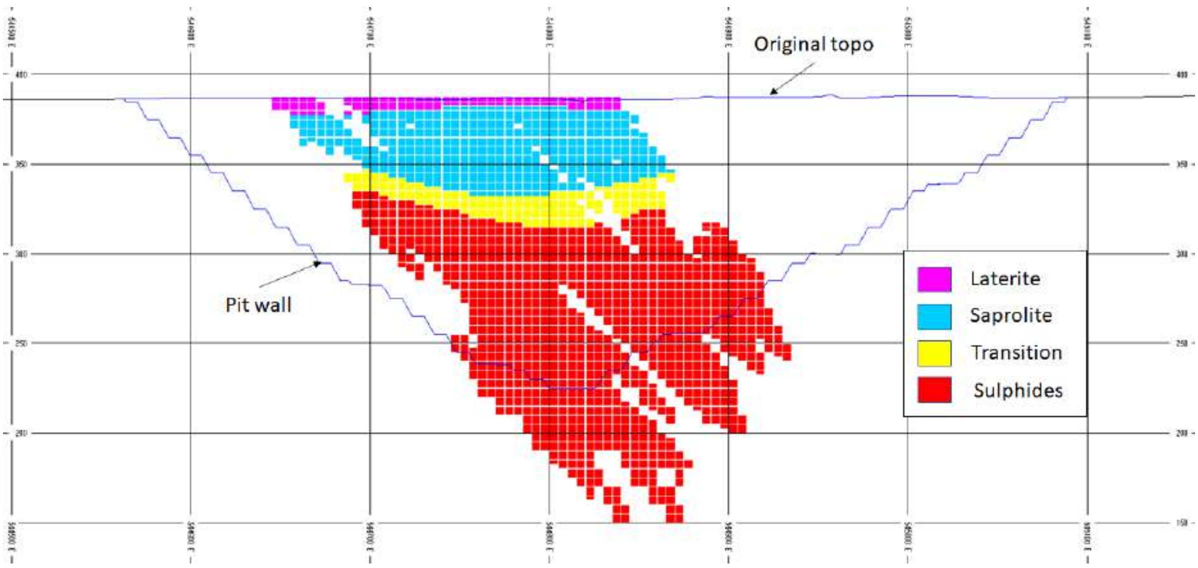
From Year 9 to Year 11, the process plant will be fed mainly with high-grade sulphides from the pit and supplemented by the ROM stockpile.

From Year 12 to Year 16, the process plant will be fed with material from the low-grade stockpiles exclusively, using a front-end loader and articulated trucks.

For mining purposes, the Kobada orebody can be considered a top-down layered sequence of the following material types:

- Laterite
- Saprolite
- Transition
- Sulphide

A typical transversal section in the central pit showing the principal geological layers is shown in Figure 16.2.



Source: DRA 2021

Figure 16.2: Typical Sections of the Kobada Orebody

The saprolite thickness can exceed 120 m in the deepest area of the pit, with an average thickness of 65 m.

In the transition and sulphide areas, competent ore found in the deepest areas of the pit will need conventional pre-split drilling and blasting to fragment the rock for excavation.

16.3 OPEN-PIT MINE DESIGN

The detailed pit design consisted of the following:

- Designing operational pits (with ramps and benches) based on the optimal pit shells and geotechnical pit slope parameters
- Developing internal pit phases (pushbacks) to moderate the annual mined tonnages

The engineered pit designs were developed based on the pit shell determined by GEOVIA Whittle software as described in Section 15. This pit shell was used as a guide to design the operational pit. The operational pit smooths the pit walls, adds ramps to access to access the pit bottom, adds safety berms, and ensures the pit can be mined with the selected equipment.

16.3.1 Design Criteria

The key design criteria used to develop the pit design are summarised as follows:

- Nominal bench height: 5 m, double benched, when possible, for final pit walls
- Overall slope angle: 40°
- Road width: 15.6 m, to accommodate two-way traffic
- Ramp gradient: maximum 10 %
- Working areas: a minimum of three independent working areas, based on the mine schedule

16.3.2 Geotechnical Pit Slope Parameters

This section summarises the work performed by OHMS, the geotechnical consultant retained by AGG.

OHMS performed a geotechnical investigation for the purpose of specifying the optimal slope geometry of the Kobada open pit mine.

From 1988 to 2019, more than 1 500 boreholes were drilled during various exploration drilling campaigns. The locations of these holes are shown in Figure 16.3.

Additionally, geotechnical evaluations were conducted in 2015 on13 boreholes.

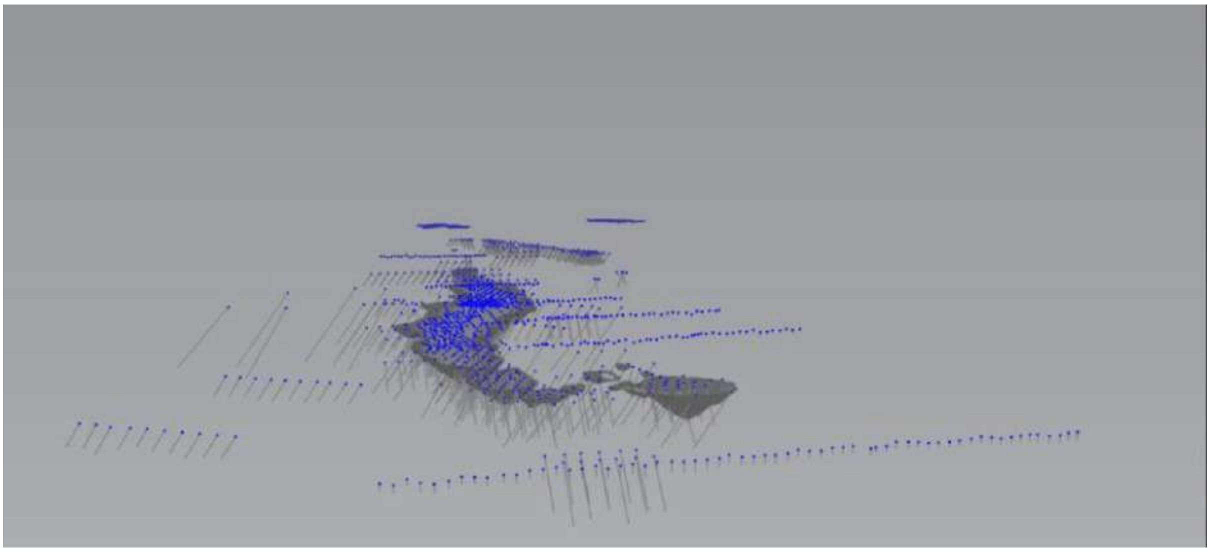


Figure 16.3: Spatial Representation of Existing Boreholes

The geotechnical analysis issued by OHMS followed the methodology shown in Figure 16.4.

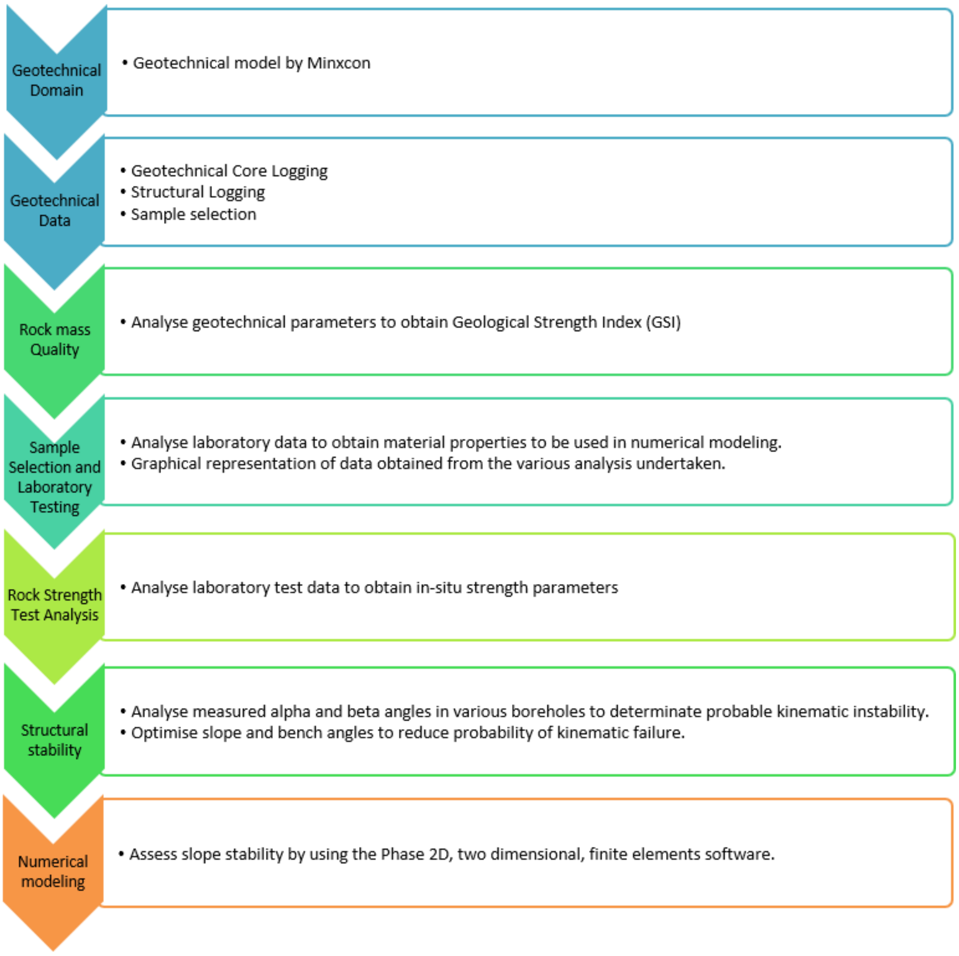


Figure 16.4: Flow Diagram of OHMS Methodology

Minxcon analysed the profiles of the boreholes of previous investigations spatially and statistically and created a model that identifies three geotechnical domains, identified as saprolite, transition and sulphide rock. By dividing the pit into zones with similar mechanical properties across the slope, it was determined that most of the slope consists of saprolite material.

To determine the values for joint condition and rock mass classification, 1,653.58 m of core was geotechnically logged during the Project in 13 newly drilled boreholes (see Figure 16.5).

All the strength test data from the 2015 drilling campaign (borehole locations shown in Figure 16.6) has been included in the geotechnical assessment. Core logging data was used to quantify the various geotechnical parameters for the different rock types; however, the slope stability was assessed using the geotechnical domains.

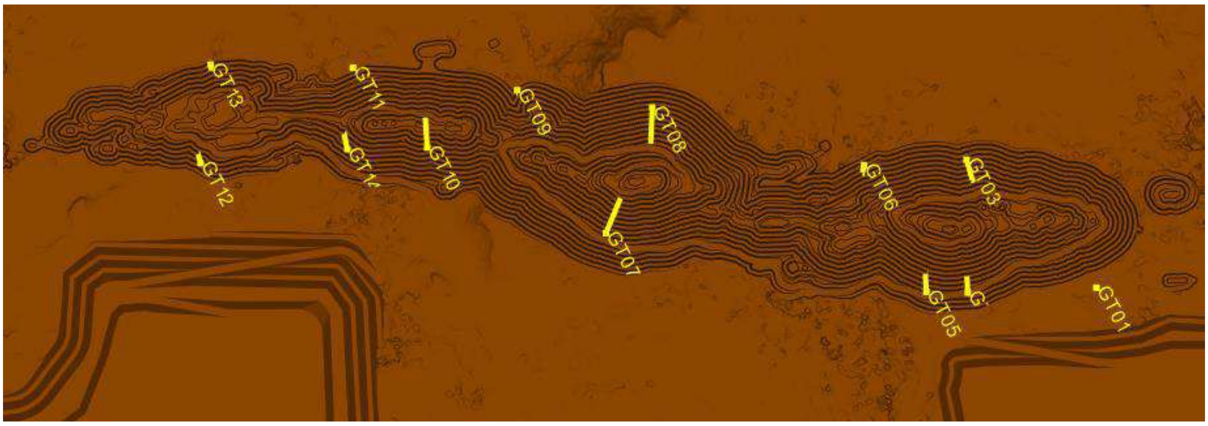


Figure 16.5: 2019-2020 Geotechnical Borehole Locations

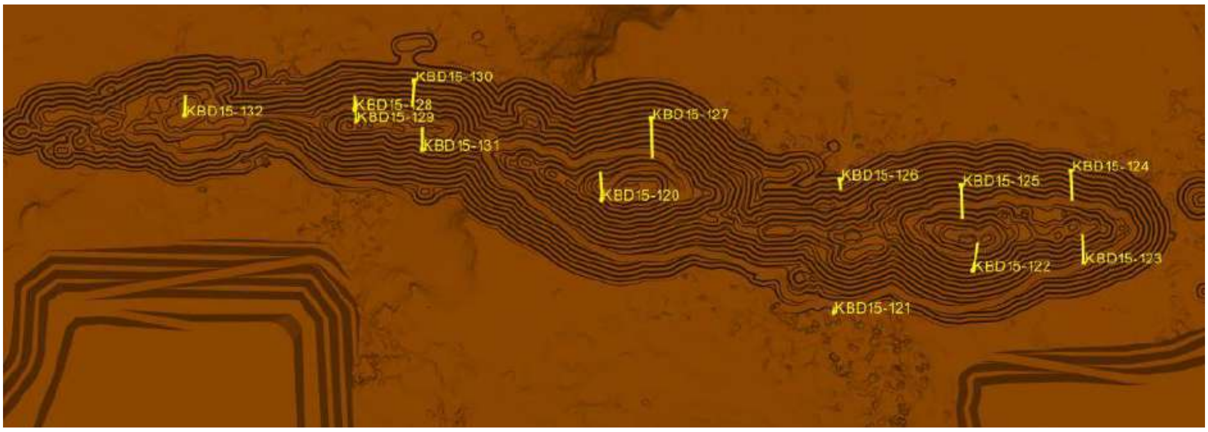


Figure 16.6: 2015 Geotechnical Borehole Locations

The method proposed by Hoek, Carter and Diederichs (2013) to determine the geological strength index (GSI) was used to obtain the mechanical properties for modelling, as this is generally accepted to be the most accurate method. The primary use of rock mass classification data was to downgrade the laboratory strength values and allow for the specification of in-situ mechanical properties of the various primary rock types that are anticipated in the future pit slopes.

A series of uniaxial, triaxial and tensile strength tests were conducted to quantify the intact strength of the relevant rocks. OHMS collected 168 samples from the newly drilled boreholes, and of these, 163 samples were tested successfully. During the 2015 campaign, 64 samples were tested, and these were also used to determine the mechanical properties of the relevant rock types. All the tests were conducted according to the prescribed ISRM (International Society for Rock Mechanics) procedures and excluded the cover layer of topsoil and laterite because those are completely decomposed rock and are generally mechanically weak.

Slope stability may be affected by discontinuities at the surfaces and the interaction between such surfaces. The interaction between the slope face and joint sets may occur spatially, which could create conditions conducive to kinematic failure. Given the northeast to southwest strike orientation of the orebody, the pit slopes would be predominantly dipping northwest and southeast.

The potential for the three most common structurally controlled failure modes was investigated and was tested on all the highwall orientations as shown in Figure 16.7.

Generally, it seems that the slope walls will be kinematically stable with an overall slope angle of 40°. According to the kinematic analysis, toppling failure may be expected on the northwest dipping slope at a 310° dip direction. However, the data set for Joint Set 3 consists of nine joints, of which only two joints are included in the 22.22% possibility for failure. When the slope rock mass features a set of continuous, closely spaced and steeply inclined structural planes with approximately the same strike as that of the slope, the slope may undergo toppling failure.

OHMS showed that the runoff distance of a perfectly square box-shaped fall body will be influenced by the mass of the fall body and the bench height and batter angle. OHMS proposed the following equation to quantify the 95th percentile runoff distance:

$$Runoff\ 95m = 11.2 + (0.191 \times Maxx) + (0.271 \times Bench\ Height) - (0.151 \times Batter\ Angle)$$

A perfectly square box-shaped fall body of 10 t will have a runoff distance of 6.5 m, where the bench height is 10 m at 62° batter angles. This implies that a minimum width of 6.5 m is required for catchment berms. Should a batter angle of 62° be used for 10 m high benches, catchment berms of 6.5 m wide will be formed where the overall slope angle is 40°, the bench stack angle is 42°, and the slope height is 180 m.

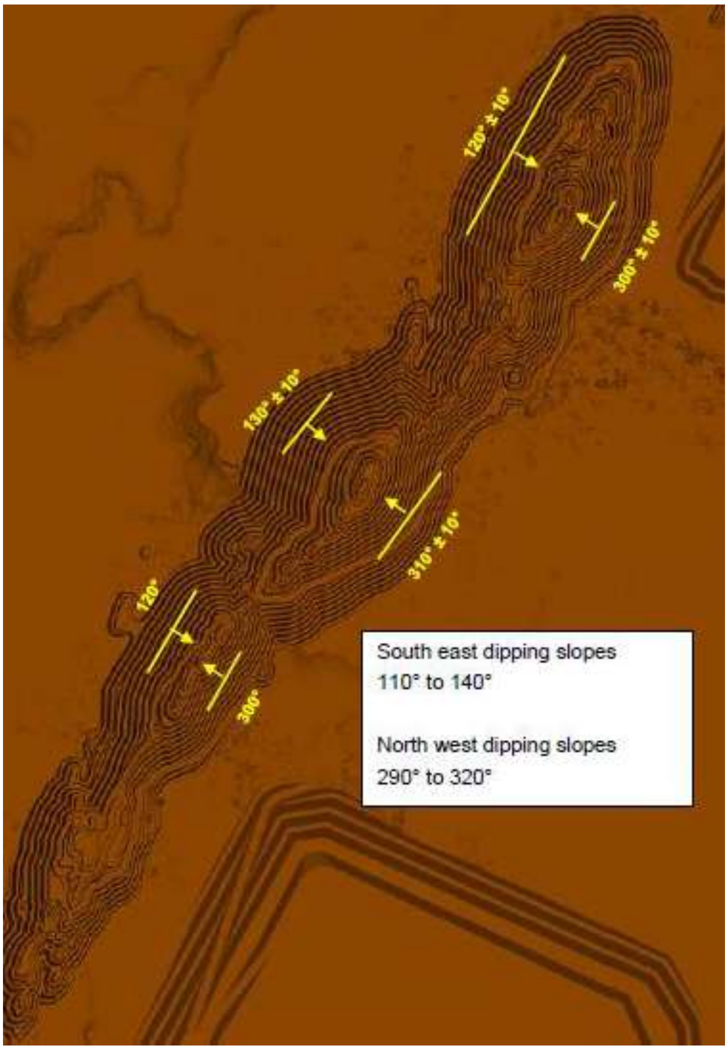


Figure 16.7: Slope Orientations

An acceptable slope dimension is shown in Figure 16.8.

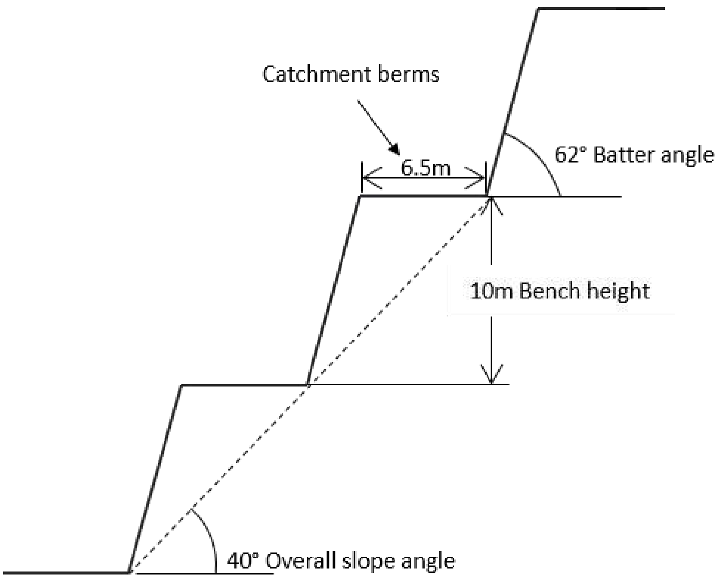


Figure 16.8: Slope Dimension – Pit Wall

Based on similar studies on ore and waste stockpiles, DRA used an overall slope angle of 18°, bench heights of 10 m for waste stockpiles and 5 m for ore stockpiles, and a batter angle of 27°. Figure 16.9 and Figure 16.10 show the ore stockpile wall and the waste stockpile wall, respectively.

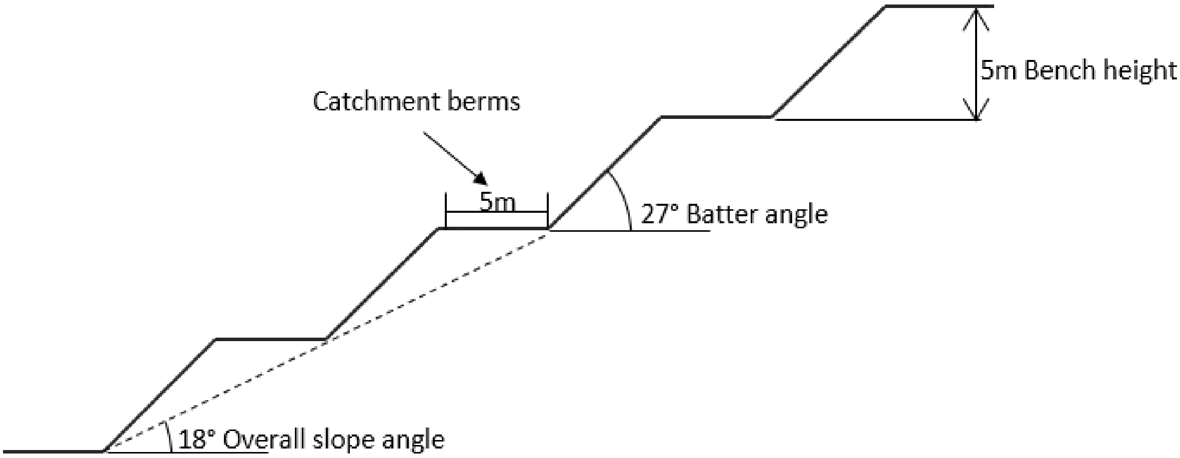


Figure 16.9: Slope dimension – Stockpile Wall

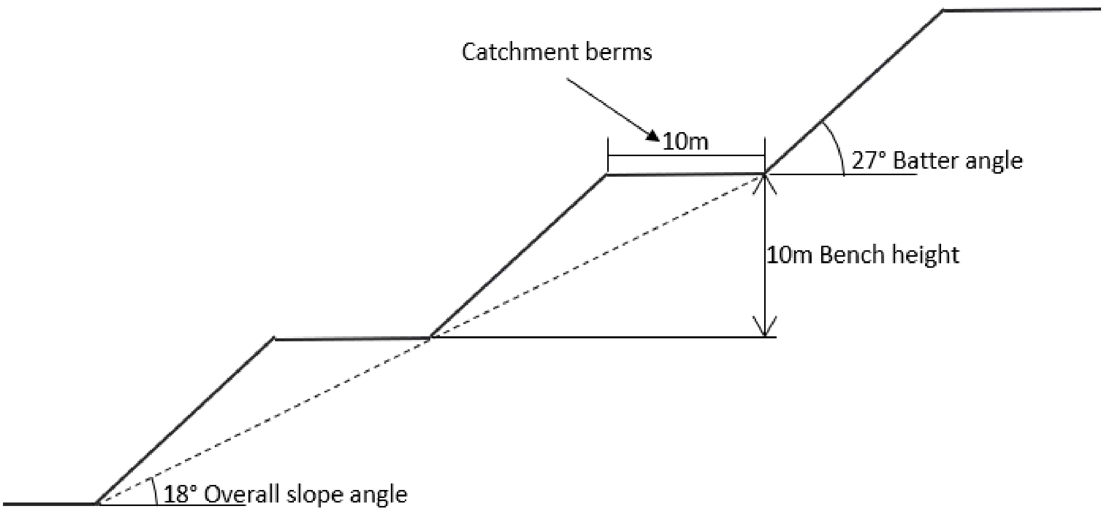


Figure 16.10: Slope dimension – Waste Stockpile Wall

16.3.3 Haul Road Design

The ramps and haul roads were designed with an overall width of 15.6 m. For double-lane traffic (see Figure 16.11), industry practice indicates the running surface width to be a minimum of 2.5 times the width of the largest truck. The overall width of a 55 t articulated haul truck is 3.9 m, which results in a running surface of 9.97 m. The allowance for berms and ditches increases the overall haul road width to 15.6 m. A maximum ramp grade of 10 % was used.

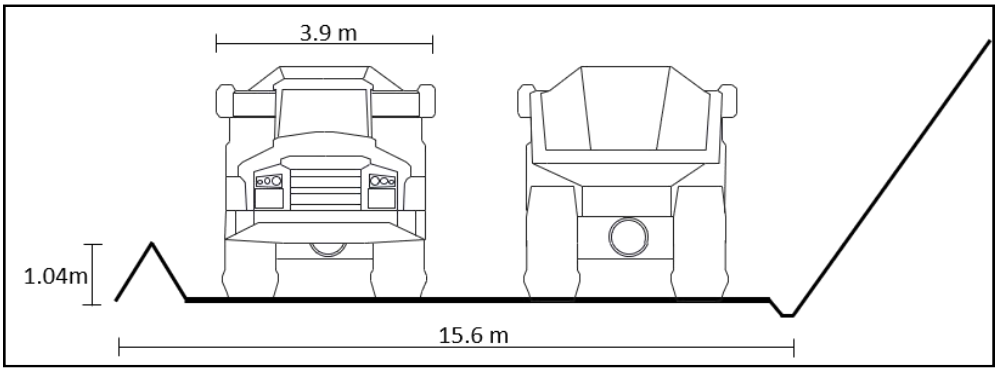


Figure 16.11: Typical Mine Ramp Configuration (In-Pit)

16.3.4 Final Pit Design

Based on the selected pit shell generated by GEOVIA Whittle software

The ultimate pit is approximately 4.3 km long, with a maximum width of 500 m and a maximum depth of 180 m. This pit is composed of several sectors, and each sector is accessed via one of the four main access ramps. All the access ramps daylight at an elevation between 375 m and 405 m. The mill is located East of the Pit.

Figure 16.12 shows the ultimate pit, along with the mill.

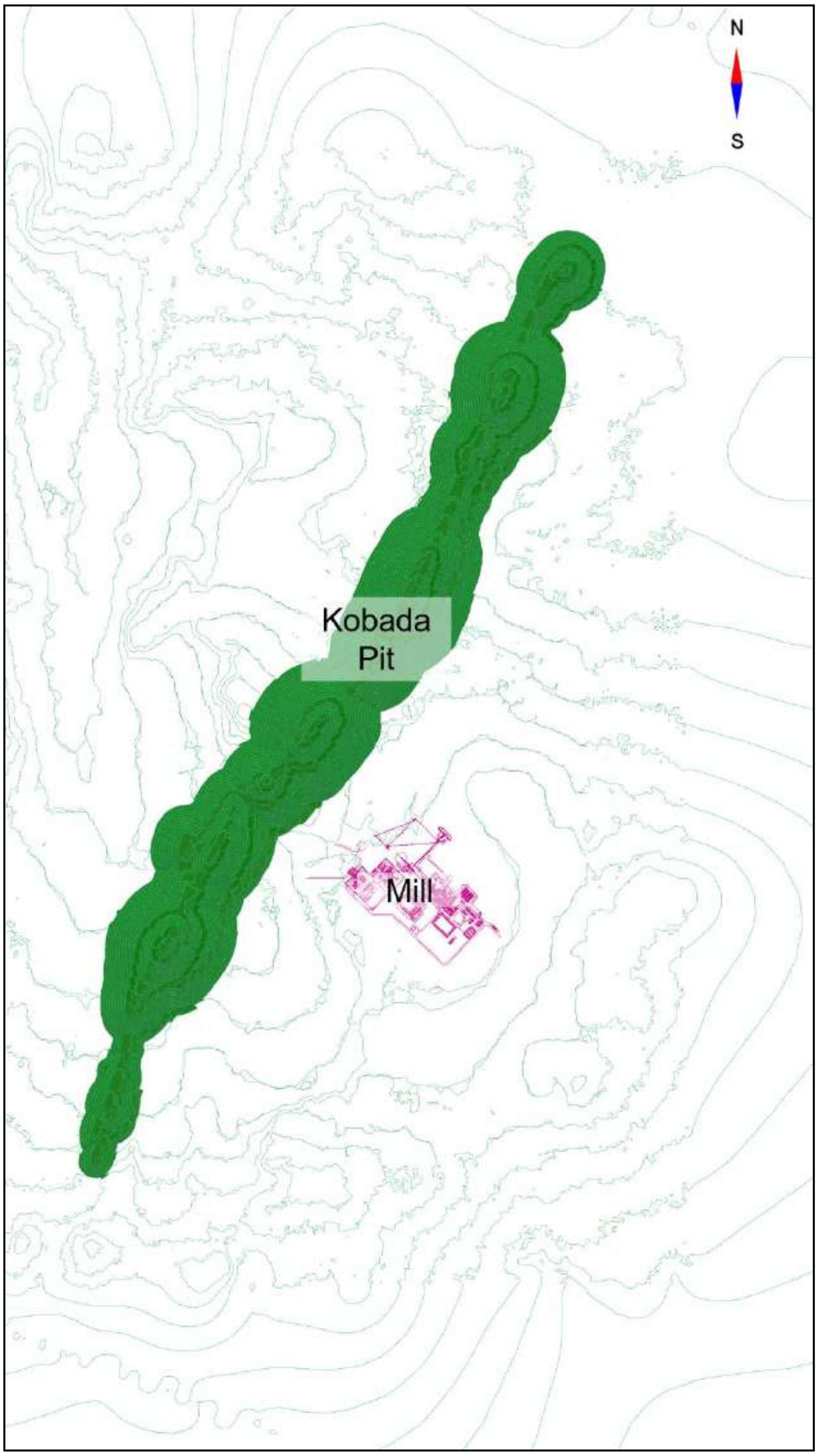


Figure 16.12: Ultimate Pit

Table 16.1 shows the total reserve by mineable material.

Table 16.1: Total Reserve by Material

Zone	Mineable Material	Ore		Waste	Total (Ore + Waste)
		kt	Au (g/t)	kt	kt
Ultimate Pit	Laterite	1,625	0.85	19,221	20,846
	Saprolite	28,273	0.88	104,774	133,047
	Transition	3,752	0.87	12,610	16,362
	Sulphide	11,362	0.84	21,299	32,661
TOTAL		**45,012**	**0.87**	**157,903**	**202,915**

Notes
1. Figures have been rounded to an appropriate level of precision
2. Due to rounding, some columns or rows may not compute exactly as shown

16.3.5 Open-Pit Phasing

16.3.5.1 Intermediate Phase Selection

To distribute the annual mined waste and ore rock tonnages, a total of three phases or pushbacks have been designed to optimise the mine schedule and maximise the Project value. Each phase independently guarantees a positive cash flow and adequate stripping ratio. The phases will also be used as orientation criteria (but not determining criteria) for the final open pit mine plan.

The criteria for the selection of the pit shells to be used for the pit design were driven by operational design restrictions related to topography, process plant location, access road and the size of each individual phase.

Each individual phase must also be big enough to be developed independently from the other phases, but not so big that the development towards the next phase is compromised due to impracticality (the minimum mining width must be respected). Due to these constraints, the Kobada Pit could not be developed in more than three phases.

Each consecutive phase will present a lower grade and a higher strip ratio.

Table 16.2 presents the total mineable material by phase.

Table 16.2: Total Mineable Material by Phase

Phase	Mineable Material	Ore		Waste	Total (Ore + Waste)
		kt	Au (g/t)	kt	kt
Phase 1	Laterite	1,469	0.86	5,125	6,594
	Saprolite	12,178	0.86	15,199	27,378
	Transition	240	0.98	72	312
	Sulphide	47	1.11	6	53
	Total	**13,933**	**0.86**	**20,403**	**34,336**
Phase 2	Laterite	-	-	2,929	2,929
	Saprolite	7,389	0.83	25,099	32,487
	Transition	1,485	0.84	2,620	4,105
	Sulphide	4,687	0.86	3,411	8,098
	Total	**13,560**	**0.84**	**34,059**	**47,619**
Phase 3	Laterite	172	1.16	11,167	11,339
	Saprolite	8,691	0.94	64,476	73,167
	Transition	2,028	0.88	9,918	11,946
	Sulphide	6,629	0.82	17,882	24,511
	Total	**17,519**	**0.89**	**103,442**	**120,961**
Total		**45,012**	**0.87**	**157,903**	**202,915**

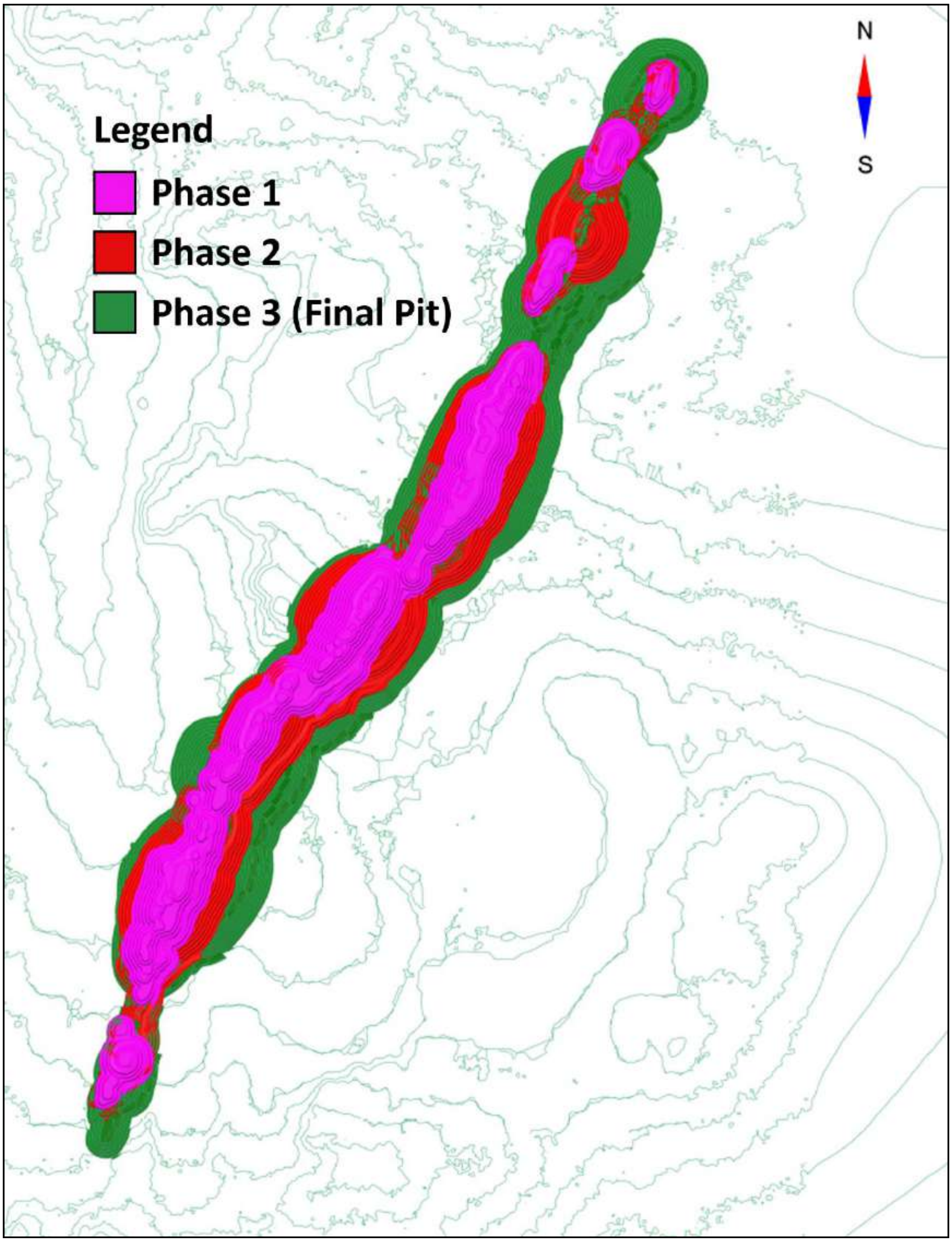
Notes
1. Figures have been rounded to an appropriate level of precision
2. Due to rounding, some columns or rows may not compute exactly as shown

Whittle optimisations were performed to obtain the best directional extraction trends considering only Measured and Indicated resources in the pit shells. Once this sequence was defined, the phase or pushback shapes were defined by selecting incremental nested shells using the Best/Worst NPV Analysis in Whittle.

Pit Shells 11 and 23, correspond to the intermediate pit shells selected for the design of Phase 1 and Phase 2. Table 15.3 shows the main values of the selected pit shells.

16.3.5.2 Phase Design

Figure 16.13 shows the phases created for the Kobada pit.



Source: DRA 2021

Figure 16.13: Kobada Phases

In terms of phase design, the minimum expansion width was considered a design criterion. For this criterion, the largest operational loading equipment recommended by the potential mining contractors was considered (100 t shovels, 6 m^3 bucket size). This equipment has a loading range of 8 m, and the minimum expansion width for loading on both sides will be 24 m, as shown in Figure 16.14. Table 16.3 presents a description of the measurements for the design criteria.

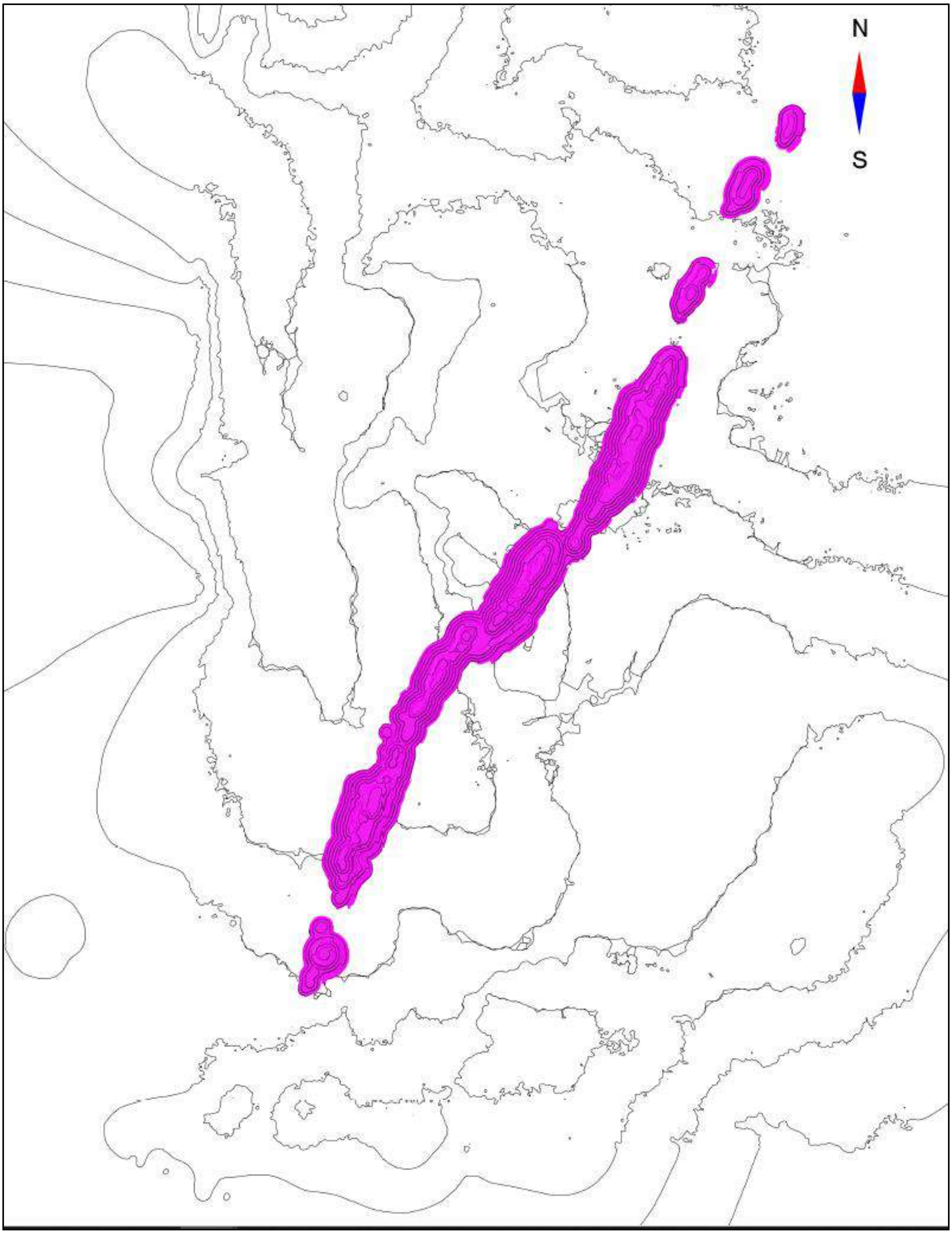
Figure 16.14: Typical Phase Expansion

Table 16.3: Minimum Expansion Widths

Description	Measurement (m)
Safety distance	4
Loading with one truck	8
Berm width	4
Minimum expansion width (2 trucks' width)	24

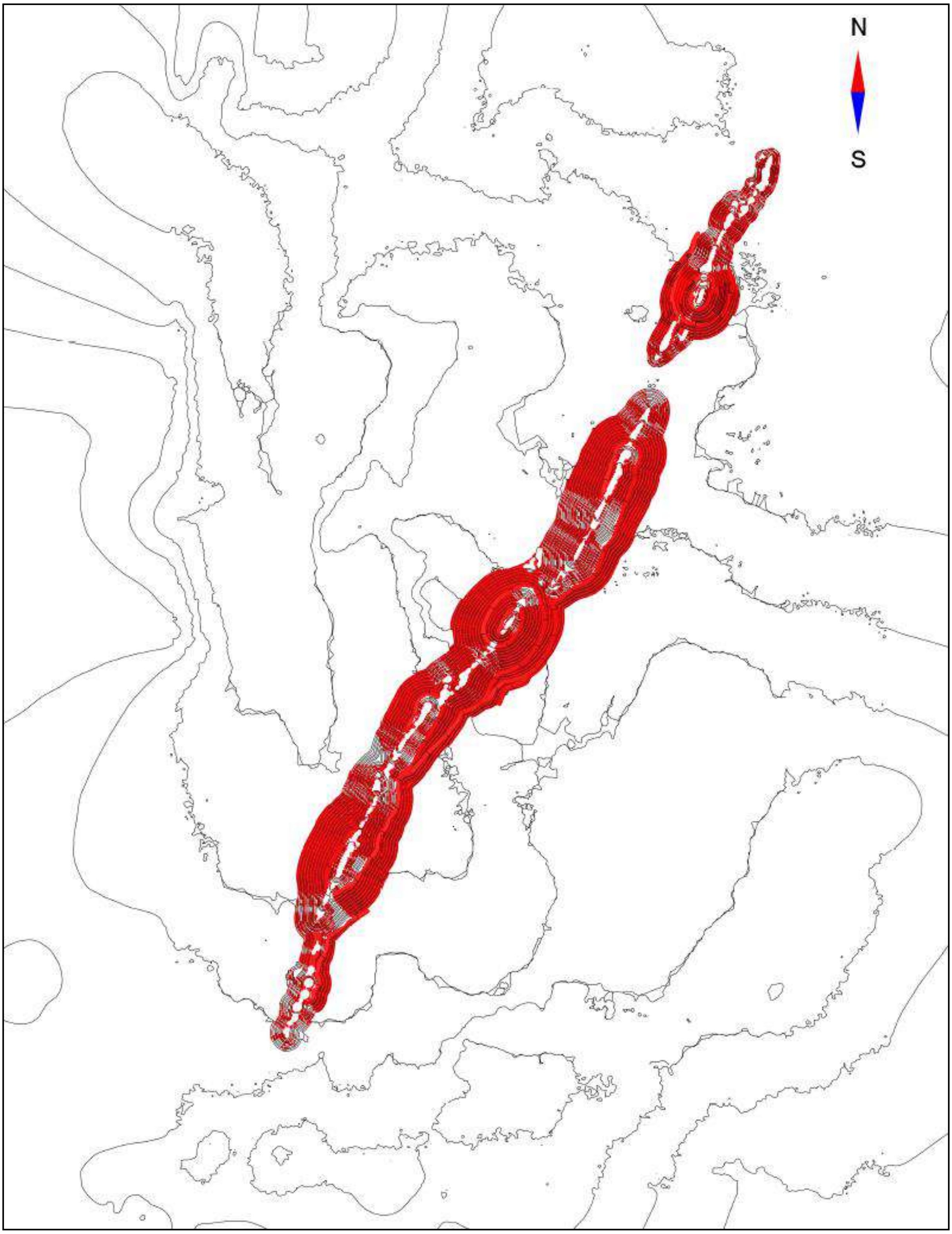
Geometric restrictions were applied to the operational smoothing of the pit envelopes, and the mining phases were obtained. The operational design of the final pit followed the indicated shells as closely as possible. Figure 16.15 to Figure 16.17 show the final geometry for the design of each of the three phases of the Kobada Open Pit. The phase designs consider the following criteria to ensure the greatest financial return:

- The phase designs were adjusted to the reference Whittle pit shells in order to ensure the extraction of the best gold grade in the first years, delaying the extraction of lower-grade material.
- Where possible, double ramp systems were developed in the phases in order to ensure operating continuity and the availability of alternate routes for extraction.



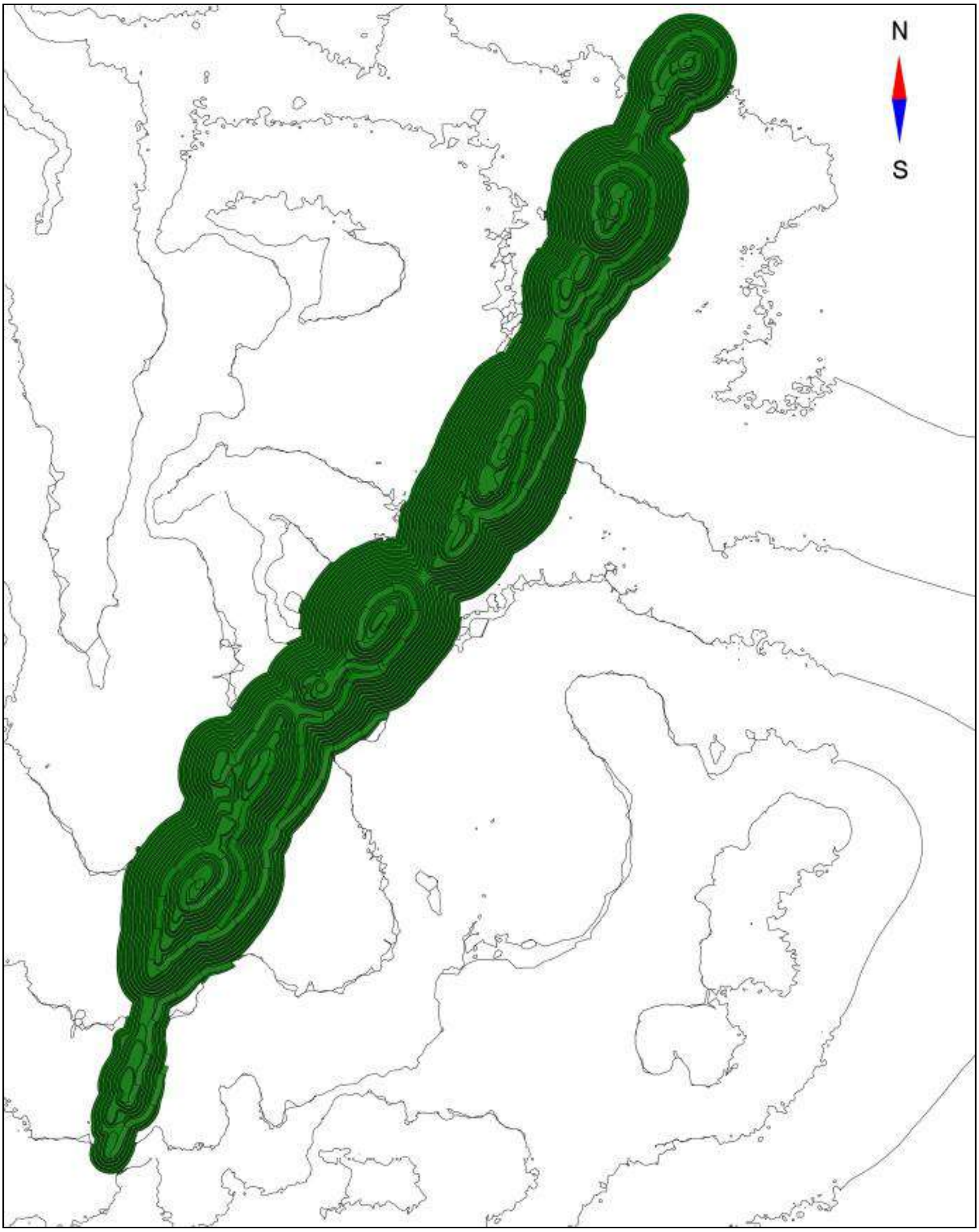
Source: DRA 2021

Figure 16.15: Kobada Phase 1 Design



Source: DRA 2021

Figure 16.16: Kobada Phase 2 Design



Source: DRA 2021

Figure 16.17: Kobada Phase 3 Design (Final Pit)

16.4 MINING PRODUCTION SCHEDULE (MINE PLAN)

DRA has developed a 16-year LOM plan for the Kobada Project using a combination of software packages, including Hexagon MinePlan™ and TALPAC 3D. Whittle was used to determine the ultimate pit limit and MinePlan™ was used to develop the extraction schedule as well as pit and stockpile design work while TALPAC 3D was used to determine equipment haulage times.

The final mining plan scheduling exercise was based on the requirement for a minimum recovery of 100 koz of recoverable gold per year for the first ten years. The production schedule consists of 6 months of pre-production stripping followed by 11 years of open-pit production, and an additional 5 years dedicated to exclusive stockpile rehandling, for a total LOM of 16 years.

This mine plan was developed using the MinePlan Schedule Optimiser (MPSO). This tool generates the most practical and most productive mining cut sequence to achieve the highest profitability while satisfying all the product quality and quantity constraints, geotechnical constraints and destination capacities. It determines the optimal schedule requirements to achieve the following criteria:

- Maximise the NPV of the project.
- Optimise production in the early years of operation by using stockpiles and concurrent open pit mining.
- Optimise the pit production rate per period.
- Operate the open pit mine according to a working schedule (see Table 16.4).

Table 16.4: Mine Working Schedule

Work Schedule	Unit	Value
Weeks per Year	Weeks	50
Days per Week	Days	7
Days per Year	Days	350
Weather Delays	Days	7
Days per Year (after taking into account weather delays)	Days	343
Shifts per Day	Shifts	2
Shifts per Year	Shifts	686
Hours per Shift	Hours	12

Objectives, constraints, and economic parameters are required to use MPSO. These parameters are set on a per period basis.

The main MPSO constraints are:

- The mill feed rate must be ramped up for the first two months until it reaches 250,000 t/month.
- A mill feed rate of 3 Mt/a must be achieved.

- All the ore material between 0.35 g/t (marginal COG) and 0.60 g/t will be placed in a low-grade stockpile.
- The high-grade oxide ore material, with a minimum 0.6 g/t grade, can be sent directly to the mill starting at year 1. If the 3 Mt/a feed rate has been reached, the remaining extracted ore must be placed in a high-grade stockpile.
- The high-grade sulphide ore material, with a minimum 0.6 g/t grade, can be sent directly to the mill starting at year 9. If the 3 Mt/a feed rate has been reached, the remaining extracted ore must be placed in a high-grade stockpile.
- The total amount of material to be moved per year must not exceed 28 Mt.
- From Years 12 to 16, the mill feed will be exclusively dedicated to stockpile rehandling, prioritising the high-grade material.

16.4.1 Pre-Production Period

During the six-month pre-production period, the following mine-related activities are planned:

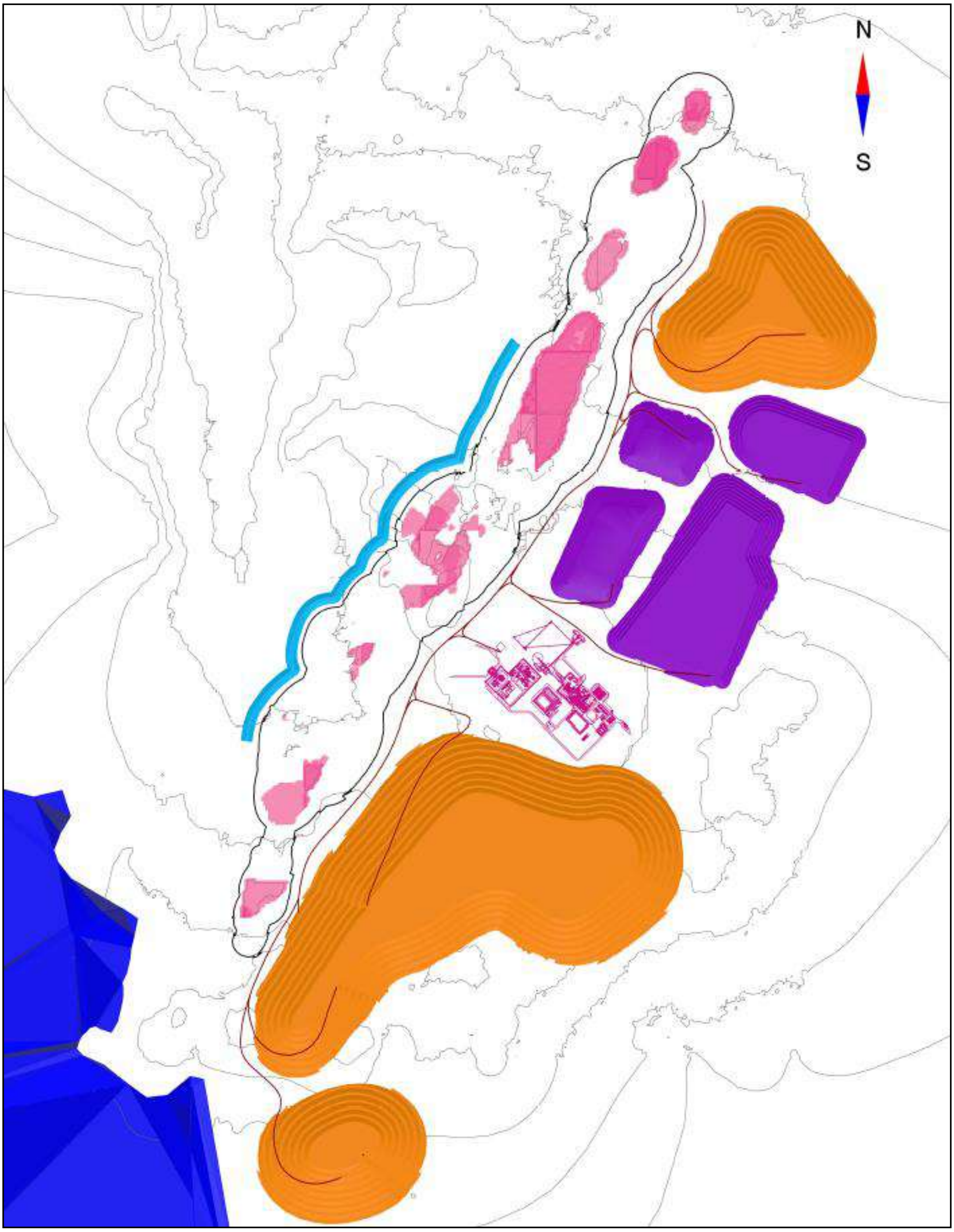
- Clear vegetation and topsoil.
- Strip overburden and waste rock to expose the ore.
- Supply material and prepare pad site for ore and waste stockpiles.
- Supply road construction material and develop haul roads from the top benches to the process plant, ore and waste stockpiles, and tailings storage facility.
- Supply construction material for the tailings dam.
- Stockpile ROM material for the process plant start-up in Year 1.

Table 16.5 summarises the pre-production period schedule. Figure 16.18 shows the progression for the first three months during the pre-production period. Figure 16.19 shows the end of period (EOP) map for the entire pre-production period. In the figures, the pink represents the area mine in that period.

Table 16.5: Pre-Production Period Schedule

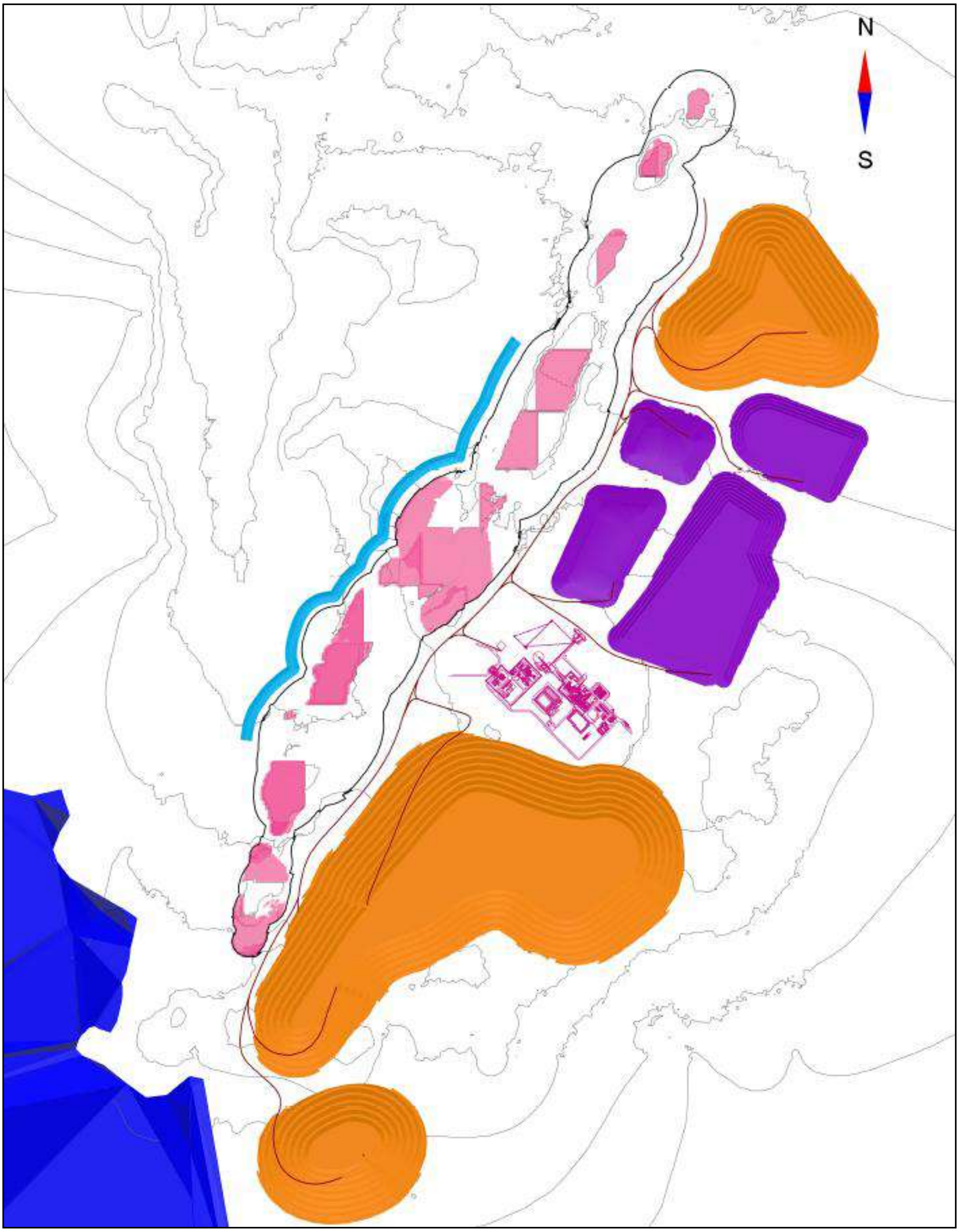
Description	Unit	Pre-Production						Total
		Month 1	Month 2	Month 3	Month 4	Month 5	Month 6	
Ore to Plant								
Tonnage	kt	0	0	0	0	0	0	0
Au	g/t	-	-	-	-	-	-	-
Ore to Oxide High-Grade Stockpile								
Tonnage	kt	143	142	153	118	257	272	1,086
Au	g/t	1.25	1.26	1.43	1.17	1.22	1.13	1.23
Ore to Oxide Low-Grade Stockpile								
Tonnage	kt	105	115	125	117	237	265	964
Au	g/t	0.45	0.45	0.45	0.45	0.45	0.45	0.45
Total Waste	kt	1,419	1,409	1,389	1,431	1,173	1,129	7,951
Total Material Moved	kt	1,667	1,667	1,667	1,667	1,667	1,667	10,000

Notes
1. Figures have been rounded to an appropriate level of precision
2. Due to rounding, some columns or rows may not compute exactly as shown



Source: DRA 2021

Figure 16.18: EOP Map – Month 3 of Pre-Production



Source: DRA 2021

Figure 16.19: EOP Map – Pre-Production

16.4.2 Production Period

The open-pit LOM is approximately 16 years. However, the process plant will fed by rehandled stockpile material during the last 5 years of operation. Phasing of the open-pit development and application of the COG strategy allow for a higher-grade ore (above 0.35 g/t Au) to be processed in the initial years of the operation.

16.4.2.1 Year 1

For the first year of production, DRA prepared a detailed monthly mining plan that includes a two-month process plant ramp-up at 50 % of mill capacity until the nominal process capacity of 250 000 t/month is reached. The process plant ramp-up is presented in Table 16.6. Note that Year 1 represents the 6-month period following the 6 months of pre-production.

Table 16.6: Ramp-Up Period

Month	Mill Feed (t)
1	125,000
2 and beyond	250,000

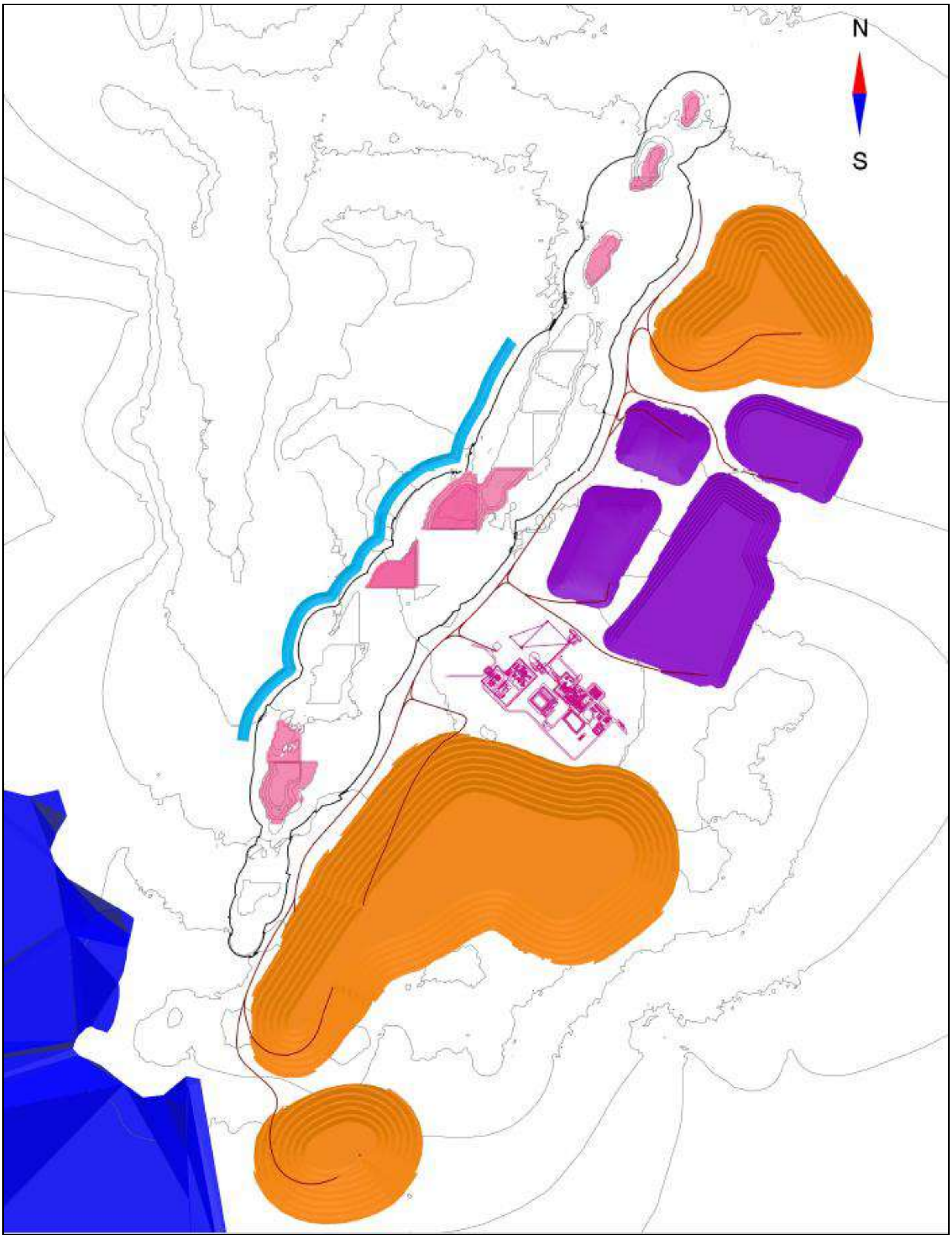
Table 16.7 presents the total material moved during the first year of production, summarised per month. Figure 16.20 and Figure 16.21 present the EOP maps for Months 3 and 6, respectively.

Table 16.7: Mining Plan – Year 1

Description	Unit	Month 1	Month 2	Month 3	Month 4	Month 5	Month 6	Total
Total Ore to Plant								
Tonnage	kt	126	246	254	250	251	251	1,377
Au	g/t	2.08	1.11	1.07	1.09	1.08	1.08	1.18
Mine to Plant								
Tonnage	kt	58	88	97	71	66	58	493
Au	g/t	2.22	0.81	0.78	0.82	0.89	2.22	1.02
Ore to Oxide High-Grade Stockpile								
Tonnage	kt	299	278	207	259	299	308	1,650
Au	g/t	1.15	1.16	1.16	1.04	1.03	1.25	1.13
Ore to Oxide Low-Grade Stockpile								
Tonnage	kt	299	176	215	222	245	308	1,464
Au	g/t	0.46	0.46	0.45	0.46	0.46	0.46	0.46
Stockpile to Plant								
Tonnage	kt	68	158	157	179	185	137	884
Au	g/t	1.96	1.28	1.22	1.18	1.13	1.12	1.25
Total Waste	kt	969	910	964	916	853	799	5,411
Total Material Moved	kt	1,693	1,610	1,639	1,648	1,648	1,665	9,903
Drilling Requirements	kt	86	73	74	73	73	76	455
Strip Ratio	-	1.48	1.68	1.86	1.66	1.40	1.10	1.50

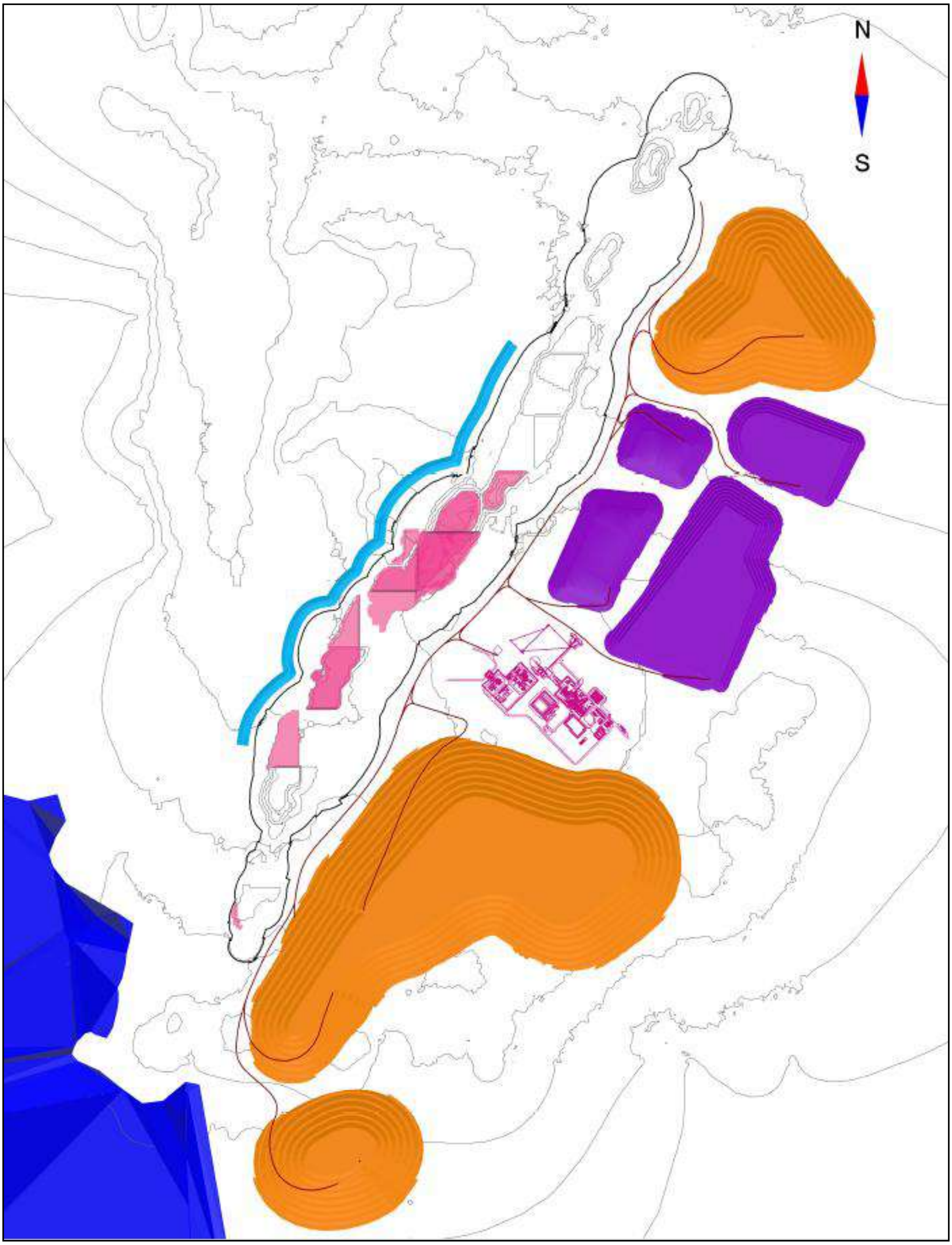
Notes

1. Figures have been rounded to an appropriate level of precision
2. Due to rounding, some columns or rows may not compute exactly as shown



Source: DRA 2021

Figure 16.20: EOP Map Year 1 – Month 3



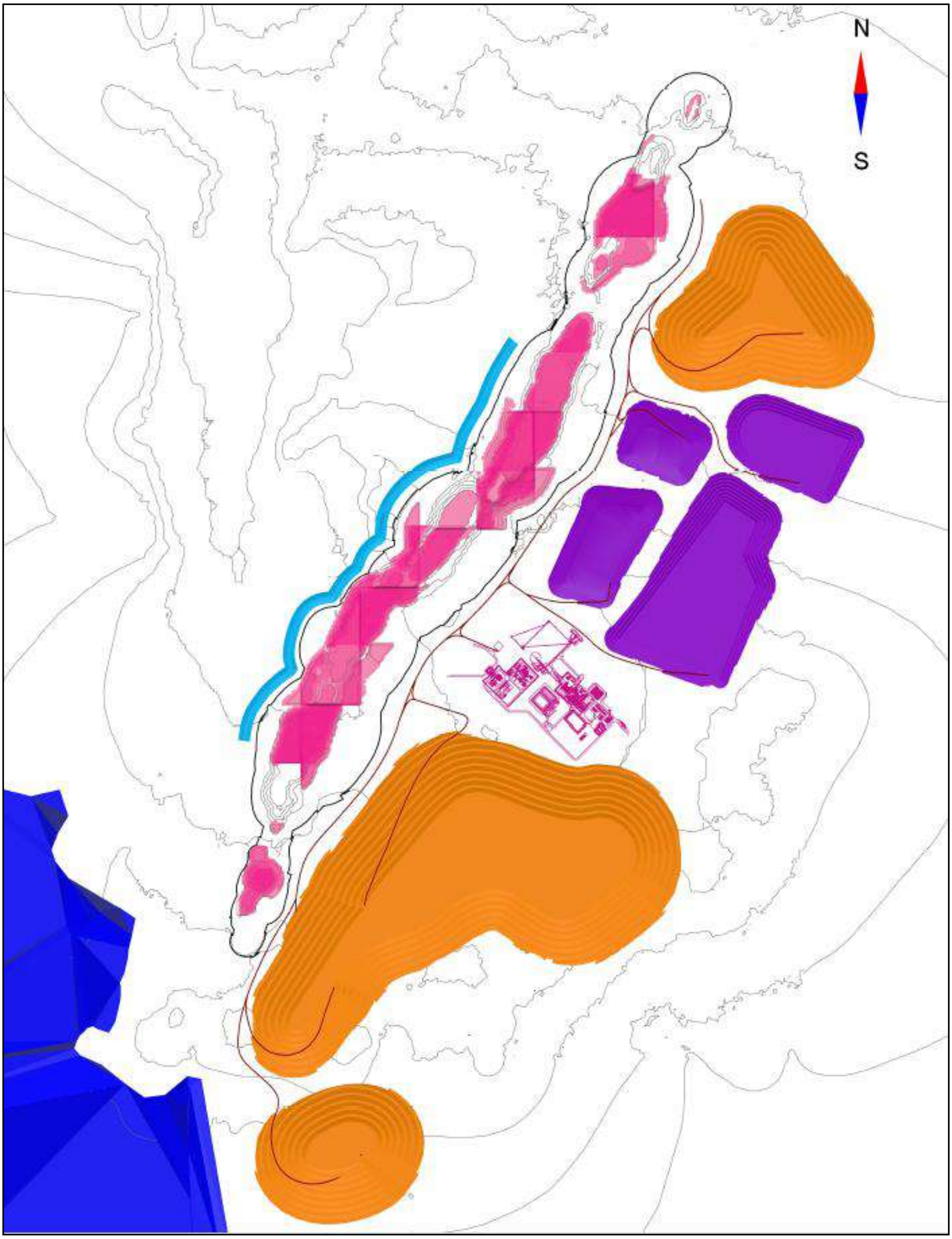
Source: DRA 2021

Figure 16.21: EOP Map Year 1 – Month 6

16.4.2.2 Year 2 to Year 11

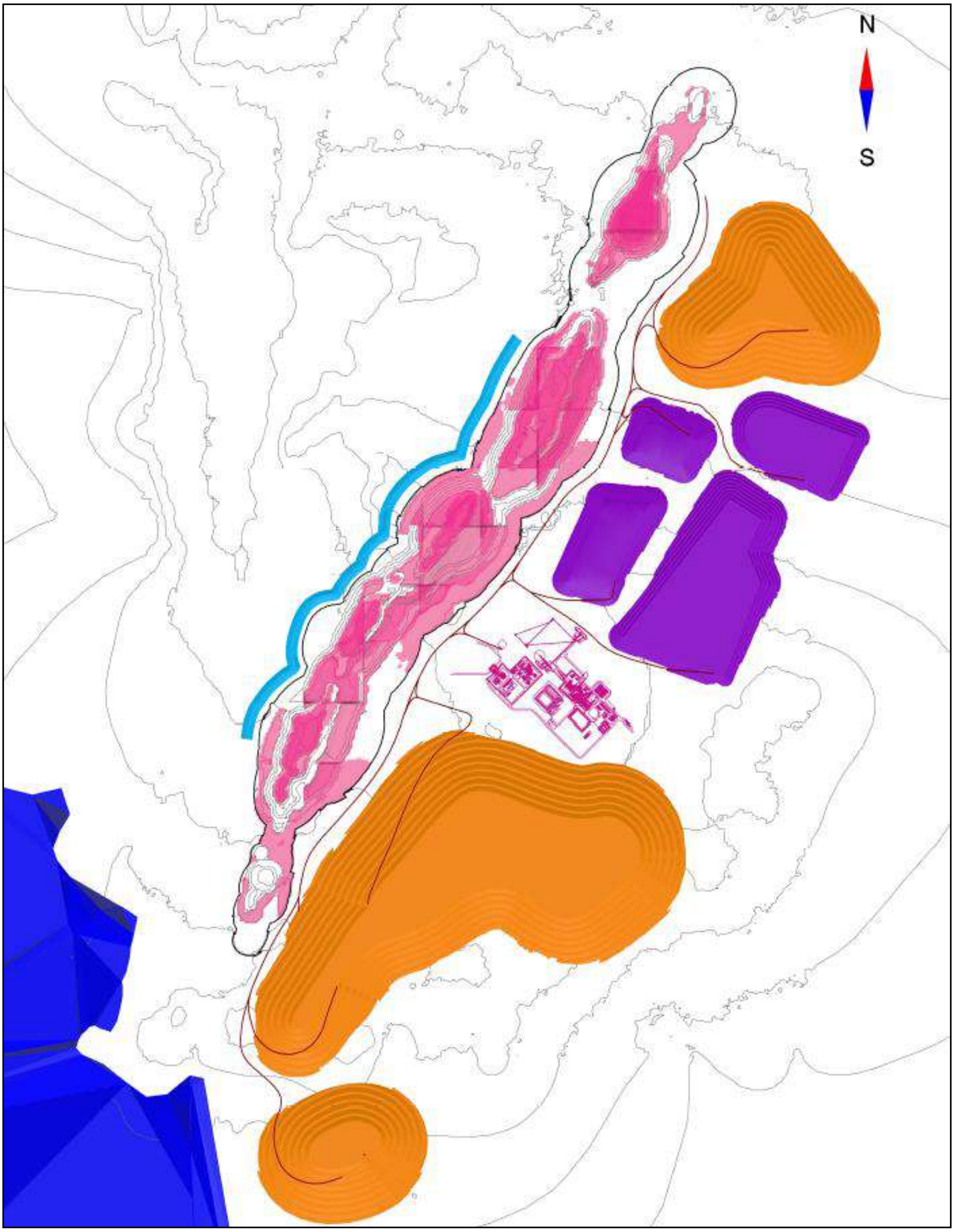
From Year 2 until Year 11, the mill will be fed directly from the mine using the high-grade material (greater than 0.6 g/t Au). All the material with a grade of between 0.35 g/t Au and 0.6 g/t Au will be sent to a low-grade stockpile.

The ore extraction rate during this period will exceed the mill capacity (3 Mt/a); therefore, the remaining high-grade ore material will be sent to a high-grade stockpile. Figure 16.22 to Figure 16.30 present the EOP for Years 2 to 10, respectively.



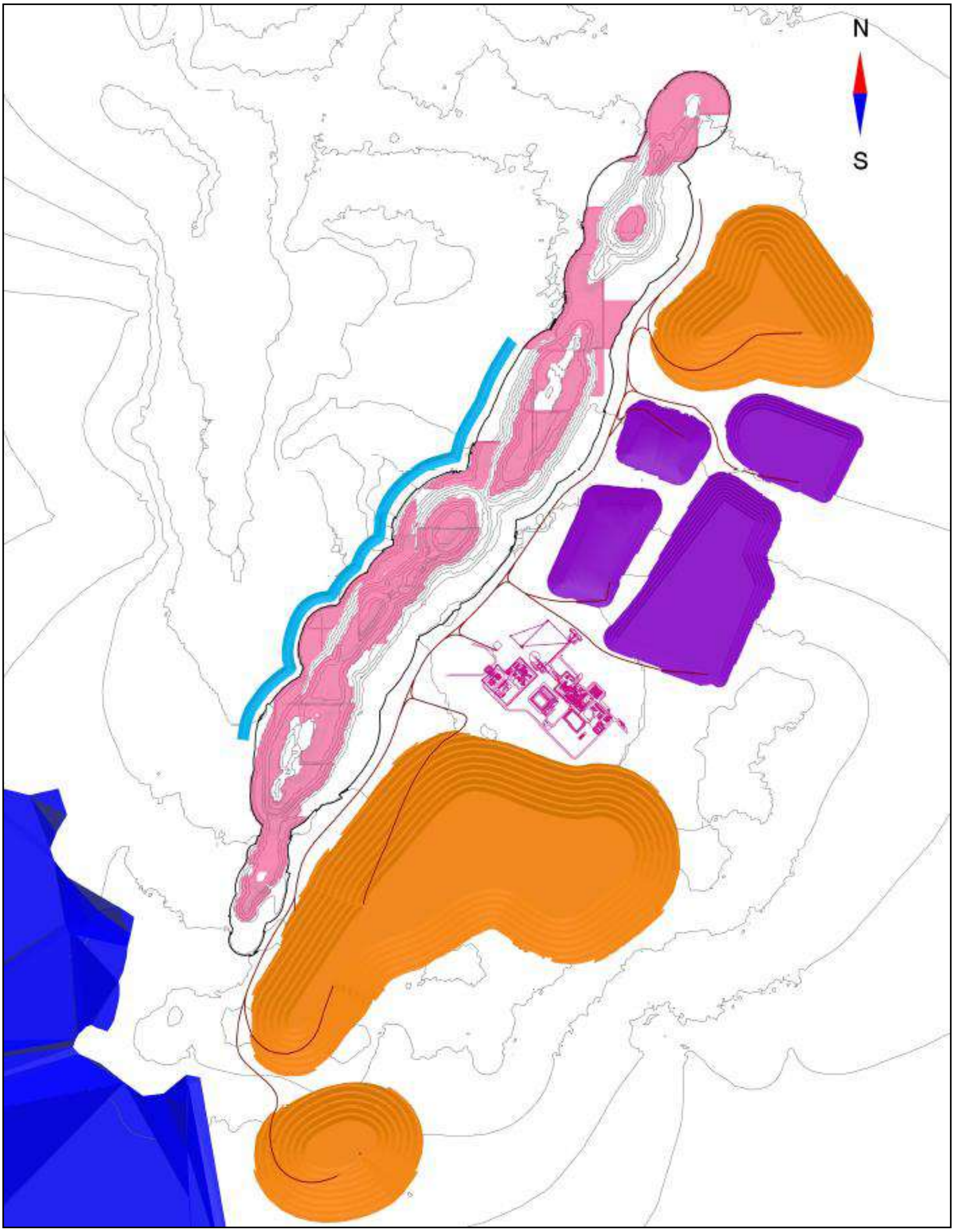
Source: DRA 2021

Figure 16.22: EOP Map Year 2



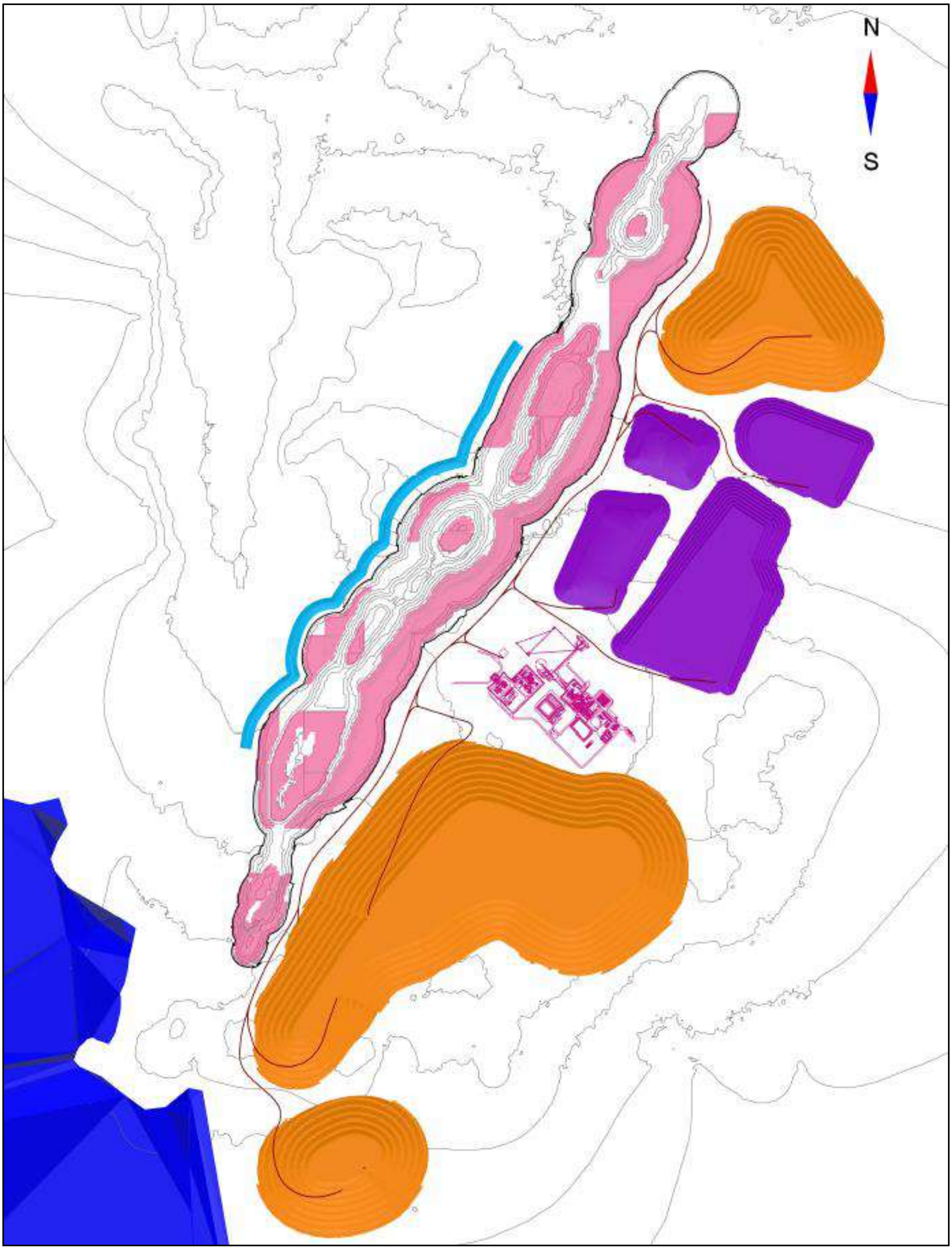
Source: DRA 2021

Figure 16.23: EOP Map Year 3



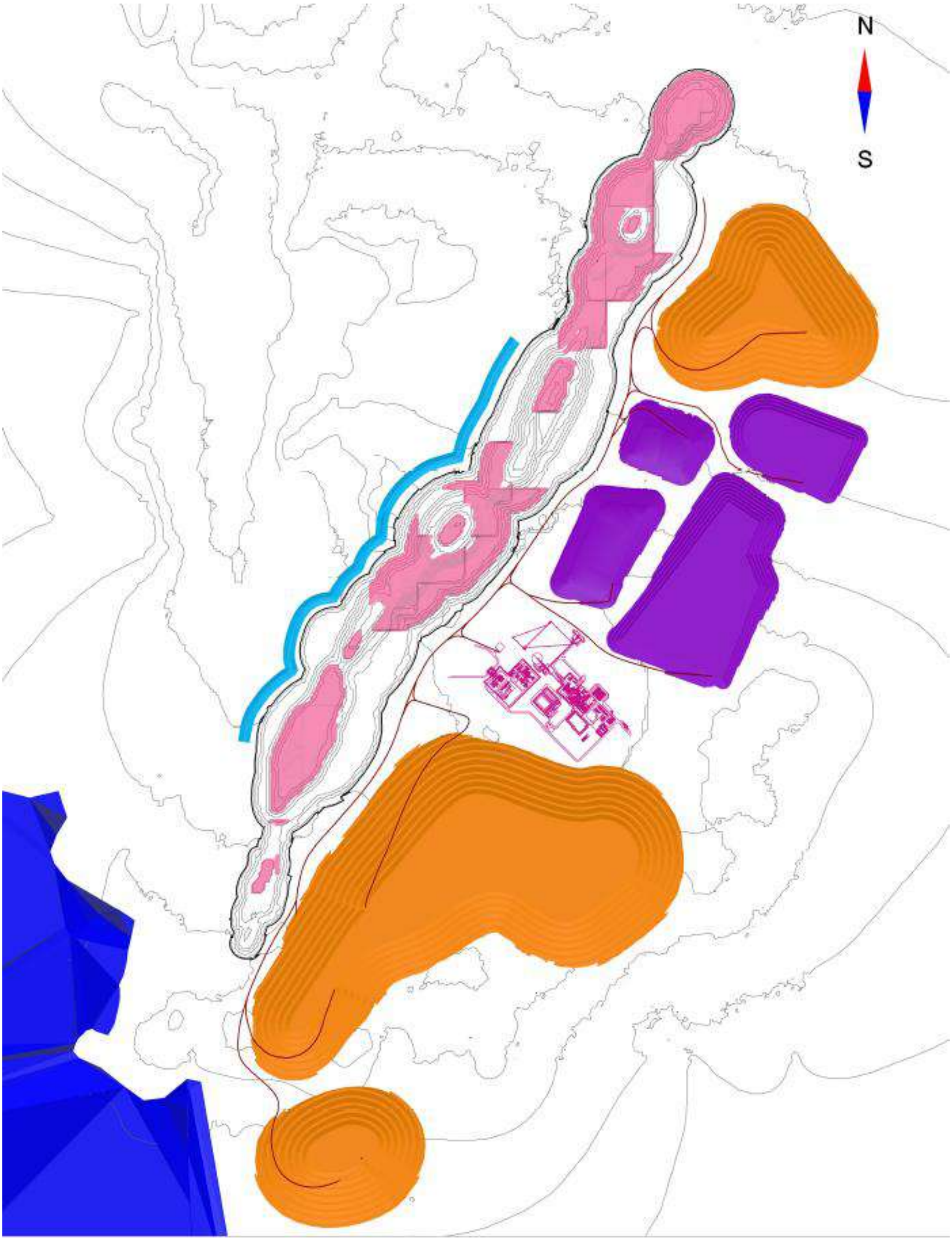
Source: DRA 2021

Figure 16.24: EOP Map Year 4



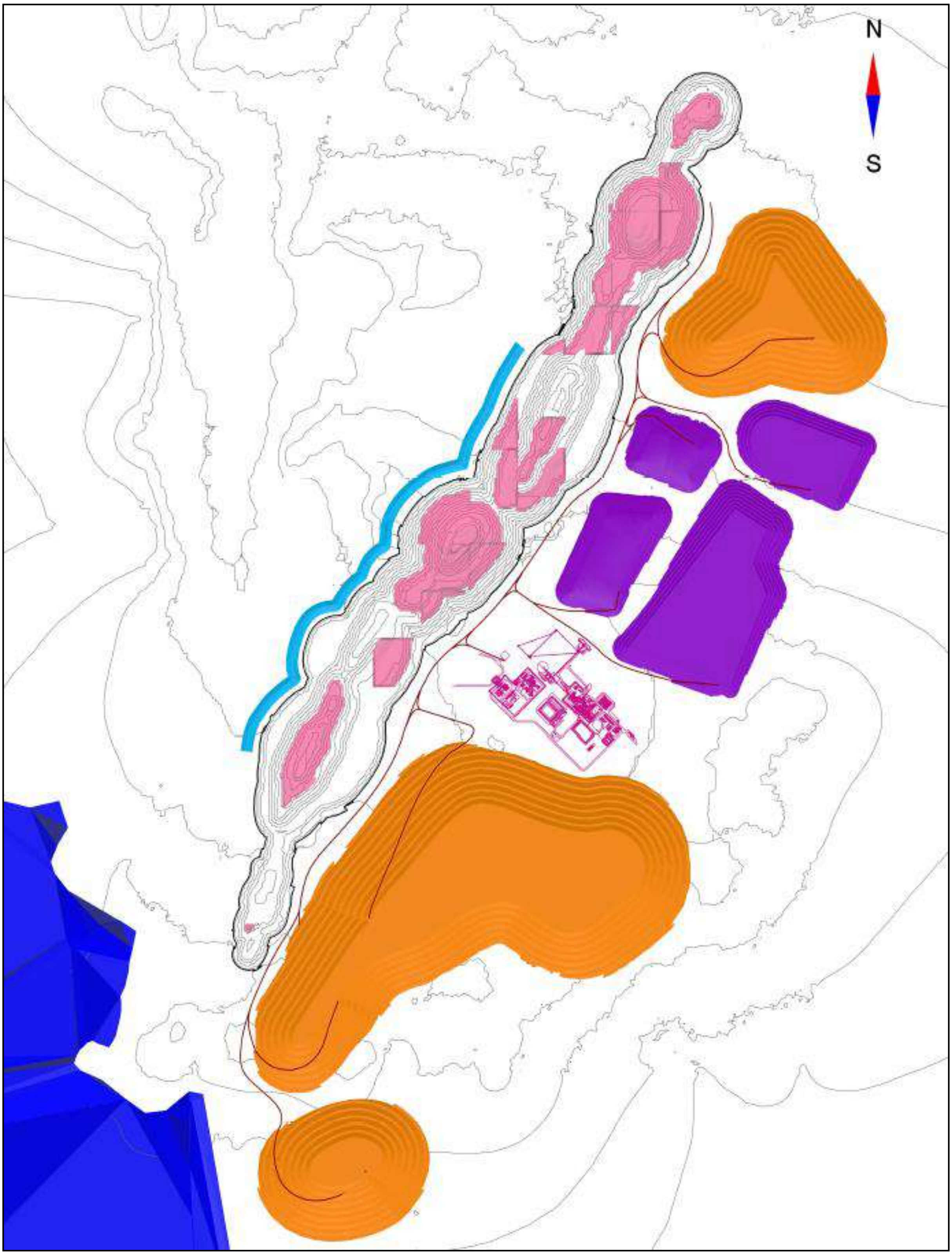
Source: DRA 2021

Figure 16.25: EOP Map Year 5



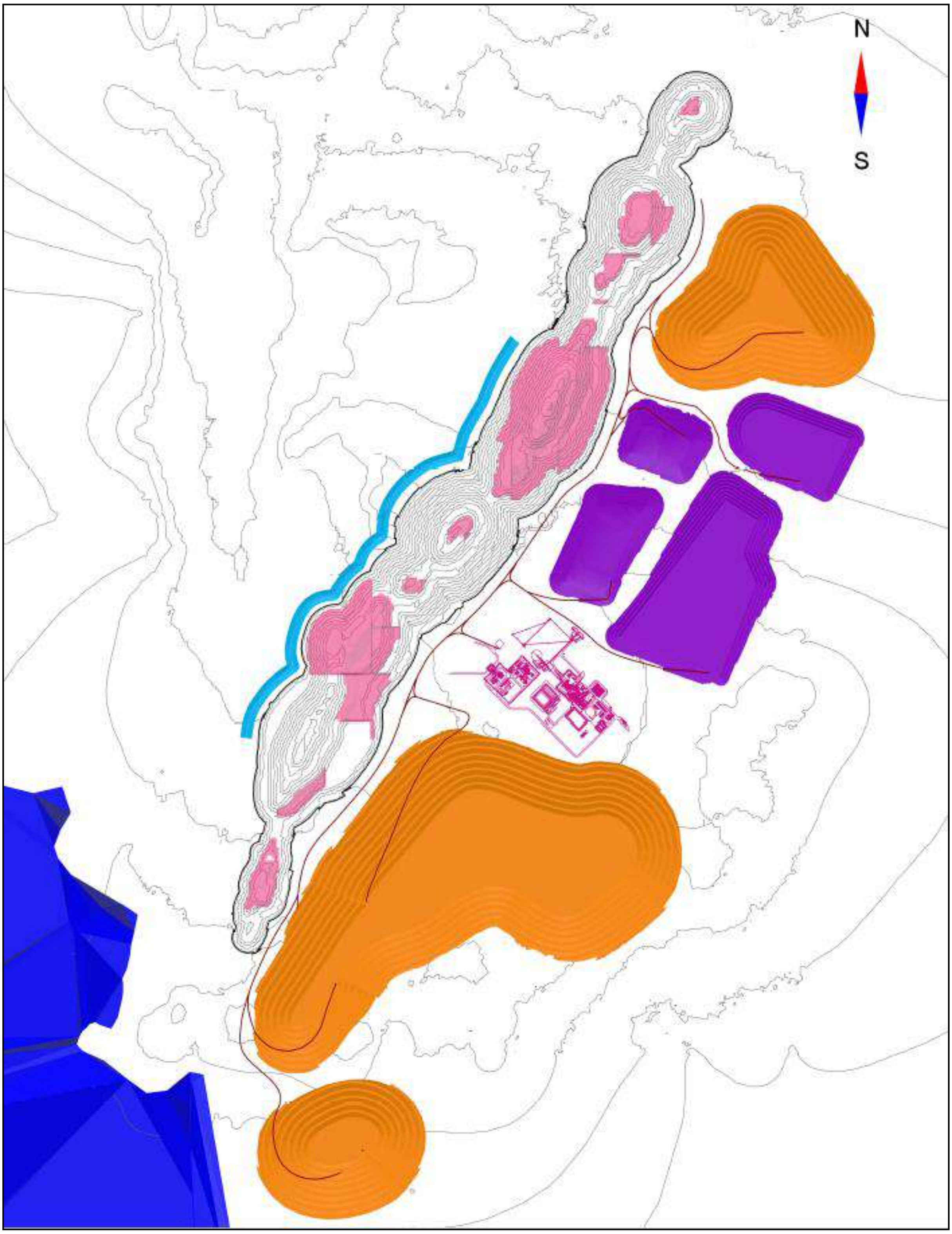
Source: DRA 2021

Figure 16.26: EOP Map Year 6



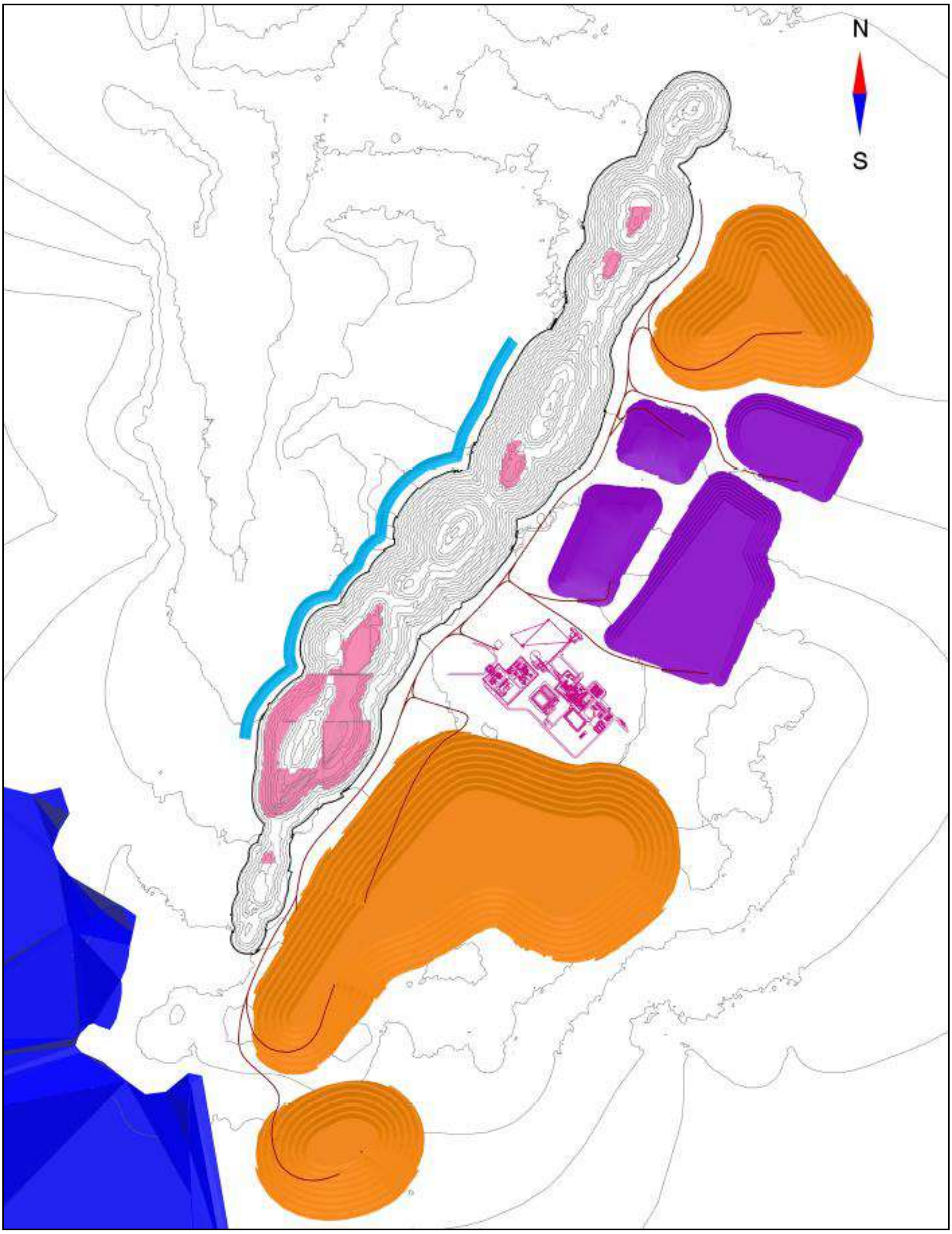
Source: DRA 2021

Figure 16.27: EOP Map Year 7



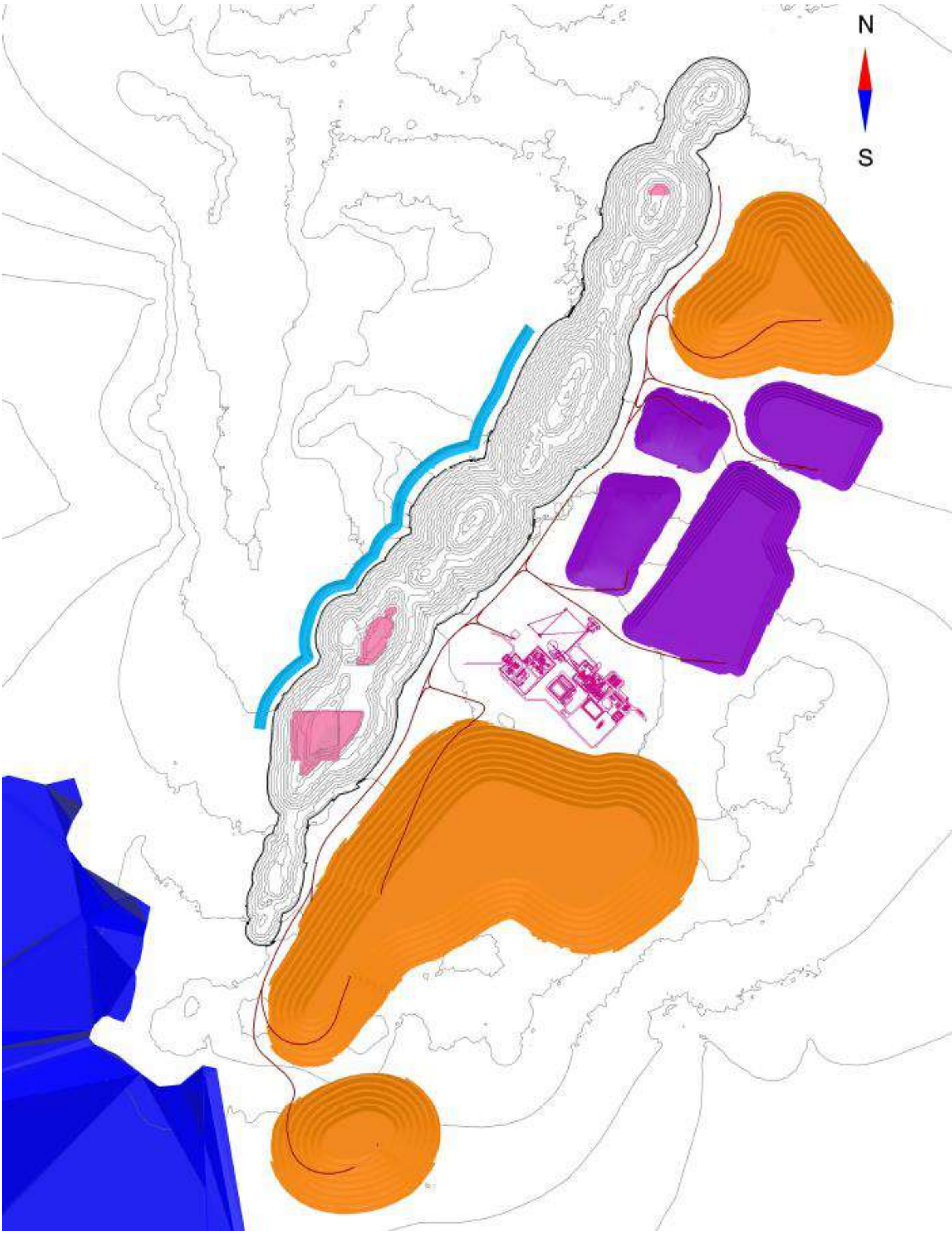
Source: DRA 2021

Figure 16.28: EOP Map Year 8



Source: DRA 2021

Figure 16.29: EOP Map Year 9



Source: DRA 2021

Figure 16.30: EOP Map Year 10

16.4.2.3 Year 12 to Year 16

The mining activities at the Kobada Open Pit will cease after Year 11 based on the current mine plan.

From Years 12 to 16, all the feed to the process plant will be reclaimed from the low-grade stockpile, thereby maintaining the 3 Mt/a feed rate. The high-grade stockpile will have been depleted by the end of Year 11.

16.4.2.4 Overall Mine Plan

Figure 16.31 shows the total material movement and grades for each period. Table 16.8 summarises the entire Kobada open-pit LOM mining plan, including the pre-production period.

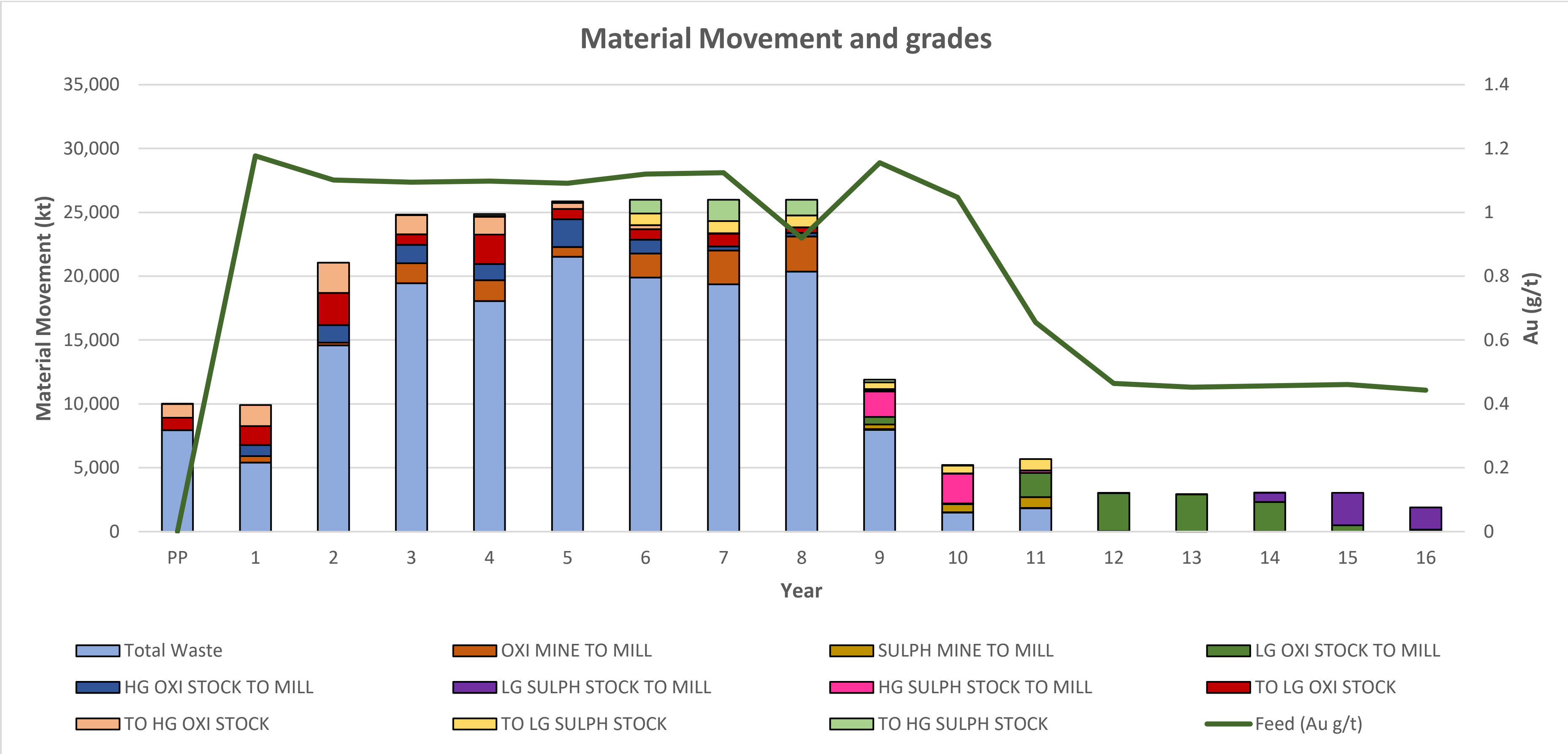


Figure 16.31: Total Material Movement and Grades

Table 16.8: Kobada Open-Pit LOM Mining Plan

Description	Unit	Pre-Production	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Total
Total Ore to Plant:																			
Tonnage	kt	0	1,377	2,997	3,007	2,906	2,930	2,964	2,961	3,030	2,996	3,059	2,957	2,998	2,918	3,031	3,041	1,858	**45,029**
Au	g/t	0.00	1.18	1.10	1.09	1.10	1.09	1.12	1.12	0.92	1.16	1.05	0.66	0.46	0.45	0.46	0.46	0.44	**0.87**
Mine to Plant																			
Tonnage	kt	0	493	1,607	1,571	1,632	763	1,883	2,640	2,763	408	657	877	14	0	2	0	40	**15,351**
Au	g/t	0.00	1.02	1.01	0.95	1.12	0.96	1.00	1.10	0.91	1.29	1.11	1.01	0.80	0.80	0.61	0.00	0.51	**1.02**
Stockpile to Plant																			
Tonnage	kt	0	884	1,390	1,436	1,274	2,166	1,081	321	267	2,587	2,402	2,080	2,983	2,918	3,029	3,041	1,818	**29,678**
Au	g/t	0.00	1.25	1.16	1.20	1.03	1.12	1.23	1.13	0.97	1.13	1.03	0.50	0.46	0.45	0.46	0.46	0.44	**0.77**
Ore to Oxide Hight-Grade Stockpile:																			
Tonnage	kt	1,086	1,650	2,372	1,493	1,392	485	306	36	0	75	0	0	0	0	0	0	0	**8,895**
Au	g/t	1.23	1.13	1.10	1.19	1.15	1.22	1.01	0.93	0.00	1.13	0.69	0.00	0.00	0.00	0.00	0.00	0.00	**1.15**
Ore to Oxide Low-Grade Stockpile:																			
Tonnage	kt	964	1,464	2,508	836	2,309	803	834	1,005	446	107	15	15	0	0	0	0	0	**11,305**
Au	g/t	0.45	0.46	0.46	0.47	0.46	0.46	0.46	0.46	0.45	0.46	0.48	0.47	0.00	0.00	0.00	0.00	0.00	**0.46**
Ore to Sulphide High-Grade Stockpile:																			
Tonnage	kt	0	0	0	34	135	78	1,085	1,672	1,241	216	38	0	0	0	0	0	0	**4,499**
Au	g/t	0.73	0.00	0.00	1.37	1.49	1.52	1.10	1.09	1.27	1.01	0.75	0.00	0.00	0.00	0.00	0.00	0.00	**1.16**
Ore to Sulphide Low-Grade Stockpile:																			
Tonnage	kt	0	0	0	15	69	39	907	957	928	542	615	886	23	0	1	0	0	**4,983**
Au	g/t	0.00	0.00	0.00	0.46	0.46	0.46	0.46	0.47	0.46	0.45	0.46	0.46	0.45	0.45	0.43	0.00	0.00	**0.46**
Total Waste	kt	7,951	5,411	14,588	19,446	18,051	21,528	19,891	19,363	20,354	7,973	1,482	1,827	21	0	1	0	31	**157,917**
Total Material Mined	kt	10,000	9,018	21,076	23,395	23,587	23,696	24,906	25,672	25,733	9,322	2,807	3,606	58	0	4	0	71	**202,951**
Total Material Moved	kt	10,000	9,903	22,466	24,831	24,862	25,862	25,986	25,993	26,000	11,909	5,209	5,686	3,041	2,918	3,033	3,041	1,889	**232,629**
Drilling Requirements	kt	500	455	1,088	1,853	3,851	2,191	6,175	10,793	14,518	8,760	2,807	3,593	58	0	4	0	71	56,718
Strip Ratio	-	3.88	1.50	2.25	4.92	3.26	9.93	3.97	3.07	3.78	5.91	1.12	1.03	0.55	0.55	0.35	2.83	0.77	3.51

16.4.2.5 Production Haulage and Cycle Time

For each production period, a haulage profile has been calculated by TALPAC 3D, a tool that generates haulage time estimates.

To use TALPAC, the haulage network as well as source and destination nodes are imported. Once the network is imported, the appropriate fleet is selected, and haulage times are determined for each source-destination combination. In this case, the haulage times were calculated using an A60H articulated truck. A maximum speed of 45 km/h and a rolling resistance of 3% was used.

Figure 16.32 shows a typical haulage network configuration for TALPAC.

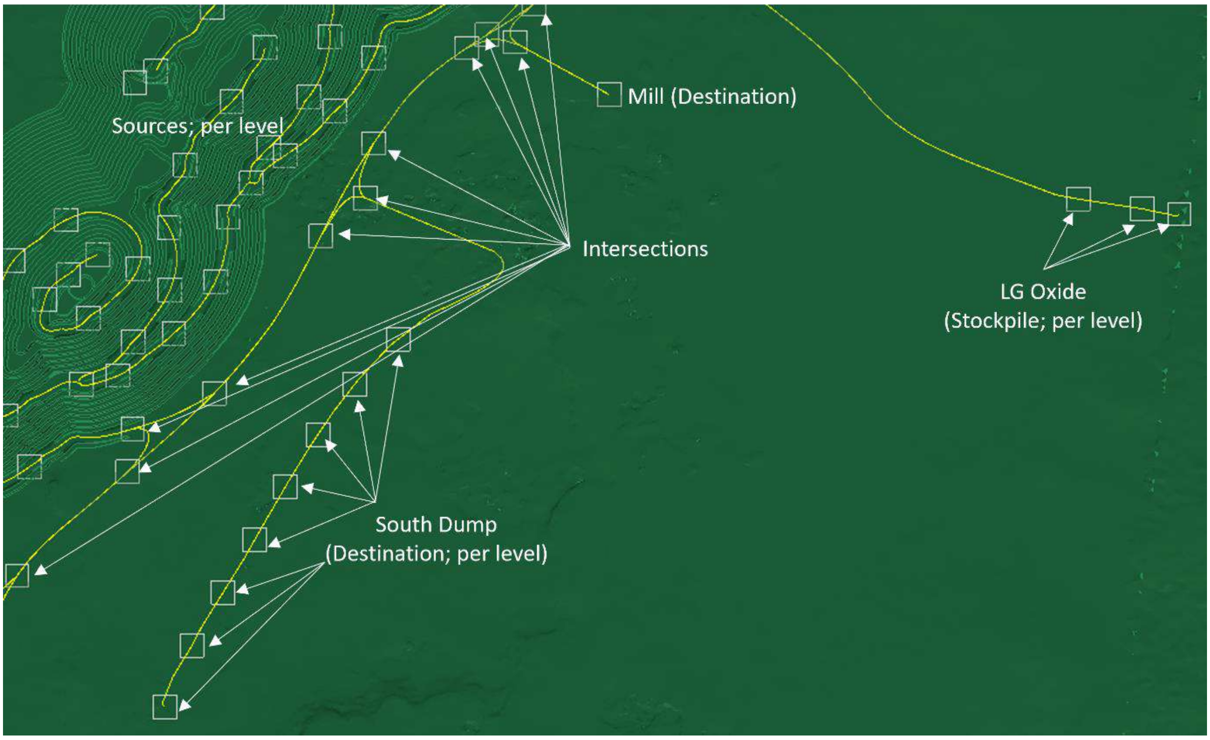


Figure 16.32: TALPAC Typical Haulage Road Set-Up

The haulage times determined by TALPAC were then used to calculate cycles times and fleet requirements. The equipment availabilities used for these calculations are listed in Table 16.9.

Table 16.9: Equipment Availabilities

Description	Unit	Trucks	Shovel	Loader	Drill
Mechanical Availability	%	85	85	85	85
Utilisation	%	90	90	90	80
Job Efficiency	%	90	90	90	90

16.4.2.6 Mining Equipment and Fleet

The mining equipment will consist of 120 t class conventional hydraulic shovels (PC1250 or similar) operating in a back-hoe configuration and 55 t class (Volvo A60H or similar) articulated off-highway trucks hauling on the designed access roads.

Front-end loaders (CAT988 or similar) will be used for stockpile rehandling and to work directly in the mine if necessary.

The backhoe excavator configuration will be used to allow selective mining of the ore zone in two flitches within the 5 m bench height. The front-end loader will be used to replace backhoe excavators during maintenance activities. Trucks will be loaded from the rear or from the sides, with the excavator loading from the bench above the truck to maximise productivities. It is possible that ore blocks will require top loading, which will require the excavator to be on the same level as the truck. However, this requirement will be minimised.

When necessary, blasthole drilling will be carried out using rotary drills (Epiroc D50 or similar), with hole diameters of 114 mm with an operating bench height of 5 m. A blasthole burden of 3.1 m and spacing of 3.5 m will be carried out using both emulsion and an ANFO (ammonium nitrate fuel oil) mixture.

To estimate the major equipment mining fleet, the basic working schedule presented in Table 16.4, the general schedule per shifts, and the expected shifts available (affected by estimated weather delays and/or holidays) must be considered. Operating time per shift represents the actual time during the shift that the equipment is productively working. The following operating delays under normal circumstances have been considered:

- Shift change: 20 min/shift
- Equipment inspection: 20 min/shift
- Lunch and breaks: 60 min/shift
- Fuelling, lubrication, and service: included at the end of shift (break)
- Total operating delays: 1.67 h/shift

The breakdown of hours has been calculated based on the general working schedule, affected by the operating delays described above, using the followed calculations:

- Total hours = $\frac{\text{Shift}}{\text{Years}} \times \frac{\text{Hours}}{\text{Shift}}$
- Hours down for maintenance = (100 % – Mechanical availability) × Total hours
- Available hours = Total hours – Hours down for maintenance
- Standby hours = (100 % – Utilisation) × Available hours
- Operating hours = Available hours – Standby hours
- Operating delay hours = $\frac{\text{Shift}}{\text{Years}}$ × Total operating delays × $\frac{\text{Operating hours}}{\text{Total hours}}$
- Net operating hours = Operating hours – Operating delay hours
- Working hours = Net operating hours × Job efficiency

Table 16.10 summarises the final breakdown of hours for the major equipment used for the Project.

Table 16.10: Breakdown of Hours

Description	Unit	Trucks/Shovel/Loader	Drill
Total Hours	h/a	8,232	8,232
Hours Down for Maintenance	h/a	1,235	1,235
Available Hours	h/a	6,997	6,997
Standby Hours	h/a	700	1,399
Operating Hours	h/a	6,297	5,598
Operating Delay Hours	h/a	875	777
Net Operating Hours	h/a	5,423	4,820
Working Hours	h/a	4,881	4,338

A support equipment fleet (such as dozers, graders, water trucks and utility vehicles), which is not directly responsible for production, will support the mining operation and be scheduled on a regular basis.

The track dozers are assigned to maintain the production areas and waste stockpiles, and to clean up the benches. Wheeled dozers, road graders and water trucks complete the remainder of the auxiliary equipment fleet.

Table 16.11 provides a summary of the suggested mining fleet for the Kobada operations. The equipment listed is used to develop, drill, blast, and haul material from the active mining levels.

Table 16.11: Suggested Equipment List

Description	PP	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
Major Equipment																	
Haul Truck	24	22	26	41	34	37	38	42	46	20	7	9	3	3	3	3	3
Shovel/Excavator	5	6	7	7	6	6	6	6	6	3	1	1	0	0	0	0	0
Production Drill	0	1	1	1	2	3	3	5	7	4	2	2	0	0	0	0	0
Wheel Loader	0	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2	2
Support Equipment																	
Track Dozer	2	4	4	4	4	4	4	4	4	4	4	1	1	1	1	1	1
Road Grader	1	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1
Water Truck	1	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1
Lighting Plant	4	5	5	5	5	5	5	5	5	5	5	2	2	2	2	2	2
Service Equipment																	
Fuel and Lubrication Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mechanics Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Boom Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tyre Handler	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Pickup Truck	4	4	4	4	4	4	4	4	4	4	4	2	2	2	2	2	2
Dewatering Pump	0	1	2	2	6	6	6	6	6	6	6	6	0	0	0	0	0

16.4.2.7 Loading Equipment

Hydraulic backhoes capable of loading 55 t haul trucks in four to five passes are considered the most appropriate loaders for this operation. Diesel-powered units are required for the Kobada Project due to the absence of a localised electrical network.

Hydraulic backhoes are particularly adept where:

- Selective mining is required to target identified layers of good mineralisation.
- Good clean up and recovery of ore is required below the normal operating levels.

16.4.2.8 Haulage

Mechanically driven 55 t articulated haul trucks have been selected as the appropriate haulage unit for both the hydraulic excavators and the wheel loaders to be used in the stockpile areas. These trucks are known to be efficient, dependable, and cost-effective in varied climatic and mining conditions over many years of operation.

Haul truck productivities were calculated on an annual basis from first principles, as explained in Section 16.4.2.5.

16.4.2.9 Road Construction and Maintenance

A comprehensive network of haul roads, minor roads, ramps, working areas and waste tipping areas will be maintained at a high standard of road repair by the two motor graders. The graders will concentrate on the main arterial haul roads while the track dozer will clean up the areas around the excavating units and the tipping areas on the waste stockpiles.

16.4.2.10 Grade Control

It has been assumed that dedicated Reverse Circulation (RC) drilling and sampling will be used for grade control. This activity will be contracted to a drilling contractor with operations in Mali.

The grade control samples will be sent to the laboratory located at the process plant and will be prepared and analysed using fire assay methodology.

Data will be processed, and blocks will be marked out as per the grade control guidelines, which will be developed prior to mining.

Ore will require in-pit geological monitoring due to the complexity of the orebody and, as far as is practical, only waste will be mined during the night shifts. Waste will be stripped off and the orebody will be exposed according to the instruction of the grade control geologist.

16.4.2.11 Ore Stockpiles and Waste Disposal

16.4.2.11.1 ROM Stockpiles

All the ROM material above the marginal COG and below 0.6 g/t Au COG will be stored in two stockpiles North of the process plant.

Each stockpile will have the following maximum capacity and height:

- Oxide Low-Grade (LG) stockpile: 7.5 Mm^3 with an average height of 20 m.
- Sulphide Low-Grade (LG) stockpile: 2.6 Mm^3 with an average height of 25 m.

All the ore material with a grade above 0.6 g/t Au that could not be sent directly to the processing plant due to crusher downtime or due to an excess of high-grade material will be stored in two additional stockpiles located close to the low-grade stockpiles. Each stockpile will have the following maximum capacity and height:

- Oxide High-Grade (HG) stockpile: 2.4 Mm^3 with an average height of 20 m.
- Sulphide High-Grade (HG) stockpile: 1.6 Mm^3 with an average height of 25 m.

The ROM material is planned to be stockpiled in 5 m high lifts. The ore stockpile layout is presented in Figure 16.33.

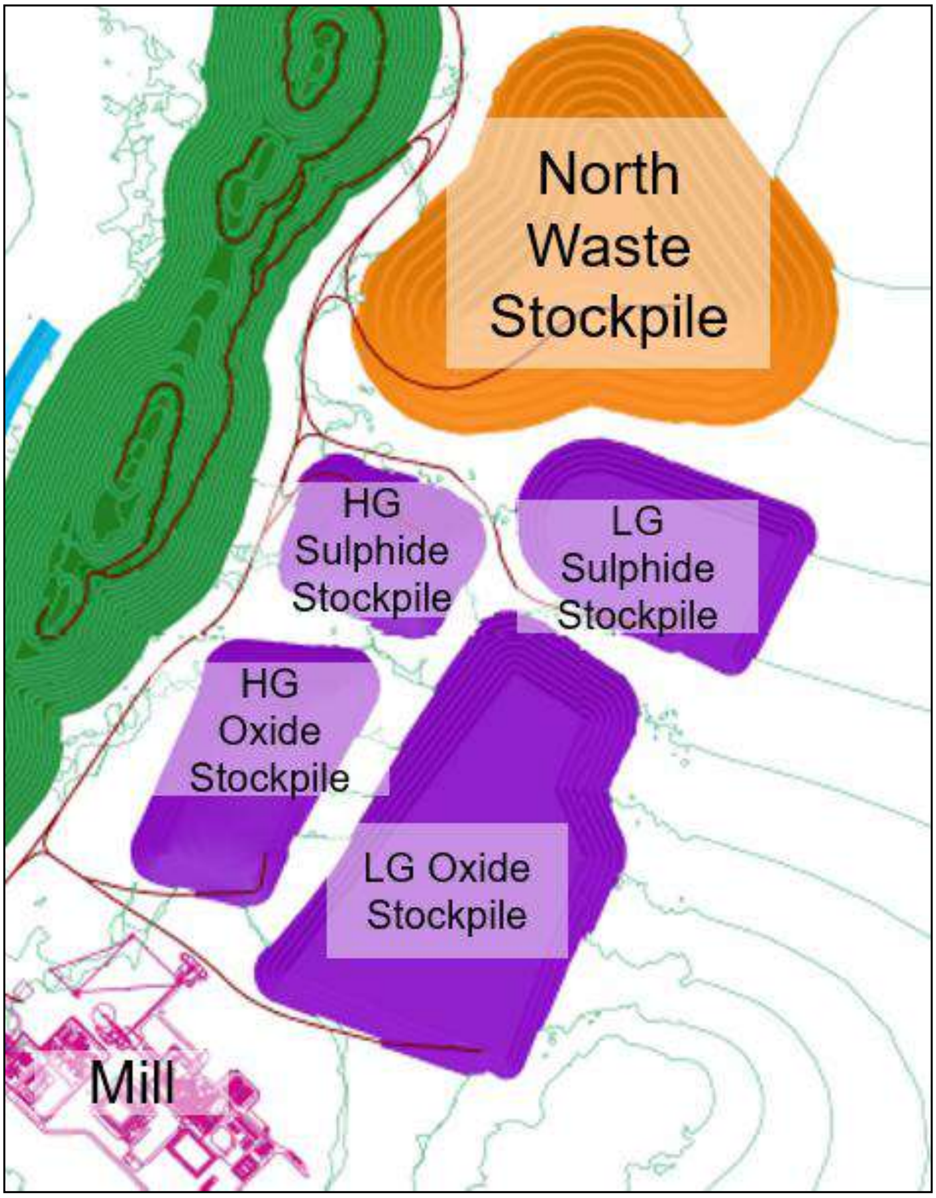


Figure 16.33: Ore Stockpile Layout

16.4.2.11.2 Waste Stockpiles

Three waste rock stockpiles have been designed on the east side of the pit. These stockpiles will contain all the material below the calculated marginal COG. The North Waste Stockpile has a maximum capacity of 19.9 Mm^3 with a maximum height of 70 m. The South Stockpile has a maximum capacity of 67.3 Mm^3 with a maximum height of 60 m. The South Waste Stockpile will be constructed in two phases: the main phase and the extension. The extension will be construction above the South Zone once its material has been fully extracted. The Sulphide Waste Stockpile has a capacity of 10.9 Mm^3 and will exclusively store sulphide waste material. This material is potentially acid generating and must be stored separately from other waste materials. The waste stockpile layout is presented in Figure 16.34.

These waste stockpiles will be constructed by end-tipping and dozing to a height of 10 m for each lift. A 35 % swell factor is used to estimate the required storage volumes in the stockpiles and waste dumps.

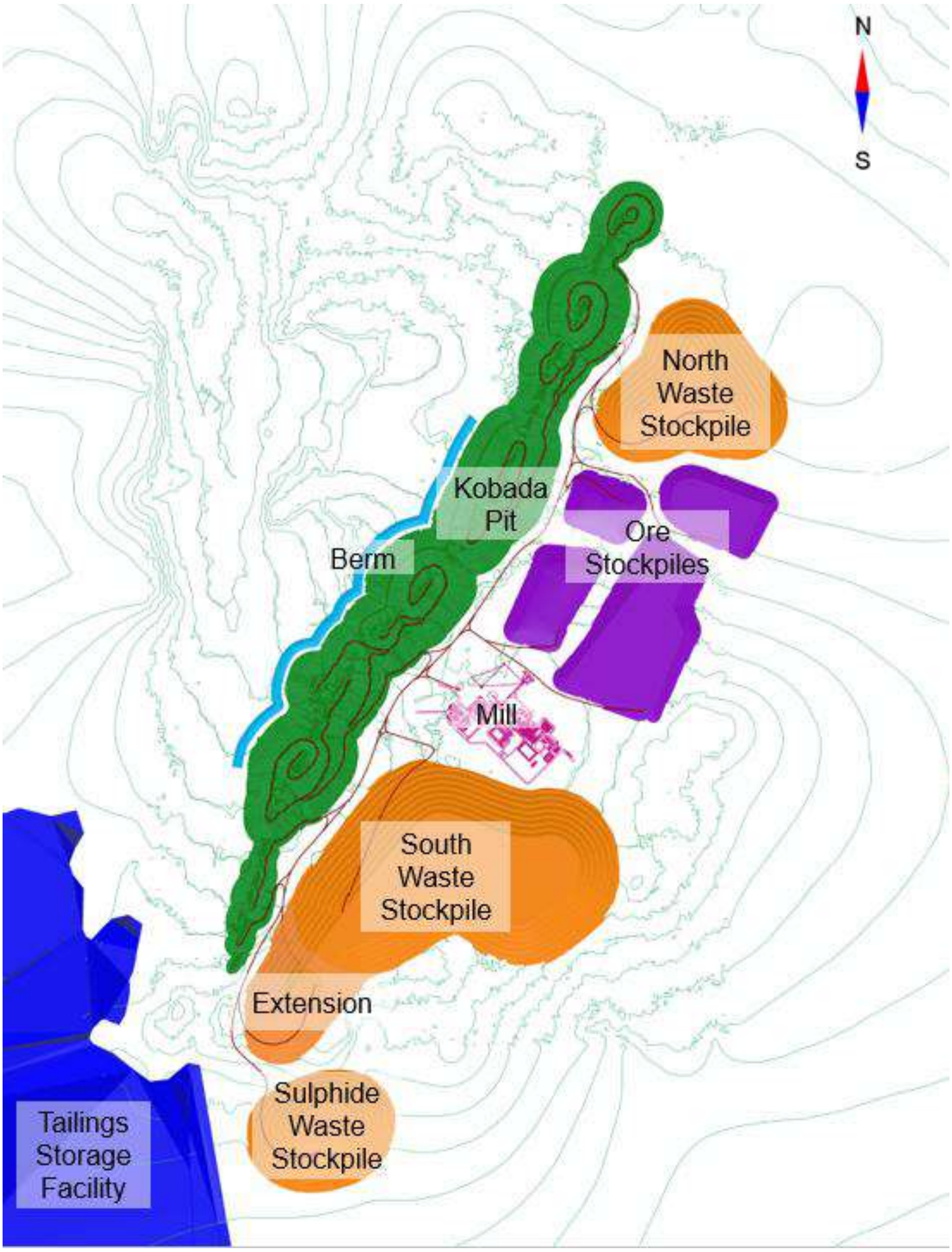


Figure 16.34: Waste Stockpile Layout

16.5 PIT DEWATERING

16.5.1 Outer Pit Dewatering

From the hydrogeological investigation, the steady-state groundwater inflow into the pit will be mitigated by the introduction of borehole pumps strategically positioned around the pit. To pump sufficient water volume from the outer pit borehole locations to the raw water storage dam located near the process plant, a total of eight borehole pumps with two dedicated transfer lines (four pumps per line) will be provided. A maximum water outflow of 96 m^3/h will be achieved with the use of the boreholes.

16.5.2 Inner Pit Dewatering

The pumping requirements to dewater the pits from rainfall and groundwater inflows were estimated following completion of the mine plan. The pit will require two diesel-powered pit dewatering systems at various identified positions in the pit. Each system will have its own dedicated dewatering transfer line to the raw water dam.

The initial dewatering system with a total combined pumping capacity of 500 m^3/h will be suitable for up to and including Year 3 and will subsequently need to be upgraded to account for the additional flow requirements and increased static head component.

In Year 4, an additional pit-bottom diesel pump system with the addition of the booster station comprising of a surge tank and 4 pumps (2 trains, duty/standby, with 2 pumps in parallel) to account for the increased flow requirement. The booster station will need to be located some 100m above pit bottom. The upgraded system will have a total combined pumping capacity of 870 m^3/h, which in conjunction with the pit borehole pumps should be sufficient to cater for life of mine (Y16), which requires a max dewatering capability of 895 m^3/h.

17 RECOVERY METHODS

17.1 PROCESS PLANT OVERVIEW

The Kobada gold processing plant is designed to process oxide ores from the three main deposits: the north, south and central ores during the first years of production followed by sulphides when the high-grade oxides are depleted. Towards the end of the mine life (Years 9-16) a mix of oxides and sulphides will be processed from stockpiles. Further exploration upside may result in additional oxides to be treated, which may defer the treatment of sulphides until later in the mine life.

The proposed process plant design is based on a well-proven and established gravity/carbon-in-leach (CIL) technology, which consists of crushing, milling, and gravity recovery of free gold, followed by leaching/adsorption of gravity tailings, elution and gold smelting, and tailings disposal. Services to the process plant will include reagent mixing, storage and distribution, and water and air services.

The plant will treat 3 Mt/a of saprolite/laterite/sulphide ore. The crushing circuit will be constructed in two phases: the first phase will be to treat oxide ore only, followed by a second phase later in the life of mine (LOM) to treat sulphides and/or a blend of sulphides and oxides. When treating oxide ore, only the primary crushing stage will be in use while three crushing stages will be required when treating the sulphide ore.

The milling circuit will also be implemented in two phases: the first phase will consist of a single-stage ball mill to treat oxides while the second phase will consist of an additional secondary ball mill, which will be installed in the later part of the LOM to treat sulphides. The circuit will consist of milling in closed circuit with a classification cyclone The discharge from the mill will discharge into the cyclone feed sump and will be pumped to the cyclone cluster for classification. A proportion of the cyclone underflow will be bled to the gravity circuit for recovery of gravity gold, with the balance gravitating to the ball mill for further size reduction. Gold will be recovered from the gravity concentrate through a combination of intensive cyanidation and electrowinning facilities. The gravity recovery tailings will be transferred back to the mill feed for further gold liberation. Gold that is not gravity recoverable is recovered through the CIL process.

The overflow from the cyclone cluster will feed the six-stage CIL circuit, where gold will be dissolved and adsorbed onto carbon. The Kobada ore exhibits varying degrees of preg-robbing as confirmed by the test work. The preg-robbing nature of the ores negates the use of a pre-leach tank. The resultant CIL tailings slurry will be subjected to a cyanide destruction and arsenic precipitation process, prior to being pumped to the tailings storage facilities.

Loaded carbon from the CIL circuit will be acid-washed prior to elution, followed by reactivation of the eluted carbon. The solution from the elution circuit will be subjected to electrowinning, where gold will be deposited onto cathodes as sludge. Periodically, the sludge will be washed off the cathodes and dried. The dried gold sludge will then be smelted to produce gold bullion, which will be shipped to the refinery.

A simplified flowsheet of the Kobada process plant is shown in Figure 17.1.

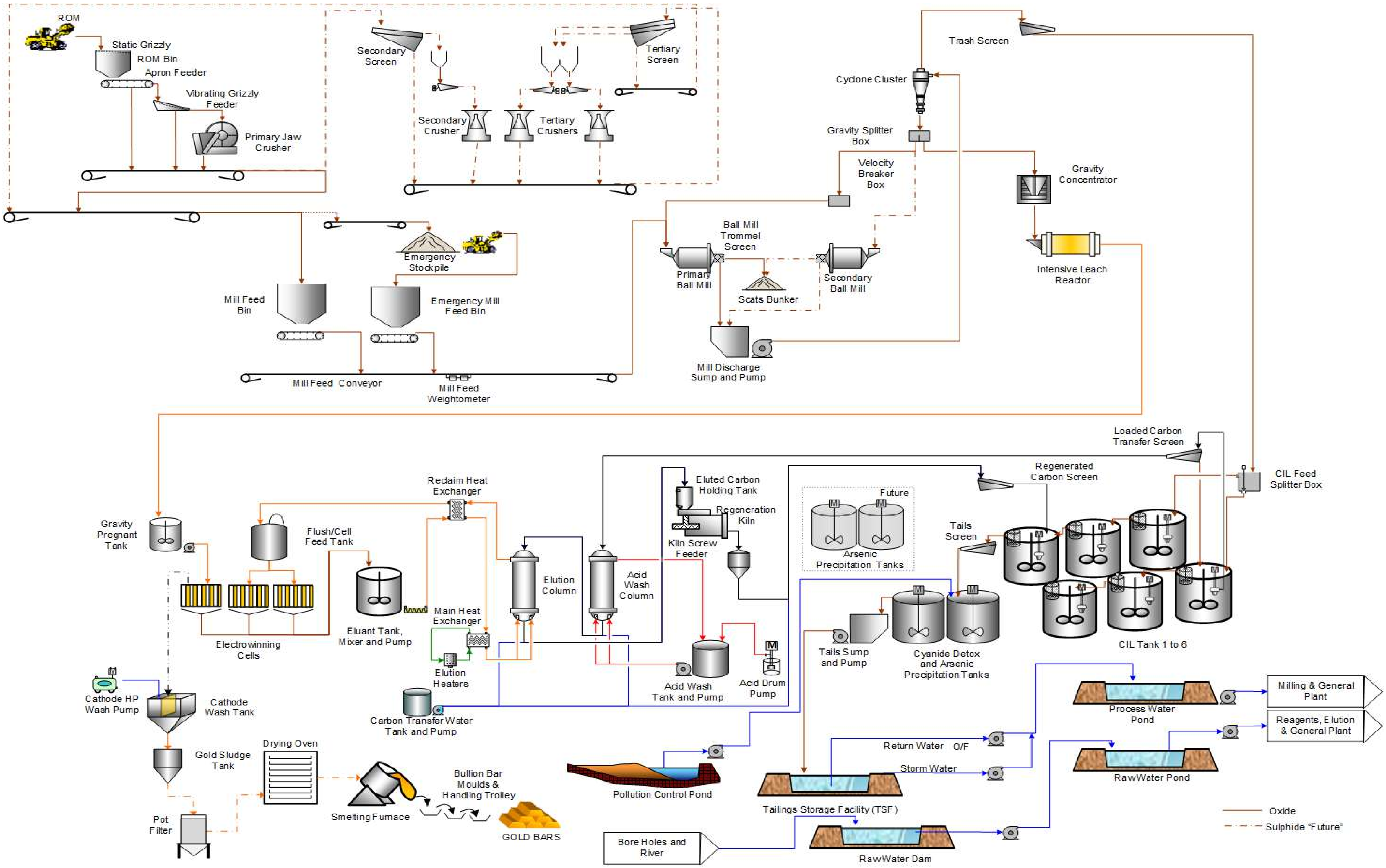


Figure 17.1: Kobada Gold Project Flowsheet

17.2 FLOWSHEET DEVELOPMENT

17.2.1 Introduction

The Kobada process plant was developed from the interpretation of the results of various test work programmes conducted by Gekko, MSA, SGS Canada laboratories on oxide and sulphide ores. This section describes the process flowsheet development of the plant that will treat 100 % oxide ore and various blends of oxide to sulphide ore.

In developing the process flowsheet, trade-off studies were conducted to optimise the process route selection. Specialised consultants, such as Orway Mineral Consultants (OMC), Peacocke & Simpson (P&S), and Kemix, were engaged to help with the flowsheet development in the areas of their expertise. The trade-off studies and simulations conducted included the following:

- Optimum process route selection (SENET)
- Comminution circuit (OMC)
- Gravity circuit (P&S)
- CIL and elution circuit (Kemix)

The conclusions from these trade-off studies and simulations led to the development of the flowsheet, which includes the following plant areas:

- Single-stage crushing facility utilising a primary crusher to treat the soft saprolite and laterite ores
- Additional secondary and tertiary crushing facility to treat sulphides
- Single-stage ball mill operating in an overflow arrangement with a ball-retaining ring to retain the media and reduce scatting when treating saprolite and laterite
- Additional secondary ball mill when treating sulphides and or blend of sulphides and oxides
- Gravity recovery and intensive cyanidation
- CIL (six CIL stages with no pre-leach tank to mitigate the preg-robbing properties of the ore)
- INCO air/SO_2 cyanide detoxification, arsenic precipitation and pumping of the detoxified tailings to the dam
- Acid wash
- Elution (pressurised Zadra method)
- Carbon regeneration
- Electrowinning and smelting
- Consumables and reagents
- Air and water services

17.2.2 Optimum Process Route Selection

In the feasibility study conducted in 2016 (Wolfe et al., 2016), which was conducted by Gekko, the process design was based on recovering gold through gravity means only, and other recovery options were not assessed. SENET undertook a trade-off study of other recovery methods to determine the optimum recovery route. Metallurgical test work to investigate recoveries using other possible process routes was conducted.

Four process route options were investigated, with each option treating 1.6 Mt/a ROM feed. The ROM feed selection was based on the 2016 feasibility study's annual throughput. Depending on the process route selected, different tonnages are processed in either leach or gravity.

The four process options that were assessed are as follows:

- **Option 1 – Gravity Only**
  This option treats 1.6 Mt/a ROM feed through crushing, scrubbing and desliming, resulting in less material being treated through gravity. For this option, SENET considered the mass rejection that was suggested by Gekko (high slimes rejection) and also what was determined through the test work conducted in 2019 (low slimes rejection).

- **Option 2 – 0.5 Mt/a CIL**
  This option treats 1.6 Mt/a ROM feed through crushing, scrubbing and desliming, resulting in less material (approximately 0.5 Mt/a) being treated through gravity and CIL. For this option, SENET considered the mass rejection that was suggested by Gekko (high slimes rejection).

- **Option 3 – 1.6 Mt/a Gravity and CIL (Whole Ore)**
  This is a conventional cyanide leach process route, where the full 1.6 Mt/a ROM ore undergoes the crushing, milling, gravity, CIL and adsorption, desorption, and recovery (ADR) processes. The desliming process is not part of this option, hence, this option has no rejection of fines.

- **Option 4 – 1.6 Mt/a Heap Leach (HL)**
  This is a conventional cyanide HL process route, where the full 1.6 Mt/a ROM ore undergoes the crushing, agglomeration, cyanide HL, and ADR processes. This option, as with Option 3, has no rejection of fines.

Capital and operating costs for each scenario were determined, and a high-level economic analysis was conducted using the 2016 mining plan. Results of the economic analysis are summarised in Table 17.1.

Table 17.1: Summary of Economic Analysis

Description	Unit	Gravity Only	0.5 Mt/a CIL	1.6 Mt/a CIL	1.6 Mt/a HL
LOM Tonnage Ore Processed	kt	12,735	12,735	12,735	12,735
LOM Feed Grade Processed	g/t	1.14	1.14	1.14	1.14
LOM Gold Recovery[a]	%	77.0	86.3	96.0	79.2
LOM Gold Production	koz	393.5	441.3	490.8	404.9
Gold Price	US$/oz	1,450	1,450	1,450	1,450
Revenue	US$ million	571	640	712	587

Description	Unit	Gravity Only	0.5 Mt/a CIL	1.6 Mt/a CIL	1.6 Mt/a HL
LOM Operating Costs	US$/oz	680	651	601	785
Total Capital Costs	US$/oz	242	229	219	247
Total LOM Production Costs	**US$/oz**	**922**	**881**	**820**	**1,033**
Pre-Tax NPV	US$ million	141	173	206	50
IRR	%	34.8	38.2	40.7	18.4
Discounted Payback Period	Years	3.39	3.17	3.04	6.04
Project Net Cash Flow Pre-Tax	US$ million	209.8	253.2	298.6	87.3
[a] Results from MMS Test Work					

From the high-level economic analysis, it was established that the whole-ore CIL option (Option 3 in the trade-off study) was the most economically viable option, and as such it was selected as the preferred processing route.

17.2.3 Comminution – Oxide Ore

OMC was requested to conduct simulations based on the comminution test work results. The results are summarised in Table 17.2 (see Section 13.1.5 for the detailed results). The simulations were conducted at 170 t/h to give a product of P_{80} passing 150 µm.

Table 17.2: Comminution Test Work Data

Parameter	Unit	Value
Ai	g	0.103
CWi	kWh/t	6.6
BRWi	kWh/t	6.6
Scrubber Energy	kWh/t	0.7
BBWi – Whole Ore	kWh/t	1.1
BBWi – Whole Ore (+1 mm)	kWh/t	12.6
Percentage of +1 mm in the ROM Feed	%	55
SG		2.5

Three circuit configurations were investigated:

1. Scrubbing Circuit – This will consist of crushing using a mineral sizer, and the crushed product will feed the scrubber. The scrubber will discharge onto the scrubber discharge screen where the screen oversize is conveyed to a secondary crusher operating in closed circuit with a secondary screen. The scrubber discharge screen undersize is pumped to a cyclone cluster. The secondary crushed material and cyclone underflow will be treated in a secondary ball mill, and discharge from the mill will combine with the scrubber screen undersize and be pumped to the cyclone. The cyclone overflow will gravitate to the CIL circuit.
2. Single-Stage SAG Mill Circuit – This will consist of crushing using a mineral sizer, and the crushed product will feed the single-stage SAG mill. The mill discharge will be

pumped to the cyclone, and the cyclone underflow will be recycled back to the mill while the overflow gravitates to the CIL circuit.

3. Single-Stage Ball Mill Circuit – This will consist of crushing using a mineral sizer, and the crushed product will feed the ball mill. The mill discharge will be pumped to the cyclone, and the cyclone underflow will be recycled back to the mill while the overflow gravitates to the CIL circuit.

Table 17.3 summarises the sizing of the equipment suitable for each comminution option

Table 17.3: Simulated Comminution Circuit Options

Parameter	Unit	Scrubbing Circuit	Single-Stage SAG Circuit	Single-Stage Ball Mill Circuit
Primary Crusher				
Type		Mineral Sizer	Mineral Sizer	Mineral Sizer
Model		MMD 500	MMD 500	MMD 500
Scrubber				
Diameter	m	3.00	N/A	N/A
Effective Grinding Length	m	6.50	N/A	N/A
Installed Power	kW	300	N/A	N/A
Scrubber Screen				
Width × Length	m × m	1.87 × 4.90	N/A	N/A
Aperture	mm	10	N/A	N/A
Secondary Crusher				
Model		HP200	N/A	N/A
Installed Power	kW	132	N/A	N/A
Crusher Screen				
Width × Length	m × m	1.87 × 4.90	N/A	N/A
Aperture	mm	10	N/A	N/A
Ball/SAG Mill				
Diameter	m	3.60	5.5	3.80
Effective Grinding Length	m	5.60	2.2	5.90
Discharge Arrangement		Overflow	Grate	Overflow
Installed Power	kW	1,100	1,400	1,400
Milling Circuit Classification				
Type		Cluster	Cluster	Cluster
Model		400 CVX10-111	400 CVX10-111	400 CVX10-111
Diameter	mm	400	400	400
Number Installed		8	8	8

The three comminution circuits were then assessed taking into account the following:

- Process risk
- Availability and maintenance flexibility

- Power efficiency
- Operating cost
- Capital cost
- Operating flexibility

The scrubbing circuit will have the highest capital cost due to the increased number of equipment required. Maintenance of all the equipment will also not be easy. The availability of the secondary crushing circuit is likely to be low, and this circuit does not offer operating flexibility. Both the single-stage SAG and single-stage ball mill circuits will be easy to operate and maintain, and the capital costs will be low because a reduced number of equipment is required. As noted by OMC: "*The biggest risk with SS SAG milling circuit is slurry pooling due to flow limitations through the grate discharge, at which it is expected that the risk can be mitigated by installing overflow ball mill with a ball retainer to minimise the scatting and retain the larger rock/ore in the mill.*"

The selected comminution circuit is thus the primary crushing single-stage ball mill. Using this information, further simulations were conducted at 380 t/h to give a product of P_{80} passing 150 µm for a milling plant capable of 100 000 oz of gold per annum. Table 17.4 summarises the findings of the single-stage ball mill sizing.

Table 17.4: Selected Comminution Circuit

Parameter	Unit	Specification
Mill Diameter (Inside Shell)	m	4.70
Effective Grinding Length	m	7.75
Imperial Measurements	ft × ft	15.4 × 25.4
Discharge Configuration		Overflow
New Liner Thickness	mm	80
Grinding Media	mm	100
Mill Speed	% Nc	75
	rpm	14.9
	Range % Nc	60 to 80
Ball Charge – Duty	%	26
Ball Charge – Maximum	%	34
Pinion Power – Duty	kW	2 183
Pinion Power – Maximum at 80 % Nc	kW	2 780
Selected Motor Size	**kW**	**2 900**

17.2.4 Comminution – Sulphide Ore

OMC was contracted to conduct comminution simulations on the sulphide ore. Three comminution options (see Figure 17.2) were investigated. For each option, two product size possibilities, which resulted in different overall plant recoveries (P_{80} passing 75 µm = 94.72 % and P_{80} passing 106 µm = 92.14 %) were looked at.

The options were as follows:

- **Option 1A/B**: Determine how much sulphide throughput can be achieved by adding a secondary and tertiary crushing circuit prior to the existing ball mill to produce a P_{80} passing 75 µm and 106 µm, respectively.
- **Option 2A/B**: Introduce additional secondary and tertiary crushing and an additional ball mill (along with the existing mill) to be used as a secondary grinding facility to treat 3 Mt/a of sulphides at a grind of P_{80} passing 75 µm and 106 µm, respectively.
- **Option 3A/B**: Introduce a SAG mill that will be used as a primary grinding facility and use the existing ball mill for secondary grinding to treat 3 Mt/a of sulphides at a grind of P_{80} passing 75 µm and 106 µm, respectively.

Figure 17.2 shows the comminution flowsheet options.

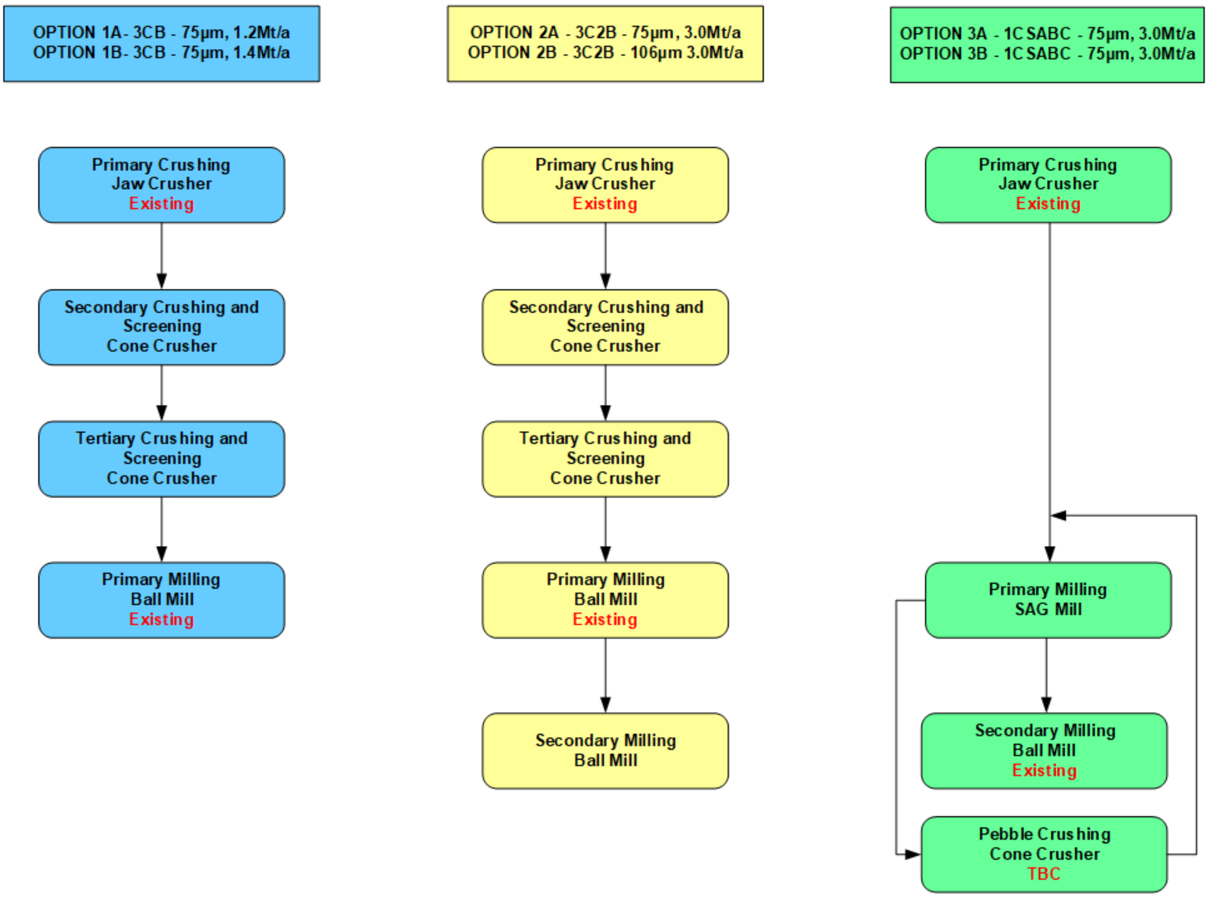


Figure 17.2: Comminution Flowsheet Options

The comminution circuits were then assessed taking into account the following:

- Process risk
- Availability and maintenance flexibility
- Power efficiency
- Operating cost

- Capital cost
- Operating flexibility

The CAPEX comparison showed that Option 3A has the highest CAPEX. Options 3B and 2A have the second highest CAPEX, with very marginal cost differences between the two options, followed closely by Option 2B. The lowest CAPEX options are Option1A and 1B.

From the OPEX assessment, Option 3, the SAG mill option, showed the highest OPEX, at US$7.17/t and US$6.24/t for the 75 µm and 106 µm product options, respectively. The SAG mill option is, therefore, eliminated as a possible circuit. Options 1 and 2 have similar costs per tonne for both the 75 µm and 106 µm product options; however, in both instances, Option 1 showed a slightly lower OPEX.

Therefore, Option 1 (additional secondary and tertiary crushing) is recommended based on the lower initial CAPEX as well as the lower OPEX. However, to maintain the required design throughput of 3 Mt/a when treating sulphides, the three-stage crushing and secondary milling option 2A is recommended.

17.2.5 Gravity Circuit Simulation

The EGRG test work on oxides showed a GRG value of 80.2 %, but the majority of the GRG is less than 38 µm in size. This fine-sized GRG presents a challenge in gravity recovery as most gravity units cannot efficiently recover ultrafine particles. P&S was requested to run a simulation in order to predict the expected gravity gold plant recoveries.

The model predicted a GRG recovery of 34.8 % when treating the bleed of the cyclone underflow. The GRG recovery when treating the mill discharge was also modelled, and this resulted in a slightly higher recovery at 42.7 %. In addition, various grind sizes were modelled as shown in Table 17.5.

Table 17.5: Oxide GRG Simulation Results

Grind P_{80} (µm)	Cyclone Underflow GRG Recovery (%)	Mill Discharge GRG Recovery (%)
150 (base)	34.8	42.7
125	36.3	43.7
100	38.0	44.8
75	42.7	48.5

17.2.6 Carbon in Leach – Elution Batch Size

The leach residence time for both gravity middlings and tailings and ROM feed ore was determined to be 16 h (see Section 13).

To determine the number of CIL stages that is required to give a solution tail of less than 0.015 ppm gold exiting the CIL circuit, the leach kinetic curves for both oxides and sulphides in Figure 17.3 was used to show the following:

- Gravity tailings, which represent the feed to the CIL circuit when the gravity circuit is in use
- Whole ore/ROM, which represents the feed to the CIL circuit when the gravity circuit is being bypassed

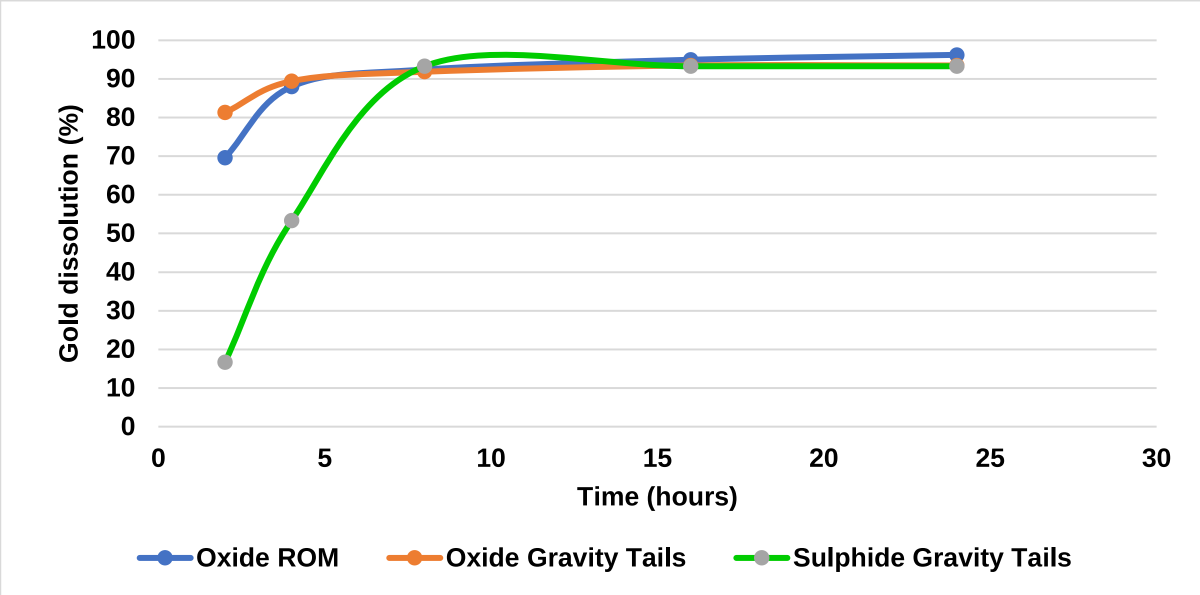


Figure 17.3: CIL Kinetics

From Table 17.6 and Table 17.7 it is evident that a six-stage CIL processing facility will be adequate to result in solution tails (highlighted in table) of less than 0.015 ppm exiting the circuit.

Table 17.6: Oxide Gravity Tails Leach Results per CIL Stage

Stage	Residence Time (h)	Ore (g/t)	Cumulative Dissolution (%)	Adsorption Efficiency (%)	Au in Solution (g/t)	Au onto C (g/t)	Au in Solution-Stage (Exit ppm)
Feed		0.81					
1	2.7	0.130	84	70	0.680	0.476	0.1099
2	2.7	0.081	90	65	0.253	0.164	0.0476
3	2.7	0.068	91.55	60	0.101	0.061	0.0218
4	2.7	0.065	92	60	0.044	0.026	0.0095
5	2.7	0.055	93.20	55	0.027	0.015	0.0066
6	2.7	0.054	93.34	50	0.013	0.007	0.0036

Table 17.7: Sulphide Gravity Tails Leach Results per CIL Stage

Stage	Residence Time (h)	Ore (g/t)	Cumulative Dissolution (%)	Adsorption Efficiency (%)	Au in Solution (g/t)	Au onto C (g/t)
Feed		1.30				
1	2.7	1.088	83.7%	70	0.326	0.76
2	2.7	0.423	91.1%	65	0.148	0.28
3	2.7	0.170	92.8%	60	0.068	0.10
4	2.7	0.068	92.8%	60	0.027	0.04
5	2.7	0.030	93.0%	55	0.013	0.02
6	2.7	0.013	93.0%	50	0.007	0.01

Table 17.8 shows the leach kinetics for the ROM feed (whole ore) leach tests.

Table 17.8: ROM Feed Leach Results per CIL Stage

Stage	Residence Time (h)	Ore (g/t)	Cumulative Dissolution (%)	Adsorption Efficiency (%)	Au in Solution (g/t)	Au onto C (g/t)	Au in Solution-Stage (Exit ppm)
Feed		1.25					
1	2.7	0.388	69	70	0.863	0.6038	0.1393
2	2.7	0.175	86	65	0.471	0.3063	0.0888
3	2.7	0.096	92.33	60	0.244	0.1464	0.0526
4	2.7	0.078	93.80	60	0.116	0.0696	0.0250
5	2.7	0.065	94.80	55	0.059	0.0324	0.0143
6	2.7	0.064	94.91	50	0.028	0.0139	0.0075

17.2.7 Loaded Carbon Movement

In order to determine the required carbon movement to yield the daily gold production for both the ROM feed and gravity tails scenarios, the dissolutions obtained in the test work were used. Table 17.9 summarises the philosophy applied.

Table 17.9: Loaded Carbon Movement

Parameter	Unit	Gravity Tails	ROM
Feed Grade	g/t	0.81	1.25
Dissolution	%	93.34	94.91
Recovered Au	g/t	0.756	1.186
Solids	% w/w	35	35
Recovered Au	ppm	0.407	0.639
Upgrade Ratio		2,200	2,200
Carbon Incremental Loading	g/t	896	1,405

Parameter	Unit	Gravity Tails	ROM
Plant Throughput	t/h	380	380
Daily Gold Recovered	g	6,895	10,820
Loaded Carbon Movement	t/d	7.7	7.7
Selected Carbon Movement	t/d	8	8

A loaded carbon movement of 8 t/d will be adequate to cater for the gold production required, and thus the Zadra elution method is proposed.

17.3 PROCESS DESCRIPTION

17.3.1 Introduction

The Kobada gold processing plant is designed to process oxide ores from the three main deposits: the North, South and Central ores during the first years of production followed by sulphides when the oxides are depleted. The process plant design utilises a combination of CIL and gravity recovery technologies to recover gold. Gravity gold is recovered from the cyclone underflow fed to the centrifugal gravity concentrator. The gravity recovery tailings are transferred back to the mill feed for further gold liberation. Gold that is not gravity recoverable is recovered through the CIL process.

The process plant consists of the following sections:

- Primary Jaw Crushing,
- Secondary Crushing and Tertiary Crushing (to be added in future when treating sulphides)
- Mill Feed Storage
- Primary Milling
- Secondary Ball Milling (to be added in future when treating sulphides)
- Gravity Recovery and Concentrate Leach
- CIL Gold Recovery
- Cyanide Detoxification, Arsenic Precipitation and Tailings Disposal
- Tailings Storage and Return Water
- Acid Wash
- Elution
- Electrowinning
- Carbon Reactivation
- Gold Room
- Reagents
- Diesel Services
- Compressed Air
- Water Services

17.3.2 Crushing and Mill Feed Storage

The crushing circuit will be constructed in two phases: the first phase will be to treat oxide ore only, followed by a second phase later in the LOM to treat sulphides and/or a blend of sulphides and oxides. When treating oxide ore, only the primary crushing stage will be in use while three crushing stages will be required when treating the sulphide ore.

The ore is fed into the ROM bin using a front-end loader or by direct tipping from haul trucks. An apron feeder, located under the ROM bin, is used to withdraw ore from the bin at a controlled rate and discharges onto a vibrating grizzly feeder to scalp off fines ahead of the primary jaw crusher. The vibrating grizzly feeder oversize material gravitates to the primary crusher while the scalped fines drop onto the sacrificial conveyor. The feed rate to the primary crusher is controlled by varying the speed of the apron feeder using a locally mounted dial-type speed controller.

For the oxide ore, the crusher product joins the apron feeder fines on the sacrificial conveyor. The sacrificial conveyor feeds onto the transfer conveyor, which transfers crushed ore either to the mill feed bin or the emergency stockpile.

For the sulphide ore,

- The crusher product joins the apron feeder fines on the sacrificial conveyor. The sacrificial conveyor diverts the ore to the secondary crusher screen feed conveyor, which transfers the material onto the secondary screen. The secondary screen oversize reports to the secondary cone crusher feed bin. The material is withdrawn from the bin using a pan feeder, which is controlled to choke feed the secondary cone crusher. The secondary crusher product reports to the tertiary screen feed conveyor where it combines with the secondary screen undersize and tertiary crusher product.
- The tertiary screen feed conveyor transfers the material to the tertiary screen. The screen oversize discharges into the tertiary crushers' feed bin. The material is withdrawn from the bin via pan feeders in a controlled rate to ensure a choke feed condition in the tertiary cone crushers. The tertiary cone crusher product reports onto the tertiary screen feed conveyor.
- The tertiary screen undersize discharges onto the tertiary screen undersize conveyor, from where it is transferred to the primary crusher transfer conveyor, which transfers crushed ore either to the mill feed bin or the emergency stockpile

A weightometer on the bin feed conveyor measures the material flow rate through the crushing circuit. The weightometer is used to control the rate of crushing and for metallurgical accounting purposes.

Normal feed to the milling circuit is by direct feed from the primary crushing circuit through the mill feed bin (surge bin). The ore is withdrawn from the mill feed bin using the mill feed apron feeder. The mill is fed by the mill feed conveyor. The feed to the mill is measured by the mill feed weightometer on the mill feed conveyor.

The feed to the mill comes from the emergency stockpile if the crushing circuit is offline for maintenance or on mechanical breakdown. The ore from the stockpile is reclaimed using a front-end loader for feed into the mill through the emergency ore bin. The emergency apron feeder is used to withdraw the ore from the emergency bin onto the mill feed conveyor.

Dust control is very important in the crushing and mill feed storage sections. Dust control is by way of both containment and suppression. Dust control conveyor skirting, dust enclosures, and dust suppression systems are included in the design as a means of containing the dust produced from the crushing section. The dust suppression system uses fine water sprays at the main dust-generating points in the crushing, stockpile, and mil feed sections.

Electric hoists are included in the crushing section to facilitate maintenance.

17.3.3 Milling

The milling circuit will be implemented in two phases: the first phase will consist of a single-stage ball mill to treat oxides while the second phase will consist of an additional secondary mill, which will be installed in the later part of the LOM to treat sulphides.

Crushed ore is conveyed from the mill feed bin to the ball mill. The ball mill has a variable speed drive, which allows the power input to the mill to be varied when requiring different milling energy inputs.

Tailings return water is the primary source of process water used for mill feed dilution and cyclone feed dilution. Fresh raw water is only used as a top-up to the process water if there is insufficient tailings return water for the process requirements. The mill feed dilution water is ratio-controlled to the mill feed rate to maintain the required mill discharge density. Mill feed dilution water is measured by a magnetic flowmeter on the process water line feeding the water into the mill feed chute.

The ball mill product discharges through the trommel screen, which removes the oversize scats. The scats produced from the ball mill are discharged from the trommel screen into the scats bund. A front-end loader periodically picks up and deposits the scats onto the emergency stockpile. The scats are reclaimed and fed back into the milling circuit using a front-end loader.

The undersize from the trommel screen gravitates to the mill discharge sump. The mill discharge slurry is diluted to the required density in the mill discharge sump. The dilute mill discharge slurry is pumped to the cyclone cluster by either of the cyclone feed pumps. The pumps are equipped with variable speed drives.

The cyclone underflow gravitates into the gravity splitter box. A portion of the cyclone underflow is directed to feed the gravity gold recovery circuit. The balance of the cyclone underflow reports back to the primary or secondary ball mill when treating oxides and sulphides, respectively.

The cyclone overflow gravitates onto the vibrating trash screen. The trash screen removes the trash oversize material (such as misplaced oversize particles, vegetal debris, plastic fragments, blast fuses, and wires) from the cyclone overflow stream before it gravitates into the CIL feed box. The trash oversize material discharges into a trash basket. Process water drains through the trash basket and gravitates to the spillage bund. The undersize from the trash screen gravitates into the CIL feed box, which feeds the first CIL tank.

The milling area is bunded to contain spillage and is equipped with two spillage pumps: one at the mill feed end, and one at the mill discharge end.

Grinding media (steel balls) are added to the ball mill by the ball loading kibble. The hopper is lifted by the crane onto the grinding media feed chute. The mill balls are discharged into the mill via the mill feed hopper by lowering the ball loading hopper to its rest position located directly above the mill feed chute.

17.3.4 Gravity Recovery and Concentrate Leach

A portion of the cyclone underflow is bled off to feed the vibrating gravity scalping screen. The vibrating gravity scalping screen is fitted with 2 mm aperture panels to scalp off the oversized material from the gravity concentrator feed. Dilution process water is added to the scalping screen feed box to achieve the optimum slurry solids concentration (~ 50 % solids by mass) required for the feed to the centrifugal concentrator. The scalping screen oversize gravitates via the oversize chute to the ball mill feed chute. The scalping screen undersize is fed to the gravity concentrator, which recovers the free gold from the gravity feed stream. Tailings from the concentrator gravitate back to the mill sump.

The centrifugal gravity concentrator operates in a batch mode on a set operating cycle. The slurry is fed to the concentrator for a pre-set time (typically 1 h to 2 h), or as determined by on-site optimisation. The duration of the concentrating cycle can be adjusted to suit the ore type being treated by using the machine countdown timer.

Ore particles are subjected to a centrifugal force of between 60 and 120 gravities in the centrifugal concentrator. Water is injected into the rotating concentrating cone through a series of fluidisation holes. The feed slurry is introduced into the rotating concentrating cone through the central vertical feed tube. Once the slurry reaches the bottom of the cone, it is forced outward and up the cone wall, filling each ring to capacity to create a fluidised concentrating bed. Once optimum fluidisation has been achieved, higher specific gravity particles are retained in the concentrating cone and lower specific gravity particles are discharged into the concentrator tailings. The quantity of high specific gravity particles in the concentrating bed increases progressively during the concentration cycle.

Water is pumped into the concentrator to keep the concentrating bed fluidised for efficient classification of the heavier gold-bearing particles from the lighter gangue material. Clean raw water is used as fluidising water, as suspended solids present in the process water may result in blinding the internal concentrating cone.

At the end of the cycle, the concentrator feed valve is closed, and the screen underflow bypasses the concentrator and goes back to the mill sump while the unit undergoes a flush cycle to discharge the concentrate accumulated in the bowl into the concentrate batch tank. Flushing the concentrator takes a period of 1 min to 2 min, after which the unit begins the next concentrating cycle.

The recovered gravity concentrate is treated through the intensive cyanidation process. Typically, enough concentrate to run the intensive cyanidation process is recovered on a daily basis. Once a full concentrate batch is collected, an automated intensive leach cycle is initiated. Intensive cyanidation is achieved using a cyanide-caustic solution. Hydrogen peroxide solution is added to the leach reactor to provide the oxygen required for the gold dissolution process. Lead nitrate is also added to aid the dissolution of gold. A batch of concentrate is treated through the intensive cyanidation process for a period of between 14 h and 24 h. The concentrate leach process produces a batch of pregnant solution. The pregnant solution is pumped to the gravity electrowinning tank located in the electrowinning area. The leached and washed solids are pumped back to the mill discharge sump.

Spillage in the gravity area is contained in a bunded area, and a spillage pump pumps it back to the intensive cyanidation system or mill discharge sump.

Owing to the use of sodium cyanide solution and caustic, a safety shower is provided in this area. The safety shower is activated by a foot pedal and equipped with an eye bath.

17.3.5 CIL Gold Recovery

The trash screen underflow slurry gravitates through the crosscut slurry sampler (to the CIL feed box). The sampler cuts samples from the CIL feed slurry stream at set time intervals to collect a shift composite sample for metal accounting purposes. The slurry gravitates from the CIL feed box to either CIL Tank 1 or CIL Tank 2, if CIL Tank 1 is being bypassed. The leach slurry is moved downstream by the pumping action of the internal impeller mechanisms of the interstage screens through the tank train to the final CIL tank. The CIL slurry tails gravitate from the final CIL tank to the detoxification and tailings pumping circuit.

The CIL circuit consists of six leach/adsorption tanks in series, with a provision to add an additional tank in the future. The dual impeller mixers maintain the slurry and carbon particles in suspension.

The CIL circuit is designed with the ability to bypass any tank if required (e.g. for maintenance). There are two launder outlets from each tank, and each outlet is equipped with a launder gate valve. The launder gate valve that allows the slurry to flow into the subsequent tank is normally open, while the second launder gate valve to bypass the subsequent tank is normally closed.

Each CIL tank is equipped with a mechanically swept wedge-wire cylindrical interstage screen, which prevents the migration of carbon from one CIL tank to another. The tank slurry levels are the same for all the CIL tanks. The transfer of slurry from tank to tank is achieved by the pumping action of the internal impeller mechanisms of the interstage screens.

The wedge-wire screens periodically become blocked with near-size carbon. Therefore, each screen is lifted from the tank onto the interstage screen wash frame for periodic cleaning. A spare interstage screen is provided to replace any screen that is removed for cleaning or repairs. A high-pressure, low-volume wash pump is used to clean the blocked interstage screens while they are in the wash frame.

Blower air is introduced into each tank for the gold leach process via the tank side-mounted spargers.

The cyanide solution is pumped to the CIL tanks through a ring main system. Free cyanide in CIL Tank 1 is measured by the free cyanide analyser, which aids in the automatic control of addition in the CIL circuit. A provision is made to manually dose cyanide into CIL Tank 2 and CIL Tank 3.

Lime is added to the CIL to maintain the slurry pH between 9.5 and 10.5. Lime is added to the process from the lime hopper through the mill feed conveyor.

Loaded carbon is transferred from CIL Tank 1 to the acid wash column via the loaded carbon screen. The design allows for the movement of a batch of loaded carbon daily.

CIL slurry-bearing loaded carbon is pumped to the loaded carbon screen using recessed impeller vertical spindle carbon transfer Pump 1 in CIL Tank 1. The intertank carbon transfer pump in CIL Tank 2 is sized to be able to transfer loaded carbon from CIL Tank 2 to the loaded carbon screen if CIL Tank 1 is offline.

The loaded carbon screen undersize gravitates to CIL Tank 1 or CIL Tank 2 if CIL Tank 1 is offline. The loaded carbon is washed with spray water and gravitates to the acid wash column. The carbon is moved upstream from the last CIL tank, countercurrent to the leach slurry flow, using the recessed impeller vertical spindle intertank carbon transfer pumps.

Spillage in the CIL area is contained in the CIL bunded area. The bunded area is equipped with two spillage pumps, which pump spillage from either side of the CIL tank train back into the CIL circuit.

The same tower crane used for maintenance at the milling section and for loading grinding media into the mill, is also used for the periodic removal of interstage screens for cleaning and for maintenance of the mixers.

Three safety showers are provided in the CIL area. There is a safety shower on either side of the CIL bunded area, and the third is located on the CIL platform.

17.3.6 Cyanide Detoxification, Arsenic Precipitation and Tailings Disposal

The plant design includes a cyanide detoxification process as well as arsenic removal by precipitation for the CIL tails slurry. Sodium metabisulphite (SMBS) and blower air are used as the detoxification reagents. Copper sulphate solution is added to provide the copper ions that act as a catalyst during the detoxification process. Ferrous sulphate is used to precipitate the arsenic in solution prior to tailings disposal.

Leached slurry from the last CIL tank gravitates to the tails screen feed box to feed the tails screen. The tails screen recovers fugitive carbon from the CIL tails slurry.

Carbon is lost in the CIL stream due to a number of reasons, which include the following:

- Fine or platy carbon passes through the interstage screen mesh.
- Carbon escapes through damaged, worn out or incorrectly installed interstage screens in the final CIL tank.

The carbon recovered on the screen gravitates to the carbon basket, from where it is inspected and assessed for re-introduction into the CIL circuit. The recovered carbon cannot be reused in the CIL circuit if most of the carbon is fine and platy. If the recovered carbon is suitable for re-introduction into the CIL circuit, fine carbon is screened out before the carbon is returned to the last CIL tank.

The undersize slurry from the tails screen gravitates to the detoxification tank, where SMBS and copper sulphate solution are added. Low-pressure compressed air (for oxygen supply) is fed into the detoxification tank via a sparging ring to ensure maximum dispersion throughout the slurry.

The cyanide species present in the tailings slurry are oxidised to the more stable cyanates by the addition of sulphur dioxide and oxygen according to the following reaction:

$$CN^{-}_{WAD} + SO_2 + O_2 + H_2O = OCN^{-} + H_2SO_4$$

Sulphur dioxide is provided from the SMBS solution. Oxygen is provided by the air blown through the slurry in the detoxification reactor. The cyanide detoxification reaction results in the formation of sulphuric acid; therefore, lime is added in the mill to ensure an optimal operating pH range of 8 to 10. The detoxification reaction requires the presence of copper ions acting as a catalyst at concentrations of approximately 50 mg/L. The copper ions are added to the process in the form of copper sulphate solution.

The detoxified slurry overflows into the arsenic precipitation tank, where the arsenic is precipitated using ferrous sulphate and low-pressure compressed air. Arsenic removal involves the formation of an insoluble ferric arsenate compound, which, when pumped to the tailings dam (together with slurry tailings), forms a sediment at the bottom of the tailings. A high Fe/As (molar ratio greater than 7) in the presence of heavy metals increases the stability of arsenic ferrhydrite. Reactions occur according to the following equations:

$$Fe(OH)_3 + H_3AsO_4 = FeAsO_4.2H_2O + H_2O$$

$$4FeSO_4 + O_2 + 6H_2O = 4FeOOH + 4H_2SO_4$$

$$AsO_3^{3-} + 1/2O_2 = AsO_4^{3-}$$

The reactions generate sulphuric acid, so in order to maintain the pH *4 and 7), caustic solution is added to the arsenic precipitation tank.

The slurry from the arsenic precipitation tank gravitates to the tails sump, from where the final tails are pumped to the tailings storage facility (TSF) using the tails pumps, in a duty and standby arrangement equipped with variable speed drives. Booster pumps will be installed as the TSF wall is raised.

A slurry sampler is installed on the arsenic tank overflow. The sampler collects samples at regular intervals. A shift composite sample is collected at the end of each shift for analysis.

Spillage in the detoxification and tailings area is contained within a bunded area, equipped with a spillage pump. Spillage is pumped back to the tails screen feed box.

Owing to the use of detoxification reagents in this area, a safety shower is located close to the detoxification reactor. The safety shower is activated by a foot pedal and equipped with an eye bath.

To prevent or minimise pollution to the environment, a pollution control pond has been provided to catch any spillages that overflow beyond the bund walls. The spillage is pumped back to the detoxification tanks so that the slurry is brought to an environmentally acceptable quality before being discarded into the TSF.

17.3.7 Tailings Storage and Return Water

Detoxified tailings slurry is pumped to the TSF. Return water from the tailings dam flows via a penstock to the return water dam. Return water is pumped from the return water dam back to the process water pond. Storm water is collected separately in the storm water dam and can be pumped back to the return water dam when needed during the dry season. An allowance has been made to pump from the storm water dam directly into the return water line to the plant.

A secondary cyanide detoxification facility is provided at the TSF to be used as and when required in instances where the plant detoxification processes have not achieved the required WAD cyanide levels. This facility is also used to destroy cyanide to acceptable levels in the excess return water, prior to discharge into the environment. Hydrogen peroxide, copper sulphate, and hydrochloric acid are used in the detoxification process.

17.3.8 Acid Wash Process

Loaded carbon is moved from the first CIL tank online into the acid wash column. The loaded carbon batch is washed with clean spray water on the loaded carbon screen and is discharged directly into the acid wash column. Once a batch has been accumulated, the carbon is washed with dilute hydrochloric acid to remove scale that builds up on the carbon in the CIL circuit.

The acid wash process is carried out in three steps: elutriation, acid wash, and rinse. At the end of the acid wash process, a batch of loaded carbon is ready for transfer into the elution column for gold stripping.

17.3.8.1 Elutriation

Elutriation is the process of partially fluidising the carbon bed to free and remove trapped light contaminants from the loaded carbon. The carbon batch is washed with raw water to remove any light trash such as slimes, plastic and organic fibrous material captured in the bed. Blowing small amounts of air through the acid wash column often assists in lifting trapped trash during the elutriation process. The trash is flushed to the tails screen feed box via a gravity pipeline.

17.3.8.2 Acid Wash

A 3 % by mass hydrochloric acid solution is made up in the acid wash tank using concentrated hydrochloric acid and raw water.

The dilute acid solution is pumped through the loaded carbon bed into the acid wash column using the acid wash pump. The acid solution overflow from the acid wash column gravitates back to the acid wash tank. The carbon bed expands with the acid wash solution upflow, and this enables optimum contact between the surfaces of the carbon particles and the slow-flowing acid solution.

The duration of the acid wash operation depends on the severity of the scaling. The duration of a normal acid wash cycle is 1 h to 2 h.

17.3.8.3 Rinse

After being washed with 3 % hydrochloric acid, the carbon is rinsed with raw water, and the rinse effluent is directed to the tails feed box.

17.3.8.4 Spent Dilute Acid Disposal

A batch of dilute acid wash solution is typically used to wash four batches of loaded carbon, after which the dilute acid batch normally becomes too contaminated to use.

The spent acid wash solution is neutralised with an excess of sodium hydroxide solution and is pumped to the tails screen feed box.

A spillage pump is provided in the acid wash section. Spillage is pumped to the tails screen feed box.

Two safety showers are provided in this area. Each safety shower is activated by a foot pedal and equipped with an eye bath.

Once the process of acid wash and rinsing has been completed (the pH of the rinse solution is equal to that of the wash water), the acid-washed carbon batch is drained by gravity into the transfer line and hydraulically transferred into the elution column.

17.3.9 Elution

The elution process utilises the pressurised Zadra elution method.

Gold is stripped from the loaded carbon by circulating a hot caustic cyanide solution, typically 3 % NaOH and 2 % NaCN, through the column at 125 °C under pressure. The eluate solution from the elution column is passed through electrowinning (EW) cells to recover the gold from the circulating eluate stream, and the electrowinning tails solution is returned to the eluant tank before being pumped back through the column.

The elution process involves two main stages:

- Stage 1: Heating the eluant solution, elution column and contents to operating temperature
- Stage 2: Circulating the eluate through the electrowinning cells and back to the eluant tank

17.3.9.1 Stage 1

The eluant solution is pumped through the elution column via a system of recovery and primary heat exchangers and back to the eluant tank. The secondary heat exchanger recovers the heat from the solution exiting the elution column. The eluant solution, preheated by the recovery heat exchanger, is passed through the primary heat exchanger where it is heated to the required temperature (125 °C) before entering the elution column.

Hot oil is circulated through the primary heat exchanger from the elution heater to heat the eluant solution to the temperature required for the elution process.

The solution (eluate) exits the elution column via externally mounted (duty and standby) Elution Strainers 1 or 2. The strainers prevent the carbon from migrating out of the column to the secondary heat exchanger.

The eluant is recycled back to the elution tank during the heating stage. Once the column, its contents, and the eluate exiting the column reach the operating temperature, the eluate is directed to the electrowinning cells and Stage 2 of the elution process commences.

17.3.9.2 Stage 2

The eluant solution at the required elution temperature is pumped through the column. The eluate exiting the column is cooled and passed through the electrowinning cells for a period of up to 14 h. The electrolyte from the electrowinning cells is circulated back to the eluant tank. The column operates under pressures of typically 300 kPa to 350 kPa.

At the end of the elution cycle, the heaters are switched off. The eluant continues to be pumped through the column until the solution exiting the column is approximately 90 °C. The cooled and eluted carbon is hydraulically transferred to the carbon reactivation section. The carbon transport water required to transfer the eluted carbon from the column to the eluted carbon holding tank is pumped from the carbon transfer tank using the carbon transfer pump.

A batch of eluant is used to strip up to four batches of loaded carbon, after which the level of contamination in the eluant becomes unacceptable for the eluant to be used for the elution process. The spent eluant is drained from the eluant tank into the elution bunded area and pumped by the spillage pump to the first CIL tank via the CIL feed box. The cyanide available in the spent eluant is utilised in the CIL circuit, and the residual gold values in the eluant are recovered onto the carbon.

A spillage pump is provided in the elution section. Spillage is pumped to the CIL feed box.

Two safety showers are provided in the elution area. Each safety shower is activated by a foot pedal and equipped with an eye bath.

17.3.10 Electrowinning

The EW circuit consists of two circuits:

- CIL gold EW circuit: Three EW cells
- Gravity gold EW circuit: One EW cell

17.3.10.1 CIL Electrowinning

The pregnant electrolyte from elution is directed to the EW cell feed tank. This allows de-aeration of the electrolyte to the cell and distributes the solution to the cells. The pregnant electrolyte gravitates from the feed tank and is equally distributed to the three EW cells. Any excess electrolyte from the cell feed tank overflows and joins the return electrolyte from the cells.

Sludging-type stainless-steel mesh cathodes are utilised to electrowin gold from the pregnant electrolyte. An electric current is applied across the cell electrodes and gold is deposited as fine sludge, loosely adhering to the pad of stainless-steel knit mesh contained in the cathode basket. The EW cycle takes place over a period of up to 14 h. Samples of the EW tails (barren electrolyte) are taken at regular intervals during the EW process. These are analysed for gold, caustic, and cyanide concentrations in solution. EW is complete once the gold tenor in the barren electrolyte reaches the required level of 5 ppm. When an EW cycle is complete, the barren electrolyte is sent, on demand, to the eluant tank.

17.3.10.2 Gravity Electrowinning

The pregnant solution from the intensive cyanidation circuit is pumped to the gravity pregnant liquor tank. The pregnant solution is pumped from the tank through the gravity EW cell using the gravity EW cell feed pump. Gold is deposited onto the stainless-steel mesh of the cell cathodes as a weakly bound fine sludge. The electrolyte tails from the gravity EW cell are recycled back to the gravity pregnant liquor tank. The electrolyte continues to circulate until the final gold tenor in the EW tailings solution reaches the required level of 5 ppm. When an EW cycle is complete, the barren electrolyte is pumped to the CIL feed box.

Cathodes are periodically lifted from the cells, using an EW hoist, and placed in the cathode wash tank. The cathode wash pump provides a high-pressure water spray to remove the sludge adhering to the cathode mesh. The sludge accumulated on the floors of the EW cells is washed into the gold sludge tank.

The sludge from the cathode wash tank is washed into the cell sludge tank. The sludge is then manually tapped off from the sludge tank into the sludge filter press using a bucket. The filter press dewaters the sludge and produces a gold sludge filter cake at approximately 60 % solids by mass. The filter cake is placed in trays and taken to the drying oven to remove surface moisture from the filtered gold sludge.

A fume extraction system on the EW cells extracts potentially poisonous and explosive gases that evolve during the EW process. A fresh air fan is installed to force air into the gold room to improve ventilation inside the building.

17.3.11 Carbon Reactivation

Eluted carbon is transferred hydraulically from the elution column to the kiln feed hopper.

Excess water and carbon fines drain through strainers fitted at the bottom and at the overflow of the hopper. The excess water and carbon fines discharge into the carbon transfer water tank.

Carbon is fed from the eluted carbon holding tank to the reactivation kiln using a variable speed screw feeder. The speed of the screw feeder is set in the vendor’s local control panel. The screw feeder moves carbon from the feed hopper into the kiln at a constant rate, as set on the vendor control panel. The carbon passes through the different reactivation zones as it is moved along the diesel-fired rotary reactivation kiln drum.

The reactivated carbon exiting the kiln is immediately quenched with water in the quench pan to prevent any oxidation reaction with atmospheric oxygen as it exits the kiln. The quenched carbon is passed over the quench screen where water sprays are applied to help remove fines before it gravitates to the eductor tank. The carbon is hydraulically transferred with the aid of the carbon transfer pumps from the eductor tank to the last CIL tank by pressurising the carbon in the tank and extruding it into the transfer line. The fines from the quench screen gravitate to the carbon transfer water tank.

New batches of activated carbon are discharged into the carbon attritioning tank where the carbon is mixed, wetted, and attritioned using a mixer. From this tank, the carbon gravitates to the quench screen for removal of fine carbon.

17.3.12 Gold Room

Filtered gold sludge is loaded onto drying trays, and the trays are loaded into the cathode sludge drying oven. The dried sludge is allowed to cool down and is then mixed with smelting fluxes at the required ratios. The fluxes and dried gold sludge mixture is loaded into the smelting crucible, which is fitted into the smelting furnace. The supply of diesel to the smelting furnace is via a ring main.

The diesel-fired smelting furnace operates at temperatures of between 1,200 °C and 1,400 °C. The furnace is fitted with a temperature control system and has a hydraulic tilting system for use during gold pour. The smelting furnace is covered by a fume hood with a flue duct that is vented outside the gold room.

At the completion of a smelt, the furnace firing system is switched off, and the molten contents of the crucible are poured into bullion moulds mounted on a cascade trolley. The bullion collects in the first mould, with any excess collected in the second mould, while the slag overflows and collects in a slag collection crucible on the last cascade.

The heavy metallic phase sinks to the bottom of the moulds whilst the light slag phase floats on top of the metallic phase. When both phases cool down and solidify, the glassy slag phase is easily broken away from the metallic phase, and the gold bar remains.

The bullion bar is further cleaned by chipping off and wire-brushing the slag adhering to the surface of the bar. The cleaned bullion bar is sampled using the prill drill. Samples are drilled out from two opposing long faces of the bar. The bar is then labelled, weighed, and stored in a safe, prior to dispatch to the refinery.

The gold room is equipped with two scales:

- Bullion scale, which measures the weight of the bullion and bullion samples
- Flux scale, which measures the weight of the flux

Fresh air is introduced into the gold room by means of a ventilation system, which includes the smelt house fresh air fan and the gold room extraction fan.

Spillage from the EW cells gravitates to a central drainpipe and discharges into the bunded area below the EW cells. The collective spillage inside the gold room is pumped to the CIL feed box. Spillage generated from the gravity pregnant liquor tank area gravitates into a separate bunded area and is pumped back into the gravity pregnant electrolyte tanks.

A safety shower is provided in the gold room area. The safety shower is activated by a foot pedal and equipped with an eye bath.

17.3.13 Reagents

17.3.13.1 Cyanide and Caustic

17.3.13.1.1 Sodium Cyanide

Sodium cyanide is delivered to site in bulk bags packed into wooden crates. The wooden crates provide additional containment in the event of spillage during transportation. The cyanide crates are transported from the cyanide storage area to the cyanide make-up area using a forklift. Cyanide is made up in batches equivalent to a whole number of cyanide bags. The required number of cyanide bags is lifted one by one, using the reagent hoist, onto the bag breaker fitted onto the cyanide make-up tank.

The cyanide make-up tank is equipped with a cyanide mixer. The cyanide make-up tank is half-filled with raw water. Cyanide briquettes are then added to the half-filled tank. The cyanide solution is mixed using the cyanide mixer to ensure that the briquettes are completely dissolved during the make-up process. The tank is topped up to level with raw water to make a concentration of 25 % cyanide by weight. Cyanide mixing continues for 30 min to 60 min to ensure that all the cyanide briquettes are dissolved. The made-up cyanide solution is pumped to the cyanide dosing tank using the single-duty cyanide transfer pump.

Two dedicated variable speed progressive cavity pumps, one duty and one standby, are used to dose cyanide to the intensive cyanidation, CIL and elution circuits.

Any cyanide spillage occurring during the make-up process is immediately hosed down with hosing water and directed to the cyanide spillage sump. It is pumped back to the cyanide make-up tank or to CIL circuit, using the cyanide spillage pump. Spillage in the cyanide dosing area is contained in a dedicated bund and gravitates to the elution area spillage bund.

A safety shower is located close to the cyanide make-up and dosing tanks and is activated by a foot pedal and equipped with an eye bath.

17.3.13.1.2 Caustic

Caustic is delivered to site (in the form of caustic pearls) in 25 kg bags packed onto pallets. The pallets are transported using a forklift from the caustic storage area to the caustic make-up tank. A pallet of caustic bags is lifted onto the platform above the caustic tank using the reagent hoist. The caustic make-up tank is half-filled with raw water, after which the operator manually lifts one bag at a time onto the bag breaker, until enough caustic has been added to make up a solution of 20 % by weight caustic solution. The tank is topped up to level with raw water, and the solution is mixed using a caustic mixer, which ensures that the caustic pearls are completely dissolved during the make-up process.

The caustic dosing pump is only run for the time required to deliver the various batch quantities of the reagent to the various distribution points: acid wash, elution, electrowinning, cyanide detoxification, and intensive cyanidation.

A safety shower is located close to the caustic make-up tank and is activated by a foot pedal and equipped with an eye bath.

17.3.13.2 Lime

Lime is brought to site in bulk bags and is added to the process through the mill feed conveyor from the lime feed hopper. The lime feed hopper has a variable speed rotary valve, which adds lime onto the mill feed conveyor at a controlled rate.

Lime is added to the mill feed to maintain the slurry in the CIL circuit at pH 9.5 to 10. An online pH analyser measures the pH in CIL Tank 1. The pH measured is used to adjust the rate of lime addition to the mill feed.

17.3.13.3 Detoxification and Arsenic Precipitation Reagents

These reagent dosing tanks are located adjacent to the detoxification facility.

17.3.13.3.1 SMBS

SMBS is delivered to site in bulk bags. The bulk bags required for batch make-up are transported from the storage area to the SMBS mixing and dosing area using a forklift.

The SMBS make-up tank is half-filled with raw water. The detoxification reagent hoist is used to lift the bags onto the bag breaker, and SMBS powder is discharged into the SMBS make-up tank. The SMBS powder is dissolved batch-wise to a 25 % concentration by weight. The tank is topped up to level with raw water, and the SMBS make-up mixer ensures adequate mixing.

The SMBS solution is transferred to the SMBS dosing tank using the single-duty transfer pump. Variable speed SMBS dosing pumps, one duty and one standby, are used to pump the SMBS solution to the detoxification circuit at a controlled rate.

The SMBS make-up and dosing tanks are equipped with dedicated extraction fan systems to extract dust and vapours formed during make-up and dosing.

17.3.13.3.2 Copper Sulphate

Copper sulphate is delivered to site in bulk bags. The bags are transported using a forklift from the storage area to the copper sulphate mixing and dosing area for make-up. The copper sulphate crystals are dissolved in raw water to make up a solution batch of 15 % concentration by weight.

The copper sulphate bags delivered to the make-up area are lifted onto the platform on top of the copper sulphate make-up tank using the reagent hoist. The operator lifts the bags onto the bag breaker, which discharges copper sulphate crystals into the copper sulphate make-up tank that has been half-filled with raw water. Once the required number of bags has been added to the make-up tank, the tank is topped up to level with raw water. The copper sulphate tank is equipped with a mixer, which ensures that the crystals are dissolved completely during the make-up process.

The copper sulphate solution is pumped to the copper sulphate dosing tank using the single-duty transfer pump. Variable speed copper sulphate dosing pumps, one duty and one standby, are used to pump the copper sulphate solution to the detoxification circuit at a controlled rate.

A safety shower is provided in the detoxification reagent make-up area, close to the tanks. It is activated by a foot pedal and equipped with an eye bath.

Spillage is contained in a bunded area. The spillage pump is used to pump the spillage from the detoxification reagent make-up area to the tails sump, and the spillage from the detoxification reagent dosing area is washed down and gravitates to the detoxification and tailings bunded area.

17.3.13.3.3 Ferrous Sulphate

Ferrous sulphate is delivered to site in bulk bags. The bags are transported using a forklift from the storage area to the ferrous sulphate mixing and dosing area for make-up. The ferrous sulphate crystals are dissolved in raw water to make up a solution batch of 25 % concentration by weight.

The ferrous sulphate bags delivered to the make-up area are lifted onto the platform on top of the ferrous sulphate make-up tank using the reagent hoist. The operator lifts the bags onto the bag breaker, which discharges the ferrous sulphate crystals into the ferrous sulphate make-up tank that has been half-filled with raw water. Once the required number of bags has been added to the make-up tank, the tank is topped up to level with raw water. The ferrous sulphate tank is equipped with a mixer, which ensures that the crystals are dissolved completely during the make-up process.

The ferrous sulphate solution is pumped to the ferrous sulphate dosing tank using the single-duty transfer pump. Variable speed ferrous sulphate dosing pumps, one duty and one standby, are used to pump ferrous sulphate solution to the detoxification circuit at a controlled rate.

A safety shower is provided in the detoxification reagent make-up area, close to the tanks. It is activated by a foot pedal and equipped with an eye bath.

Spillage is contained in a bunded area. The spillage pump is used to pump the spillage from the detoxification reagent make-up area to the tails sump, and the spillage from the detoxification reagent dosing area is washed down and gravitates to the detoxification and tailings bunded area.

17.3.13.4 Other Reagents and Consumables

17.3.13.4.1 Hydrochloric Acid

Hydrochloric acid is delivered in 1,000 L intermediate bulk containers (IBCs) at 33 % hydrochloric acid concentration by weight. When required, the hydrochloric acid containers are transported using a forklift from the storage area to the acid wash bund, close to the concentrated acid wash tank. The total volume of acid is pumped from the IBC to the acid wash tank using the acid transfer drum pump.

17.3.13.4.2 Activated Carbon

Activated carbon is delivered to site in 500 kg bulk bags. A forklift is used to transport the bags from the storage area to the CIL tank area. The tower crane is used to lift the bags onto the CIL platform. The fresh carbon make-up can be added to the carbon attritioning tank, from where it is conveyed with the regenerated carbon to the last CIL tank, CIL Tank 6. It can also be added to CIL Tank 5 if CIL Tank 6 is offline.

17.3.14 Diesel Services

Diesel is transferred via a supply tanker from the fuel farm into the diesel storage tank. A dedicated diesel supply pump supplies diesel via a ring main system to the elution heaters, carbon reactivation kiln, and smelting furnace in the gold room.

17.3.15 Compressed Air

The air blowers, two duty and one standby, supply the low-pressure air requirements to the CIL and detoxification tanks.

The high-pressure compressors, one duty and one standby, supply the high-pressure plant air and instrument air requirements.

Compressed air is filtered and stored in the general plant air receiver, from where it is distributed to the intensive cyanidation area and other areas in the plant for general use.

Instrument air is passed through either of the duty or standby air filters and the instrument air dryer. Dried instrument air is filtered again through another duty or standby air filter set before it is stored in the instrument air receiver. The instrument air receiver distributes instrument air to all the air-operated instruments throughout the plant.

Blower air to the CIL and detoxification tanks is supplied by dedicated air blowers.

17.3.16 Water Services

17.3.16.1 Process Water Distribution

The process water pond supplies the required process water needs of the plant.

Tailings return water is preferentially pumped directly to the process water pond. An additional source of process water is raw water overflow from the raw water pond if the process water source is unavailable. Raw water can also be pumped directly from the raw water supply line to the process water pond if needed for top-up or during plant start-up and commissioning.

Process water is used in the milling, gravity dilution and detoxification sections and is supplied by dedicated operating and standby Process Water Pumps 1 and 2. It is also used as service water for flushing, hosing, and spraying applications. Process water is distributed to the plant by three separate streams with dedicated pumps for each stream. Spray water is supplied by the duty and standby high-pressure, low-volume pumps. Hosing water is supplied by a single dedicated pump. Process water is supplied by the duty and standby high-volume, low-pressure pumps. The use of different pumps for these streams allows more efficient sizing and utilisation of pumps.

Spray water is used on the mill trommel screen, trash screen, gravity scalping screen, carbon safety screen, loaded carbon screen, and reactivated carbon screen. The hosing water pump is sized so that it is capable of being used as a standby pump for the spray water pump.

Spillage in the process water distribution area is collected in the area sump and pumped back into the process water pond using the process water spillage pump.

17.3.16.2 Raw Water Supply and Distribution

The raw water required in the plant is sourced from the Niger River and from boreholes around the plant area. The raw water is abstracted from the river via one of two raw water supply pumps and pumped to the raw water supply dam for storage. The raw water is supplied to the plant via the raw water dam transfer pumps.

The raw water is pumped from the supply dam and stored in the plant raw water pond. An allowance has been made for the raw water from the supply source to be diverted to the process water pond for top-up and during plant commissioning. Raw water is used for gland service, gravity concentrator fluidising water, reagent make-up, washing, potable water supply, and fire water. Each system is supplied by dedicated pumps, depending on the duty. Gland service water is used for selected slurry pumps in the plant. The raw water pumps supply water to the gland water tank in addition to the crushing, gravity and intensive cyanidation, CIL, acid washing, elution, EW, carbon reactivation, detoxification and all the reagent make-up sections of the plant. The distribution of gland water to the slurry pumps in the milling and detoxification sections of the plant is performed by a dedicated set of operating and standby pumps. Water from the gland water tank is also treated and used as potable water. The gravity concentrator fluidisation process makes use of dedicated concentrate fluidising water pumps.

The raw water pond is also reserved for firefighting water. The fire water system is part of a vendor package, which includes a primary electric pump, diesel-driven standby pump, and a jockey pump to maintain the required pressure in the fire water system.

Spillage from the raw water distribution area gravitates to the process water spillage sump and is pumped back into the process water pond.

17.3.16.3 Potable Water Distribution

Raw water supplied to the water treatment plant is sourced from the raw water pond and is treated in the potable water treatment plant for potable water distribution. The potable water is stored in the potable water storage tank and delivered to the potable water hydrospheres using the potable water pumps.

The potable water hydrospheres are used to maintain the required pressure in the potable water distribution header. Potable water is distributed to the plant safety showers, site administration offices, the on-site laboratory, and the ablution block.

18 PROJECT INFRASTRUCTURE

18.1 PROJECT ON-SITE INFRASTRUCTURE

The Kobada Gold Project is a greenfield project, and as such minimal infrastructure has been established on the project site. The on-site infrastructure required will be related to the processing plant and the supporting facilities as follows:

- In-plant access roads
- Plant buildings
- Plant reagents and consumables stores
- Process plant site drainage
- Sewage disposal
- Security
- Water supply
- Communications
- Power supply
- Fuel supply and storage

Refer to the overall process plant general arrangement as shown in Figure 18.1.

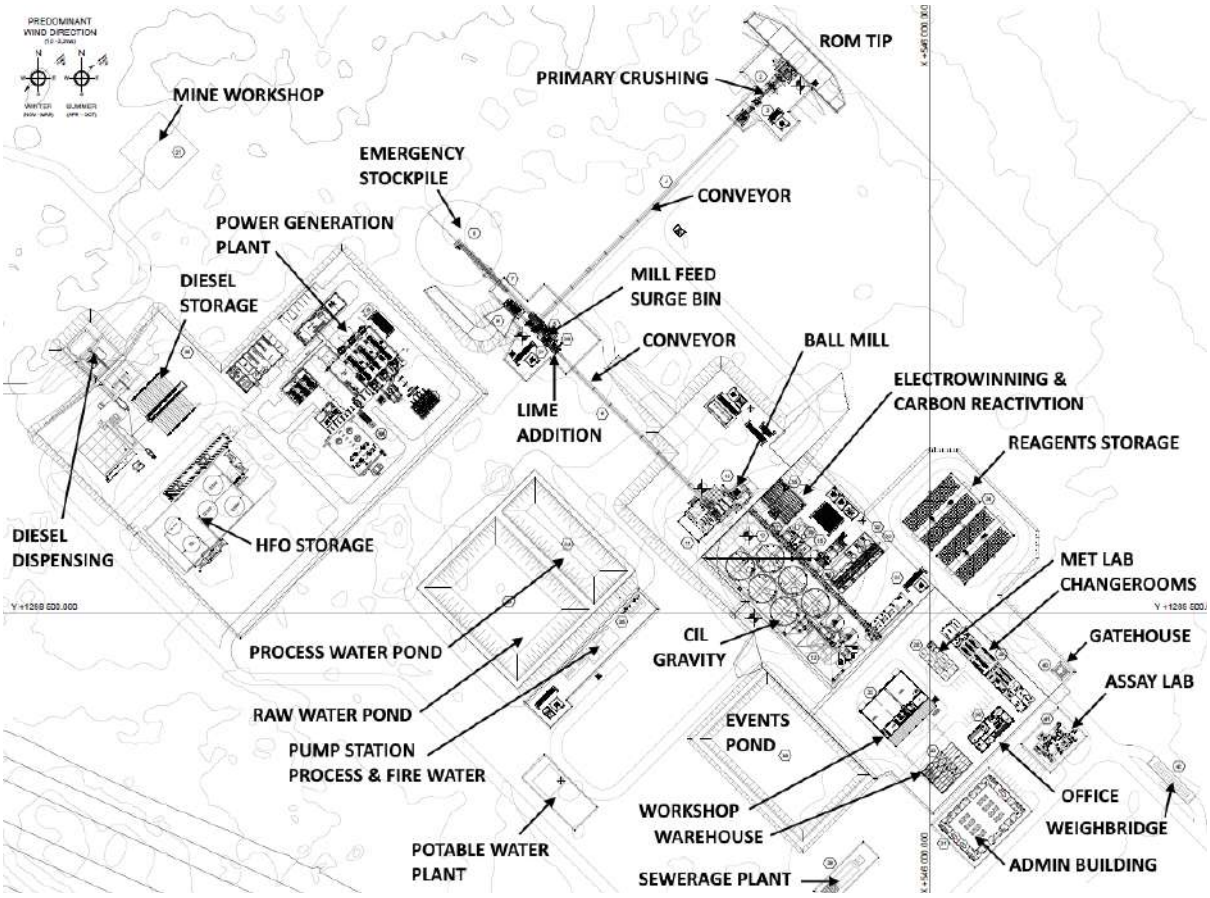


Figure 18.1: Process Plant General Arrangement

18.1.1 In-Plant Access Roads

In-plant roads will be constructed to provide unrestricted access to the following key areas within the process plant fenced perimeter:

- Reagents stores
- Workshops and warehouse
- Plant metallurgical laboratory
- Process and raw water dam pump stations
- Mill (access for crane hard stand area)
- CIL tank southwest perimeter leading to the process plant front-end materials handling areas

The road construction comprises ripping and recompacting to a depth of 200 mm; thereafter, backfilling with suitable borrow material and finally compacting it to 95 % MOD AASHTO.

The in-plant road network is shown in Figure 18.1.

18.1.2 Plant Buildings

The plant buildings will consist of the following:

- Gatehouse
- Security office and change house
- Plant control room
- Plant workshop
- Plant office building
- Plant metallurgical laboratory
- Plant warehouse

The position of the plant buildings is shown in Figure 18.1.

18.1.2.1 Gatehouse

A prefabricated, Chromadek panel gatehouse (4.6 m × 5.5 m) will be constructed at the main access gate to the plant and includes observation windows on three sides, complete with a fitted desk arrangement. The building also includes a single ablution facility. The prefabricated structure will be founded on a mesh reinforced concrete slab.

18.1.2.2 Security Office and Change House

A prefabricated, Chromadek panel security office and change house building (60.4 m × 10 m) will be constructed near the main access to the plant and will include a security office, change house, laundry, first-aid room, and dining/conference room. The prefabricated structure will be founded on a mesh reinforced concrete slab. A 2 m concrete apron slab, which will be covered by an awning, will form a walkway around the building.

Personnel access control will be monitored from the fully furnished security office, which contains a closed-circuit television (CCTV) system and is located adjacent to the plant reception. All workers and visitors will enter the plant through the reception area. Workers can then either proceed into the change house or into the secured area of the plant via access-controlled doors.

The change house will contain both male and female “clean” change areas consisting of lockers and benches for the process plant staff to store their clothes when they arrive for their shift. The staff will then move through a search zone into the “dirty” change area where they will change into their work clothes and proceed into the medium-security area on the way to the process plant. At the end of the shift, the staff will return to the ‘dirty’ change area, remove their dirty work clothes (which will be washed in the adjacent laundry), shower, and then proceed through the search area to the ‘clean’ change area, where they will retrieve their own clothes.

The change house has been sized for a maximum of 94 workers passing through the change house per shift with an average of 64 workers per shift for three shifts per day. A total of 250 workers will utilise the change house (including those on rest and relaxation (R&R)), with a total of 194 workers using the change house in a 24 h day (excluding those on R&R).

The male change house has been sized as follows:

- 125 double lockers in the “clean” change area (1 locker per person)
- 125 double lockers in the “dirty” change area (1 locker per person)
- 20 showers (1 shower per 5 people maximum per shift)
- 8 toilets (1 toilet per 12 people maximum)
- 8 wash-hand basins (1 wash-hand basin per 12 people)
- 6 urinals (1 urinal per 15 people maximum)

The female change area has been sized as follows:

- 6 double lockers in the “clean” change area
- 6 double lockers in the “dirty” change area
- 2 showers
- 1 toilet
- 2 wash-hand basins

A fully equipped laundry room (5.000 m × 5.035 m), where all the overalls from the shifts will be washed, will be provided adjacent to the change house. Periodic ad hoc inspections and screening of dirty overalls will be performed by the process plant security to identify any potential gold losses entrapped in staff workers clothing.

A first-aid room (5.000 m × 5.085 m) will be provided for the treatment of minor injuries sustained within the process plant, or as a holding room before evacuation from the process plant site.

A fully furnished dining room (10.120 m × 6.850 m), designed to accommodate 90 shift workers, will be provided. A section of the room can be partitioned off to form a conference/training room. The dining room will be utilised only for the serving of food. The food will be prepared off site in the main kitchen of the New Camp.

18.1.2.3 Plant Control Room

A dedicated plant control room will be located adjacent to the mill building. The control room will house the supervisory control and data acquisition (SCADA) system. The control room will consist of a converted 12 m container, elevated on a structural steel platform.

18.1.2.4 Plant Workshop

A workshop will be established to enable maintenance and repair of the plant equipment and is located on the southeast side of the process plant, adjacent to the CIL tanks.

The workshop will consist of a steel pre-engineered portal frame building enclosed by Chromadek roof and side-wall sheeting. The steel columns will be founded on reinforced concrete plinths and spread footings, and the workshop floor will consist of a mesh reinforced concrete slab.

The workshop floor area will be 450 m^2 (30 m × 15 m), which will be split into a mechanical repair shop of 300 m^2 and an electrical repair shop of 150 m^2, both of which will be serviced by a single 5 t overhead gantry crane.

A two-storey brickwork annex will be provided adjacent to the workshop floor area within the steel portal frame structure. An instrument workshop, tool store, male and female ablutions, and crib room will be provided on the ground floor, and four fully furnished offices for plant maintenance staff and a 12 m × 5 m store will be provided on the first floor.

A 30 m × 6 m storage area will be available adjacent to the two-storey brickwork annex under the workshop roof and will be enclosed by security fencing.

The appropriate plant workshop tools will be provided for the workshop.

18.1.2.5 Plant Office Building

A fully furnished, prefabricated, Chromadek panel construction plant office building will be provided for the process plant management personnel.

The plant office building will be located on the southeast side of the process plant and will comprise the following:

- 17 m^2 plant manager's office
- 22 m^2 mechanical and electrical office
- 12.5 m^2 metallurgists' office
- 2 × 12 m^2 plant supervisors' offices
- 20 m^2 meeting room
- 10 m^2 clerk office
- 11 m^2 storeroom
- 7.5 m^2 kitchenette
- Male and female ablutions

18.1.2.6 Plant Metallurgical Laboratory

A dedicated plant metallurgical laboratory will be located southeast of the plant. The plant metallurgical laboratory will house the necessary metallurgical test equipment, complete with shelving, and will be furnished to suit. The plant laboratory will consist of a converted 12 m container and will be founded on a mesh reinforced concrete slab.

The plant metallurgical laboratory staff shall utilise the adjacent change house ablutions as required.

18.1.2.7 Plant Warehouse

A 330 m^2 enclosed steel structure warehouse will be provided southeast of the process plant and adjacent to the plant workshop for the purpose of storing spares and equipment for the process plant. Suitable shelving and racking will be incorporated in this building to store and manage the stored items.

An unfenced laydown area is located adjacent to the plant warehouse.

The warehouse includes a brickwork stores dispatch room with a service counter and waiting area (5.300 m × 4.000 m), and a warehouse management office (4.300 m × 2.400 m).

18.1.3 Plant Reagents and Consumables Stores

This reagents' building location is shown in Figure 18.1.

The reagents stores are located northeast of the process plant, in close proximity to the reagents make-up area to facilitate unrestricted access to the make-up area. Reagents will be offloaded north of the stores, and a ring road has been provided around the storage area to facilitate the safe exit of the delivery trucks. The layout of the storage area has been designed to facilitate the first-in last-out principle, with roller shutter doors provided on both the north and south sides of the stores and a central 4 m wide alley that will allow forklift access.

The reagents will be contained within the following storage areas:

- Reagents Store A: Sodium cyanide/acids/basics/toxic reagents
- Reagents Store B: Quicklime

The reagents stores will comprise a steel-clad structure and concrete floor slab, complete with drainage and spillage handling facilities.

A concrete, paved area will be provided on both sides of the reagents store to facilitate loading into the store, and movement from the store to the reagents make-up area.

One month's stockholding of the reagents has been assumed when sizing the stores.

Reagents will be accommodated in the storage areas as set out below.

18.1.3.1 Reagents Store A

The store is 36 m × 21.9 m and 6 m in height and is divided into two sections:

- Section 1 – Sodium Cyanide storage
- Section 2 – Acids, Bases and other Toxic Reagents

18.1.3.1.1 Section 1 – Sodium Cyanide

The sodium cyanide storage area will accommodate 187 bulk boxes of cyanide, which will be stacked 2 boxes high. A concrete slab, complete with drainage and spillage handling facilities, will be provided for the safe offloading of the sodium cyanide boxes. The sodium cyanide section of Store A will be located within a fenced, gated and locked area.

18.1.3.1.2 Section 2 – Acids/Bases/Toxic Reagents

Section 2 of Store A will accommodate the following reagents:

- Copper sulphate (supplied in 25 kg bags): Provision for a total of 117 pallets stacked 2 high (16 bags per pallet)
- Sodium metabisulphite (supplied in 25 kg bags): Provision for a total of 507 pallets stacked 2 high (16 bags per pallet)
- Caustic soda (supplied in 1 t bulk bags): Provision for 20 bags stacked 2 high
- Sodium nitrate (supplied in 25 kg bags): Provision for 1 pallet (16 bags per pallet)
- Borax (flux) (supplied in 25 kg bags): Provision for 2 pallets (16 bags per pallet).
- Hydrochloric acid (supplied in 1 200 kg IBCs): Provision for 15 IBCs stacked 2 high
- Hydrogen peroxide solution (supplied in 1 200 kg IBCs): Provision for 2 IBCs stored in a 12 m container, which will be fenced and located a minimum of 10 m away from any other infrastructure.

18.1.3.2 Reagents Store B: Quicklime

The store is 36 m × 21.9 m and 6 m in height to accommodate 259 bulk bags of quicklime stacked 2 high.

18.1.4 Process Plant Site Drainage

The process plant area will be constructed with berms and side drainage as required to ensure that any water runoff not contained in the bunded areas and returned to the process is diverted to a drainage channel discharging to a silt trap prior to discharge into the storm water dam.

18.1.4.1 Berms

Storm water cut-off berms will be constructed to prevent storm water from entering lower lying areas from areas with a higher elevation. The berms will be constructed using the material from the bulk excavations when the bulk earthworks are carried out.

18.1.4.2 Side Drains

Surface drainage of the plant area will be achieved by using side drains. The surface of the plant shall be sloped to allow the water to flow freely away from the plant. The plant roads will be built to have a single cross fall of between 2 % and 4 % in the direction of the side drain.

18.1.5 Sewage Disposal

A 55 m^3/h containerised sewage treatment plant will be provided east of the main plant terrace for the treatment and disposal of the sewage generated.

Sewage reticulation piping and manholes will be provided to facilitate the flow of sewage under gravity to a collection manhole located adjacent to the sewage treatment plant. The sewage will be pumped via a submersible pump into the containerised treatment plant.

The technology selected is compact, simple and robust, and is based on a standard activated sludge system, where the Biochemical Oxygen Demand is broken down using air and bacteria that grow in this medium. This system provides optimised nitrification and an effluent quality to a standard that complies with the requirements of the Department of Water Affairs and Forestry for the release of treated effluent back into the environment, in accordance with the General Limit Values in terms of Section 39 of the National Water Act, 1998 (Act No. 36 of 1998).

18.1.6 Security

The plant site will be enclosed by a 2.1 m high Econo Mesh fence to keep out range animals and unauthorised people. Access to the plant site will be restricted to one access point at the main gate, which will be equipped with a gatehouse, manned 24 h/d. Other emergency access gates will be provided but will be kept locked at all times.

A 1.2 m high security fence will be erected around the perimeter of the process and raw water dams.

A 2.4 m high security fence, including 700 mm ripper flat wrap at the top of the fence and 730 mm ripper concertina coil at the base of the fence, will be erected around the gold room.

The gold will be transported to Bamako once a week, with suitable security arrangements in place.

Furthermore, the plant will be fitted with CCTV cameras installed at strategic locations. The cameras will be integrated with the plant's overall network, and dial-up into these cameras via the Internet will be enabled. Views from the cameras will be fed to a central security control room situated in the security office and change house building.

18.1.7 Water Supply

To ensure an uninterrupted supply of water to the plant, water will be supplied via on-site raw water and process water ponds. These ponds will be fed from two sources:

- The raw water dam (RWD), which is fed from the Niger River (435 m^3/h design flow rate) and the pit dewatering facilities to the RWD (variable during dry/wet season)

- The TSF water return system (340 m^3/h design flow rate)

A high-level water balance for the plant was completed, which incorporated the interaction of the various flows for the process plant and TSF. This water balance was used as the basis for sizing the water storage dam. Water stored in the RWD will be pumped to the process plant for make-up operations.

A floating barge pump system, which houses two 280 kW diesel-driven pumps, will be installed in the Niger River, 9.6 km away. This system abstracts water from the river to feed the RWD located closer to the plant. This RWD provides 20 000 m^3 of water storage dedicated to the process plant.

Two centrifugal pumps will be installed at the RWD to enable pumping of water to the raw water pond located in the plant. These pumps will be sized to cater for commissioning and the dry and wet seasons, where raw water demands vary significantly. The RWD perimeter and pump station will be fully fenced complete with a small guard hut to facilitate security for the area.

The pit dewatering requirements to cater for the ground and rainwater ingress have been met by means of borehole pumps (eight pumps at 12 m^3/h each) located around the pit. Furthermore, two portable (163 kW diesel motor driven) dewatering trailer systems have been allowed for, each rated for a dewatering capacity of 250 m^3/h.

A floating barge system will be installed at the TSF to house the two 110 kW water return pumps. This system pumps the return water directly to the process water pond located in the plant.

18.1.7.1 Potable Water Distribution

Raw water will be supplied from the raw water pond to the potable water storage tank situated in the potable water treatment plant. Potable water will be supplied to all the areas inside the plant via buried overland piping or piping running above ground in the plant area. Potable water will be supplied directly to safety showers.

The potable water plant will be a containerised unit capable of producing 15 m^3/h of potable water. The water will be chemically oxidised, and the pH will be adjusted. Then the water will flow through an arsenic removal filter and an activated carbon filter.

18.1.7.2 Fire Water Distribution

There will be an electric and a diesel-powered fire water pumping system. The electric-powered pump will be used in the event of a fire, and the diesel pump will be a backup in case the motor control centres (MCCs) are on fire. A jockey pump will be provided to maintain the pressure in the fire water header during normal plant runs. An alarm will be sounded at the plant site for low system pressure.

The fire water system will consist of a buried fire water loop and hydrant system at the plant site, ancillary buildings and at the process plant. Hose cabinets will be placed at the fire hydrant locations, and the system will be supplemented with portable fire extinguishers placed within the process plant facilities. The administration building, mine change house and canteen will have sprinkler systems, hose reels and portable fire extinguishers.

A complete self-contained fire alarm system will be installed in all buildings in order to comply with the local codes and insurance underwriter's regulations for fire protection.

18.1.8 Communications (IT Network)

The communications system for the plant will be specified by AGG. A provision for this system has been made under the Owner's Pre-Production Cost.

18.1.9 Power Supply

After considering the results established in the technical and commercial evaluation, AGG plans to have electricity supplied by an Independent Power Producer (IPP) to reduce the initial upfront capital.

Three options were evaluated based on reliability, utilisation, and redundancy in order to achieve the best cost of energy:

- Thermal power generation
- Hybrid system with thermal and solar photovoltaic (PV) power generation
- Hybrid system with thermal, solar PV and energy storage system power generation

The operational cost of thermal power plants is significantly high due to their fuel consumption, and consequently the cost of energy is high since it is related to the fuel cost. Commercial solar PV power has been proven to provide lower cost energy for longer life cycle projects. A hybrid system uses PV power generation to reduce the loading of the thermal generators, which results in a considerable saving on fuel consumption and lowers the environmental impact of the emissions produced by the plant. However, the PV penetration is limited due to the minimum loading requirement for the generators, as well as the operating reserve required for the PV generation capacity, which ensures network stability in the event of a sudden change in PV generation capacity or load requirements. The option that was found to achieve the best cost of energy was the hybrid system in combination with a suitably sized battery energy storage system to provide operating reserves and to enable the thermal plant to run at its best efficiency point to allow for higher solar PV penetration and lower fuel consumption.

This option is designed to deliver the lowest cost of energy and reliable electricity, but it also includes a strong renewable energy component that will significantly reduce the carbon footprint of the mine.

18.1.9.1 Power Demand

An outline of the electrical power demand is shown in Table 18.1, based on the mechanical equipment list, mining schedule and plant infrastructure.

Table 18.1: Electrical Power Demand

Year	Maximum Demand (kW)	Average Demand (kW)	Annual Energy (kWh)
Oxides – Primary Crushing			
1	9,656	5,054	44,273,040
2	9,818	5,139	45,016,827
3	9,822	5,141	45,034,536
4	9,830	5,145	45,074,382
5	9,848	5,155	45,154,074
6	9,832	5,146	45,083,237
7	9,952	5,209	45,632,222
Sulphides – Secondary and Tertiary Crushing			
8	14,476	7,577	66,374,141
9	17,654	9,241	80,947,294
10	17,604	9,214	80,718,041
11	10,610	5,553	48,647,216
12	10,559	5,527	48,416,997
13	10,662	5,581	48,886,291
14	16,383	8,575	75,116,972
15	18,117	9,762	85,511,449
16	7,670	4,015	35,170,503

The average steady-state continuous power demand for year 1 to 7 is estimated at 5 141 kW, with the ball mill being the only load of critical relevance to the maximum start-up energy demand. The ball mill drive will be driven by a squirrel cage motor, and the start-up current will be limited with a Variable speed drive (VSD). It is anticipated that the VSD will reduce the starting power demand of the ball mill to a maximum of 1.6 times the rated capacity of the motor.

The start-up sequence of the ball mill will last for approximately 20 s to 30 s, and it will increase the plant's maximum power demand to 9 952 kW. The power plant is suitably designed to deliver the required power for the start-up duration without interruptions to the other loads and will limit the voltage regulation to within 10 % of the rated system voltage.

After year 7 the mine schedule will start to process sulphides, thus the process plant will be upgraded with a 5 000 MW ball mill and a secondary and tertiary crushing circuit. The additional process equipment will increase the plant's maximum power demand to 18 117 kW and the power plant will be upgraded to deliver the required power.

18.1.9.2 Thermal Generation

The IPP evaluated the use of natural gas, Diesel and HFO. Natural gas supply is currently being expanded in Ghana with one energy vendor planning a liquified natural gas supply to Mali in future. This expansion is subject though to sufficient demand. Although the supporting infrastructure is not yet ready to support a binding commercial offer in Mali at this time, and thus excluded from this study, the natural gas option should be re-considered in future as the sustainable supply becomes more viable.

The IPP selected the generators based on their reliability and fuel efficiency, but also because they are widely used in the region, which will ensure fast and cost-effective maintenance for the system, without affecting the mining operations.

The medium-speed HFO generators are combined with high-speed diesel generators to ensure that the overall system is running at its optimal load capacity. The generator units will be designed, financed, owned and operated by the IPP for sixteen years. The capacity charge offered by the IPP allows the investment to be amortised fully and all the debt to be paid off after the sixteen-year contract term.

The generators are configured to operate in a continuous operating mode. This will ensure that a reliable and steady power supply is provided throughout the contract period, and that the equipment supplier's warranty and service requirements for the generators are not compromised. The HFO and diesel generator parameters are provided in Table 18.2 and Table 18.3, respectively.

Table 18.2: HFO Generator Sets

Item	Description
Model	6CM25 or similar (medium-speed engine)
Engine Manufacturer	Caterpillar
Prime Rated Power	2 400 kVA/ 1 920 kWe at 0.8 pf (power factor)
Rated Voltage	11 kV
Total Generation Capacity	7 436 kWe at 0.8 pf
Fuel Consumption at 75 % Loading	185 g/kWh

Table 18.3: Diesel Generator Sets

Item	Description
Model	3516C or similar (high-speed engine)
Engine Manufacturer	Caterpillar
Prime Rated Power	1 893 kVA/1 515 kWe at 0.8 pf (power factor)
Rated Voltage	11 kV
Total Generation Capacity	13 635 MW at 0.8 pf
Fuel Consumption at 75 % Loading	249 L/h

Mali has a very hot climate, which can put significant strain on equipment. The IPP designed their system to be able to effectively run at an average temperature of 45 °C.

18.1.9.3 Solar PV Plant

The IPP shall only select Tier 1 manufacturers for components such as inverters, transformers, the SCADA system and metering equipment.

The PV modules shall use mono PERC (passivated emitter and rear cell) crystalline bi-facial technology supplied by Tier 1 manufacturers with a 25-year linear degradation guarantee. The modules shall comply with IEC 61215 and IEC 61730. The parameters of the PV plant are shown in Table 18.4.

Table 18.4: Solar Photovoltaic (PV) Plant

Item	Description
AC power rating	9 MW
DC power rating	11 MWp
Production, Year 1 - P50	23 937 MWh
Specific production	2 217 kWh/kWp/yr
Guaranteed Degradation	0.47 %
PV Module Size	465 Wp
Number of PV modules	23 220

The PV array will consist of single axis trackers to maximise the potential solar resource and provide a flatter generation profile as shown in Figure 18.2.

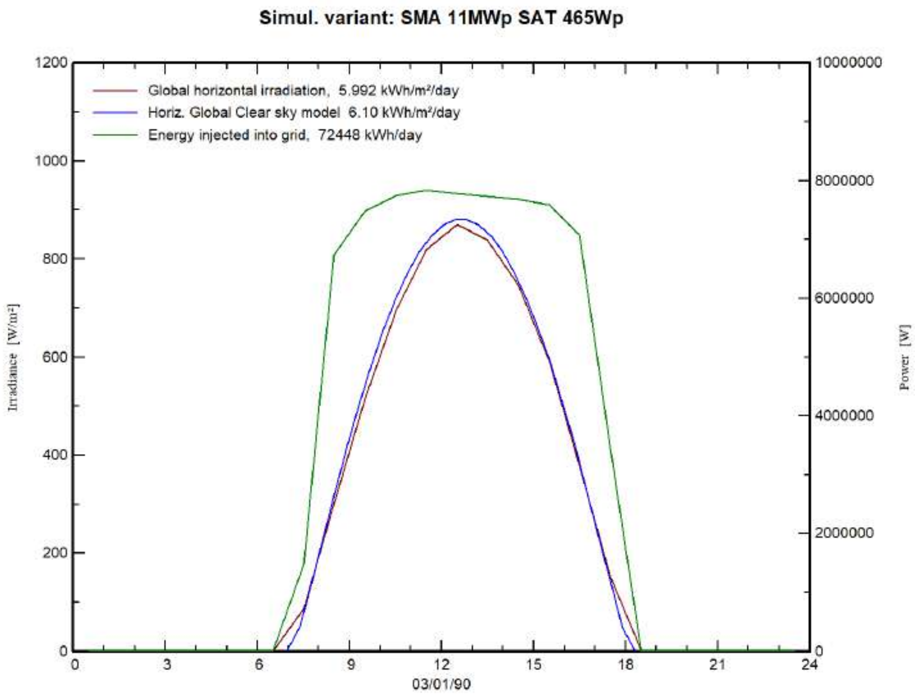


Figure 18.2: Typical Daily Solar PV Generation Profile – Single Axis Tracker

18.1.9.4 Battery Energy Storage System (BESS)

The BESS consists of batteries and a power conversion system that charges and discharges the batteries.

Due to the cobalt content of lithium nickel manganese cobalt oxide (NMC) batteries, they are one of the safest battery technologies available. They also have among the highest energy densities, charge/discharge ratings (C rate), and life cycle of all batteries. For these reasons, this battery type was selected for the project.

The selected power conversion system is of the grid-forming (black-start) type, which will add additional redundancy to the thermal plant as the systems do not require a reference grid to function, and they can generate a reference grid for solar PV inverters.

The final solution will be containerised with all the required cooling and protection systems to allow for maximum reliability and ease of maintenance.

18.1.9.5 Benefits of a Hybrid Power Plant

The power system is aimed at both reducing the cost of energy and the impact on the environment, while also increasing the reliability of the power system. This enables African Gold Group to achieve its sustainable development goals.

The IPP will build, install, and operate a highly efficient hybrid power plant that will combine renewable energy, energy storage and thermal generators. This efficient solution will enable significant emissions reduction compared to a conventional thermal plant:

Heavy Fuel Oil consumption	⌄ 43%
Carbon dioxide	⌄ 39%
Carbon monoxide	⌄ 34%
Unburned hydrocarbons	⌄ 22%
Sulphur dioxide	⌄ 39%
Nitrogen oxides	⌄ 26%

In addition to the environmental benefits, the significant fuel offset from the renewable energy component of the hybrid power system also de-risks the mine against fuel cost fluctuations. A 59 % increase in fuel costs has only a 22 % increase in the resulting energy costs.

Figure 18.3 is a graphical representation of the sensitivity to fuel cost for a Diesel-only plant compared to the hybrid solution. This is for illustrative purpose only, as the hybrid design will change according to the expected fuel cost.

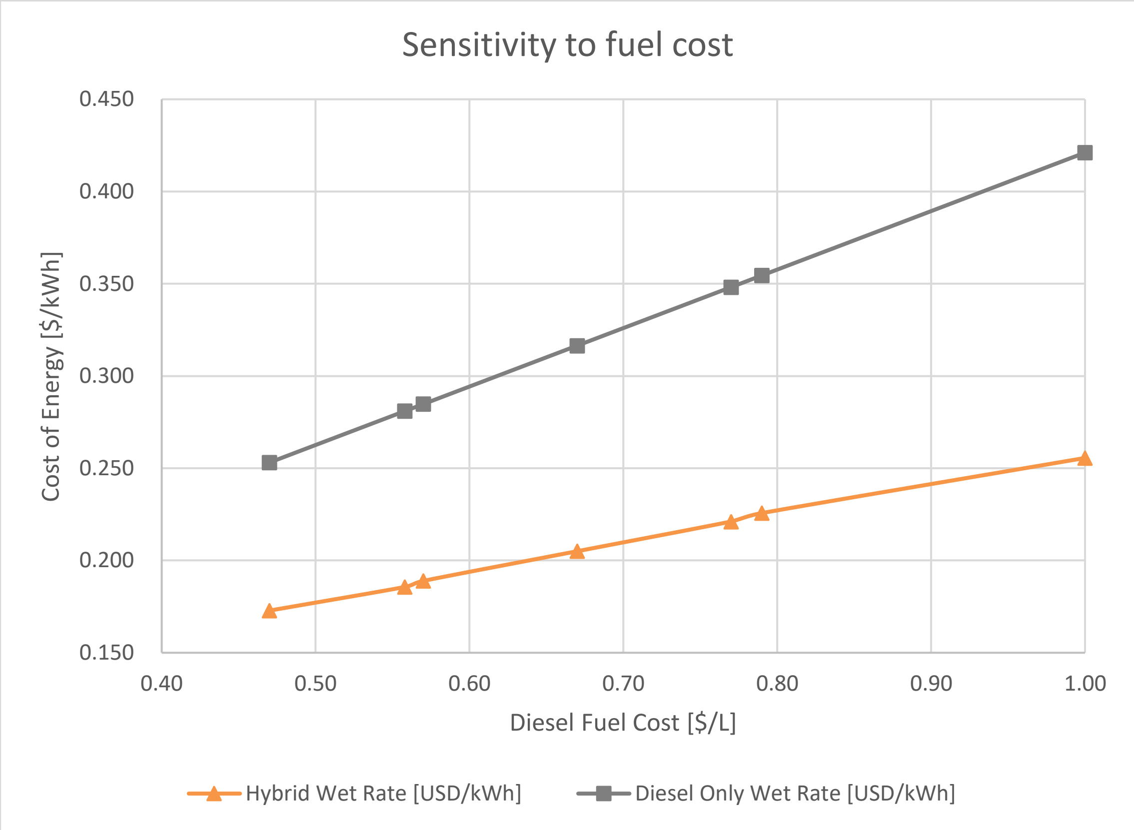


Figure 18.3: Sensitivity of cost of energy vs. diesel cost fluctuation

18.1.9.6 Hydroelectric Plant

18.1.9.6.1 Hydroelectrical Energy

The Sélingué Dam is located approximately 40 km from Kobada. With a power output of 44 MW, the dam has an energy output of 200 million kilowatt-hours per year. The dam provides Bamako, Kati, Koulikoro, Ségou, Fana, Dioïla, Yanfolila and Kalana with electricity. It was brought into service in 1980, and renovated between 1996 and 2001. The utilisation of the hydro resource is currently unknown. There may be an opportunity to construct an overhead line to this energy source to offset the thermal plant requirement, thus eradicating Scope 2 emissions completely by utilising only renewable energy for all the electrical requirements.

18.1.9.6.2 Natural Gas

Preliminary indication is that construction of liquified natural gas (LNG) facilities in Ghana is well under way with expected completion towards the end of 2022 with LNG export capabilities to Mali to follow soon after. This will further reduce reliance on diesel and HFO with significant reduction in CO_2e emissions.

18.1.9.7 Electrical On-Site Infrastructure

The electrical on site infrastructure caters for the following process plant infrastructure:

- Office building
- Metallurgical laboratory
- First-aid room
- Warehouses
- Workshops

Power reticulation will be done at the following voltages:

- Medium voltage: 11,000 V
- Low voltage: 400 V
- Control voltage: 110 V AC

18.1.9.8 Medium-Voltage Switchgear

A common 11 kV containerised medium-voltage substation has been allocated for the on-site and off-site mine infrastructure.

The switchgear will be rated using the following details:

- Medium voltage: 11 kV
- Frequency: 50 Hz
- Main bus bar rating: 1 250 A
- Basic insulation level: 95 kV
- Short circuit rating: 25 kA – 3 s

The protection scheme requirements given in Table 18.5 to Table 18.8 are proposed general requirements. Once the implementation phase of the project is reached, then the protection schemes will be finalised, with the emphasis on stability and reliability.

Table 18.5: Power Plant Incomers

ANSI Code	Description	ANSI Code	Description
50/51	Phase Overcurrent	81L	Under frequency
50G/51G	Sensitive Earth Fault	81H	Over frequency
50BF	Breaker Failure	87	Differential Protection
27	Undervoltage Relay	32P	Directional Active Overpower
46	Negative Sequence Unbalance	26/63	Thermostat
59	Overvoltage Relay		

Table 18.6: Feeders and Overhead Line Feeders

ANSI Code	Description	ANSI Code	Description
50/51	Phase Overcurrent	50BF	Breaker Failure
50G/51G	Sensitive Earth Fault	50N/51N	Earth Fault

Table 18.7: Transformer Feeders

ANSI Code	Description	ANSI Code	Description
50/51	Phase Overcurrent	50BF	Breaker Failure
50G/51G	Sensitive Earth Fault	26/63	Thermostat
50N/51N	Earth Fault		

Table 18.8: Ring Main Unit Feeders

ANSI Code	Description	ANSI Code	Description
50/51	Phase Overcurrent	26/63	Thermostat
50G/51G	Sensitive Earth Fault	87	Differential Protection
50N/51N	Earth Fault		

18.1.9.9 Transformers

Distribution transformers will be manufactured in accordance with IEC 60076 and other relevant international standards and will be as follows:

- Dry-type, housed within a building
- Insulation medium air ON/AN (oil natural/air natural)
- Three-phase
- Copper windings
- Vector Group Dyn 11
- Offload tap changer ± 2 × 2.5 %

18.1.9.10 Low-Voltage Distribution

The maximum transformer rating for low-voltage supplies will be 2 500 kVA. Each transformer will feed a 400 V MCC that supplies power to a dedicated section of the plant. Feeds to the MCCs will be single feeds only.

The MCCs will feed the lighting distribution boards that will supply lighting and small power distribution (normal) at 400 V/230 V (single-phase or three-phase).

18.1.9.11 Motor Control Centres

Five containerised MCCs have been allocated for the process plant. The MCCs will be of the compartmentalised, non-withdrawable type with moulded case circuit breakers, magnetic contactors and earth bus, and they shall comply with IEC 61439-2 and carry the South African Bureau of Standards' (SABS) stamp of approval.

The MCCs will have the following features:

- Enclosures will be of the general-purpose type for indoor service (IP 54 as defined in IEC 60529) or weatherproof type for outdoor service (IP 65), as required.
- Main buses will have a minimum capacity of 400 A.
- Each MCC will have a single 230/110 V AC control transformer rated for each MCC's control circuit requirements.

- The control circuit will have uninterrupted power supply backup rated at 10 kVA, in the event of a power failure.
- Outgoing power and control wiring will be brought out to terminals in the wireways.
- Siemens equipment will be used with the SIMOCODE Pro V relay for efficient and reliable motor protection.

18.1.9.12 Electrical Motor Control Stations

The electric motors are as per the requirements on the equipment list. Motors shall be of Efficiency Class IE3 (premium efficiency) in accordance with IEC 60034-30-1.

The control of these motors can be summarised as follows:

- Start-Stop/Emergency-Stop push-button stations will be located at each motor.
- Stations located in wet process areas or outdoors will be of watertight construction.

18.1.9.13 Earthing and Lightning Protection

Provision has been made for earthing of all electrical equipment and buildings where applicable.

The earthing philosophy for the supply of plant equipment shall be the TN earthing system, where one of the points in the generator or transformer is connected to earth, usually the star point in a three-phase system. The enclosure of the electrical device is connected to earth via this earth connection at the transformer.

Provision has been made for earth resistivity testing prior to the installation. Earth mats shall be installed at all medium-voltage substations, ring main units and transformers.

The minimum earth wire size provided is 70 mm^2 between the MCCs, substations and the power plant.

High mast lighting shall form part of the plant's lightning protection system by serving as lightning surge arrestors.

18.1.9.14 Electrical Cables

The following shall apply to electrical cables:

- All cables specified shall comply with the relevant part of SANS 1507.
- All outdoor cables shall either be buried in the ground or placed on cable racking.
- Cables shall cross underneath roads in dedicated sleeve polyvinyl chloride pipes.
- Grouped cables shall be de-rated in accordance with SANS 10142-1 for 600 V/1 000 V cables.

18.1.9.15 Cable Racking

Cable racking shall be used where cables are running on structures or indoors, or where cable support is required.

18.1.9.15.1 General Process Plant Area

Cable racking shall have the following specifications:

- Hot dipped galvanised steel
- Heavy duty application
- Welded construction
- Standard straight length of 6 m
- Side rail height of 75 mm

18.1.9.15.2 Cyanide, Acid and Sodium Metabisulphite Exposed Area

Cable racking shall have the following specifications:

- Materials of construction using 316 stainless steel
- Heavy duty application
- Welded construction
- Standard straight length of 6 m
- Side rail height of 75 mm

18.1.9.16 Lighting

Provision has been made for high-pressure sodium vapour (HPS) lighting, which will be structure-mounted to ensure safe working conditions. Lighting will also be installed to ensure that visual security monitoring can be conducted at all times in and around the process plant and associated infrastructure to maintain a safe work environment. The final design and layout will be confirmed during the implementation phase.

18.1.9.17 Fire Detection System

Provision has been made for fire detection systems for all medium-voltage switches, MCCs, servers and control rooms. These rooms will also be equipped with handheld firefighting equipment. The fire detection systems will be integrated with the plant's central control system to alert the plant operators of any fire incidents.

18.1.10 Process Plant Diesel Storage

The diesel for plant usage will be supplied by means of a diesel bowser, to a self-bunded diesel storage tank. The self-bunded storage tank can store 12 000 L of diesel.

18.1.11 Tailings Storage Facility

18.1.11.1 Introduction

In 2020, AGG commissioned a DFS of the Kobada Gold Project, under the study lead of SENET. Epoch Resources (Pty) Ltd (Epoch) was in turn appointed by SENET to undertake the DFS design of the tailings storage facility (TSF) associated with Kobada. The facility has since increased in size, and the TSF design has been updated to accommodate the revised required storage capacity, as documented in this report.

18.1.11.2 Project Location and Topography

The Kobada site (see Figure 18.4) is located in southern Mali, in proximity to the borders of Guinea and Mali, approximately 120 km southwest of Bamako and near the Niger River.

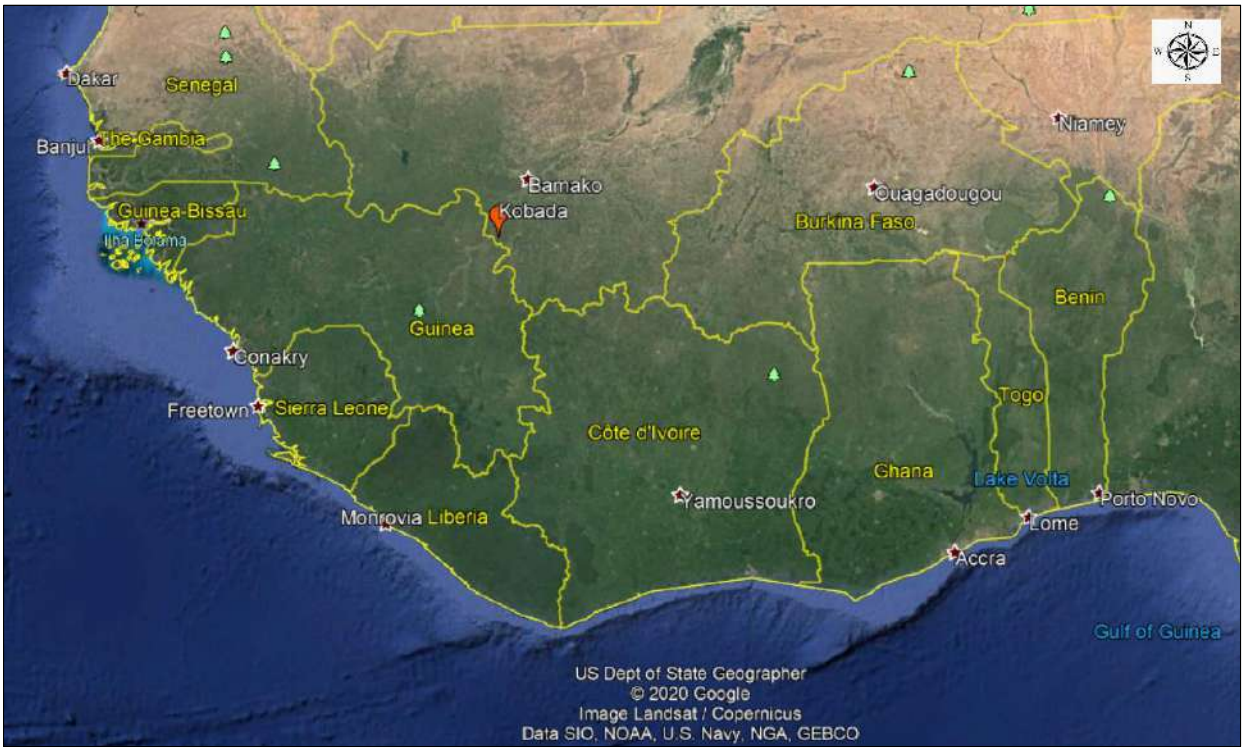
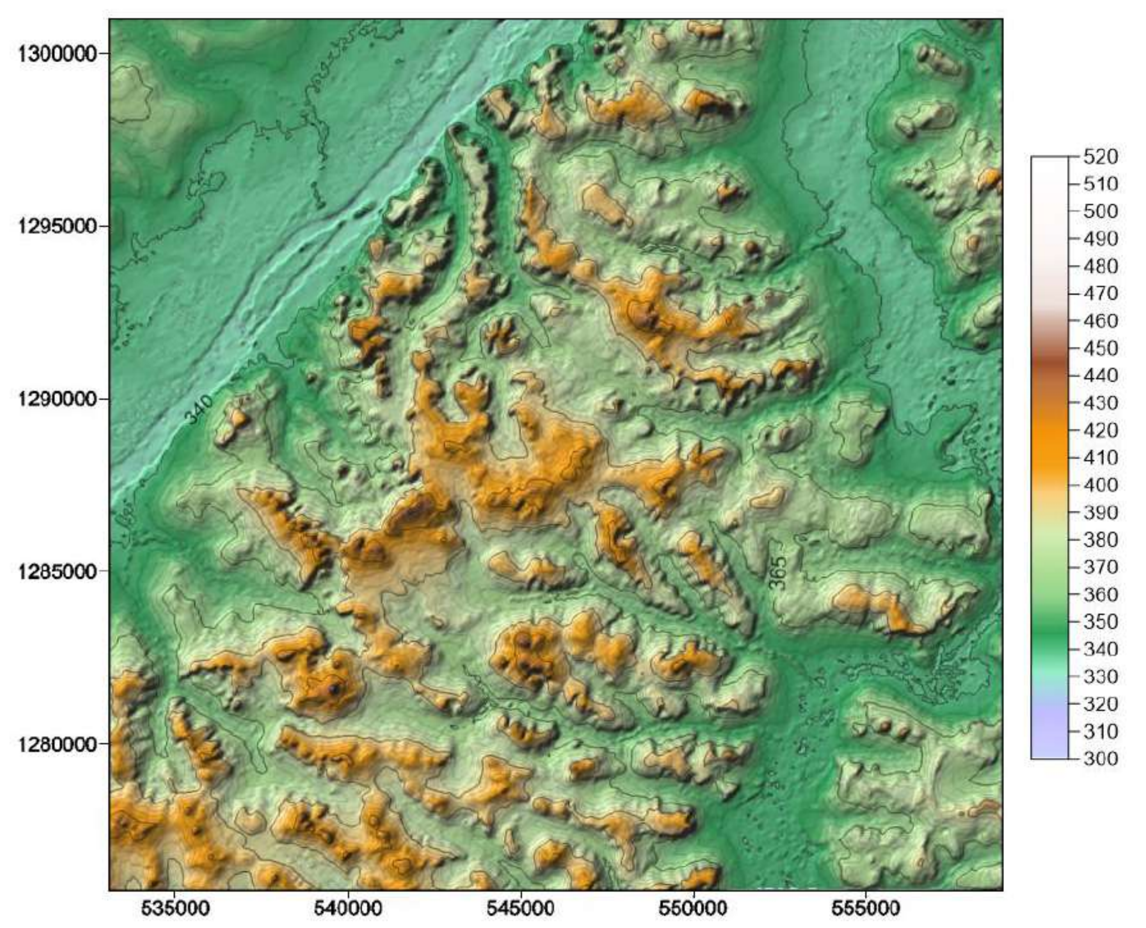


Figure 18.4: Location of the Kobada Gold Project

The topography of the site and its surrounding areas, as shown in Figure 18.5, is characterised by flat undulating valleys of the Sahelian savannah. The elevation of the site varies between 330 mamsl and 460 mamsl, with an average of 350 mamsl. The primary drainage occurs in an easterly direction towards the Fié River that feeds into the Niger River.



NOTE: Elevations given in metres above mean sea level (mamsl).

Figure 18.5: Topographical Map of the Kobada Site

18.1.11.3 Terms of Reference

The terms of reference for the TSF DFS are as follows:

- Design a TSF with sufficient capacity to accept and store the LOM production of tailings.
- Design the TSF in compliance with applicable local legislation and prescribed internationally accepted guidelines.
- Determine the LOM costs for the TSF and provide an estimate of the capital cost thereof, based on the Association for Advancement of Cost Engineering International (AACE) Cost Estimate Classification System Class 2 accuracy (+20 %; −15 %).

The battery limits for the DFS design of the TSF are as follows:

- The perimeter fence around the TSF
- Downstream of the point where the slurry delivery pipeline crosses the toe line of the TSF wall
- Upstream of the suction end of the floating barge

- The top surface of the decant/excess water pond on the TSF (i.e. excluding the decant barge)

18.1.11.4 Design Criteria and Project Information

18.1.11.4.1 TSF Design Parameters

The key TSF design parameters are summarised in Table 18.9 with references to the source of the information where applicable.

Table 18.9: Key TSF Design Parameters

Item	Design Criteria	Value	Source
1	Tailings Material	Gold	Kobada Gold Project
2	Typical Tailings Deposition Rate	3 Mt/a	SENET
3	LOM Storage	15.1 Years	SENET
4	Total Tonnage	45.15 Mt	SENET
5	Tailings Specific Gravity	Oxides: 2.75 Sulphides: 2.78	Test work from Specialised Testing Laboratory (Pty) Ltd (ST-Lab)
6	Tailings Particle Size Distribution	Oxides: 85 % passing 75 µm sieve Sulphides: 83 % passing 75 µm sieve	Test work from ST-Lab
7	Slurry Percentage Solids by Mass	35 % solids by mass	SENET
8	Tailings Void Ratio	1.2	Epoch estimated from oxides test work from (ST-Lab)
9	Placed Dry Density	1.25 t/m^3	Epoch – calculated from assumed void ratio and SG for oxides
10	Freeboard	1 m above the Inflow Design Flood	International Commission on Large Dams (ICOLD) guidelines recommendation for Very High Hazard Dam
11	Emergency Spillway	Designed to pass the Inflow Design Flood, with a spill depth of 500 mm serving as temporary storage	Canadian Dam Association (CDA) guidelines recommendation for Very High Hazard Dam
12	Geochemical Characteristics of Tailings	Presence of elevated arsenic concentrations to be precipitated out in the process plant to levels within the World Health Organisation (WHO) Guidelines (< 100 mg/L).	SENET
13	TSF Lining System	TSF will be HDPE lined	Project decision based on tailings geochemistry and generally accepted practice
14	Depositional Methodology	Spigot/open-ended discharge	Epoch
15	Type of Facility	Full containment	Epoch

Item	Design Criteria	Value	Source
16	Return Water Management Strategy	Return water sump	Epoch
17	Storm Water Management Strategy	Storm water diverted around tailings dam	Epoch
18	TSF Decant System	Barge system	Epoch
19	Maximum Height of TSF	36 m	Epoch
20	Peak Ground Acceleration (PGA)	0.06 g	Based on seismic hazard assessment from existing mine site located 330km south-east of Kobada, and CDA guidelines recommendation for Very High Hazard Dam
21	Survey Information	Minimum contour interval of 1 m and an accuracy of 0.1 m*	SENET
* Based on received PhotoSat survey			

18.1.11.4.2 Design Legislation/Codes/Guidelines

As Mali does not have any existing TSF design legislation, codes, or guidelines, the following guidelines were adopted for the TSF design:

- International Commission on Large Dams (ICOLD), *Tailings Dams Safety* (Draft)
- Canadian Dam Association (CDA), *Dam Safety Guidelines* (2013);
- Canadian Dam Association (CDA), *Application of Dam Safety Guidelines to Mining Dams* (2014);
- South African National Standard (SANS) 10286:1998, *Code of Practice for Mine Residue*
- South African Institution of Civil Engineering (SAICE) – Geotechnical Division, *Site Investigation Code of Practice* (2010)

18.1.11.4.3 LOM Tailings Production

A total of 45.15 Mt of tailings shall be produced during the LOM of 15 years (starting in the middle of Year 1 and ending in the middle of Year 16), as per Table 18.10.

Table 18.10: LOM Tailings Production Plan

Year	Tonnage per Annum	Cumulative Tonnes	
1	1,375,000	1,375,000	Oxides
2	2,998,000	4,373,000	
3	2,997,000	7,370,000	
4	3,011,000	10,381,000	
5	3,018,000	13,399,000	
6	2,982,000	16,381,000	
7	3,007,000	19,388,000	
8	3,011,000	22,399,000	
9	3,013,000	25,412,000	
10	3,028,000	28,440,000	Sulphides
11	3,016,000	31,456,000	
12	2,971,000	34,427,000	
13	2,993,000	37,420,000	
14	3,000,000	40,420,000	
15	2,973,000	43,393,000	
16	1,760,000	45,153,000	

18.1.11.4.4 General Project Information

The following general project information has been made available to Epoch for the DFS:

- A 1 m contour interval PhotoSat survey of the overall mine site with file name "kobada_wo4116_1m_contours_2019dec12", provided by SENET, on which the DFS design has been conducted
- An image in jpg format containing coordinates of the Mining and Exploration Licences with file name "Kobada Concessions_150918"; and
- The following reports:
  - Kobada Gold Mine Surface Water Quantities Assessment. Midrand (Peens & Associates Civil Engineering and Training Consultant (Pty) Ltd, 2020)
  - Executive Summary Report on Static Test Data for Mineral Samples (ABS Africa, 2021)
  - Report on A Geotechnical Investigation for Definitive Feasibility for the Tailings Storage Facility for the Kobada Gold Project Mali (Inroads Consulting, 2020)

18.1.11.4.5 Climatic/Hydrological Information

The meteorological data for the design was adopted from the Kobada Gold Mine Surface Quantities Assessment report by Peens & Associates (2020).

The information provided has been used in the determination of

- Design flood storage and flow depths through emergency spillways
- Freeboard requirements
- Design of storm water runoff diversions and associated control structures
- A deterministic monthly water balance (based on average normal year monthly, wettest year monthly, driest year monthly rainfall and evaporation figures) to estimate the volumes of excess slurry water and storm water runoff available for return to the plant, and the frequency and volumes of such water discharged

18.1.11.4.5.1 Rainfall and Evaporation Data

Kobada lies within the zone of West Africa where the wet season occurs between July and October. The mean annual runoff ranges between 2 L/s/km^2 and 8 L/s/km^2 and over 60 % of the total runoff is concentrated in the period from the middle of August to the middle of October. Low flow occurs from December to June. On average, the wettest month is August. The mean annual rainfall for the normal year is 1 250 mm. The monthly rainfall figures for normal, wettest and driest years are presented in Table 18.11.

The mean annual potential evaporation used was that of Kalana Town, located 104 km south of Kobada, and is 2 623 mm. The highest monthly evaporation occurs in March. The open surface evaporation is estimated to be 1 888 mm, based on the estimated ratio of 0.72 between potential evaporation and open water evaporation for Lake Chad. Monthly evaporation figures for potential and open surface evaporation are presented in Table 18.12.

Table 18.11: Monthly Rainfall Data

Month	Normal Year (mm)	Driest Year (mm)	Wettest Year (mm)
Oct	102	63	137
Nov	10	0	10
Dec	2	0	0
Jan	0	0	0
Feb	1	0	1
Mar	6	0	6
Apr	32	5	40
May	87	68	107
Jun	179	154	192
Jul	242	170	285
Aug	329	295	365
Sep	263	206	318
Total	**1 252**	**961**	**1 461**
Source: Peens & Associates, 2020			

Table 18.12: Monthly Potential Evaporation Data

Month	Potential Evaporation (mm)	Open Surface Evaporation (mm)
Oct	199	143
Nov	204	147
Dec	223	161
Jan	249	179
Feb	276	199
Mar	303	218
Apr	269	194
May	234	168
Jun	188	135
Jul	160	115
Aug	153	110
Sep	165	119
Total	**2 623**	**1 888**
Source: Peens & Associates, 2020		

18.1.11.4.5.2 Extreme Design Storm Events

Design storm events have been determined and are presented as follows:

- Rainfall depths as a function of the recurrence interval and event duration (see Table 18.13)
- Rainfall intensities as a function of the recurrence interval and event duration (see Table 18.14).

Table 18.13: Design Rainfall for Daily Events

Duration (d)	Design Rainfall (mm)								
	Recurrence Interval								
	1:2	1:5	1:10	1:20	1:50	1:100	1:200	1:1000	1:10 000
1	87	158	214	276	368	449	536	765	1 210
2	104	176	233	296	390	472	561	793	1 253
3	112	185	243	307	401	484	573	807	1 274
4	120	194	253	317	412	495	585	820	1 296
5	129	203	262	327	423	507	598	834	1 317
6	137	212	272	337	434	518	610	848	1 338
7	145	221	282	347	445	530	622	862	1 360
30	336	427	504	582	697	796	906	1 180	1 852
Source: Peens & Associates, 2020									

Table 18.14: Rainfall Intensity as a Function of Recurrence Interval and Duration

Duration (d)	Rainfall Intensity (mm)								
	Recurrence Interval								
	1:2	1:5	1:10	1:20	1:50	1:100	1:200	1:1000	1:10 000
0.1	15	27	36	47	63	76	91	130	206
0.25	28	51	68	88	118	144	172	245	387
0.5	40	73	98	127	169	207	247	352	556
1	52	95	128	166	221	270	322	459	726
2	63	114	154	199	265	324	386	551	871
3	68	123	167	215	287	351	418	597	943
4	71	130	175	226	302	369	440	627	992
5	74	134	182	235	313	382	456	650	1 028
6	76	137	186	240	320	391	466	666	1 052
8	78	142	192	248	331	404	483	689	1 089
10	80	145	197	254	339	413	493	704	1 113
12	82	149	201	259	346	422	504	719	1 137
18	85	155	209	270	361	440	525	750	1 185
24	87	158	214	276	368	449	536	765	1 210
Source: Peens & Associates, 2020									

18.1.11.4.5.3 Probable Maximum Precipitation

The 24 h duration probable maximum precipitation (PMP) has been determined as 1,703 mm by Peens & Associates (2020).

18.1.11.4.5.4 Runoff

The records of the Douna Station in the Bani River were deemed to be the only records that were of sufficient quality and length to be utilised in the long-term analysis and were, therefore, adopted to determine the monthly and annual runoffs. Monthly runoff figures for normal, wettest and driest years are presented in Table 18.15.

Table 18.15: Monthly Runoff Figures

Month	Driest Year (mm)	Normal Year (mm)	Wettest Year (mm)
Oct	21	45	66
Nov	7	20	28
Dec	2	7	9
Jan	1	3	4
Feb	1	2	2
Mar	0	1	2
Apr	0	1	1

Month	Driest Year (mm)	Normal Year (mm)	Wettest Year (mm)
May	0	1	1
Jun	1	1	1
Jul	2	4	5
Aug	12	22	28
Sep	24	45	64
Total	**71**	**152**	**211**

18.1.11.5 Site Selection and 2020 DFS

The site selection assessment undertaken prior to the initial DFS conducted in 2020 is summarised in the Kobada TSF DFS Site Selection Report (2020). From this study, the site named “Option 8” was selected as the preferred site out of four options for the DFS design. The site options considered in the initial site selection prior to the DFS are shown in Figure 18.6.

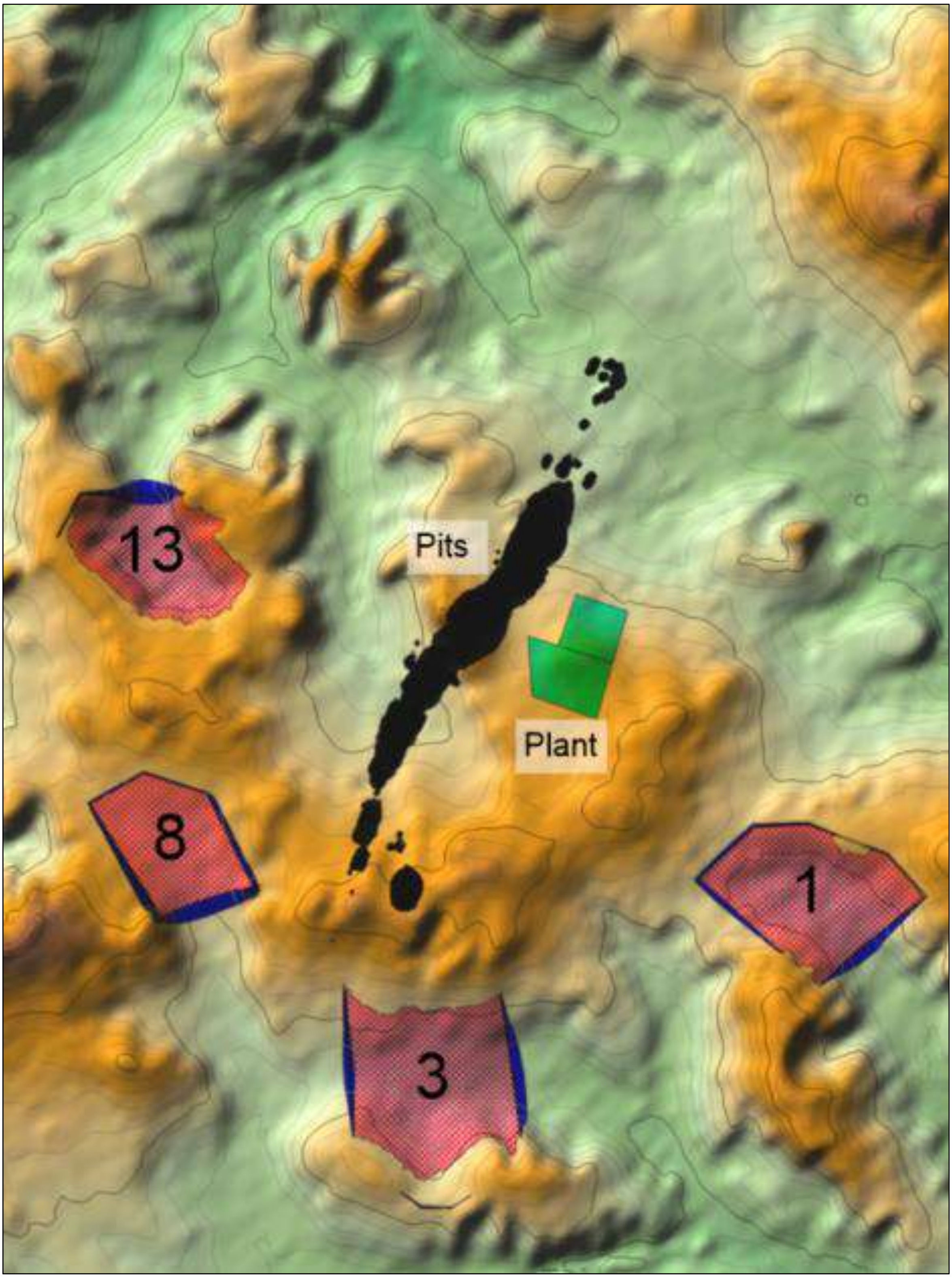


Figure 18.6: Site Option Considered in the Initial Site Selection Assessment

The site was further revised to store 26 Mt for the DFS completed in 2020, documented in the Kobada TSF DFS Report (2020), and thereafter revised to 45 Mt, on which the current 2021 DFS is based. Figure 18.7 and Figure 18.8 depict the 2020 DFS vs 2021 DFS update TSF footprints and embankment walls, respectively.

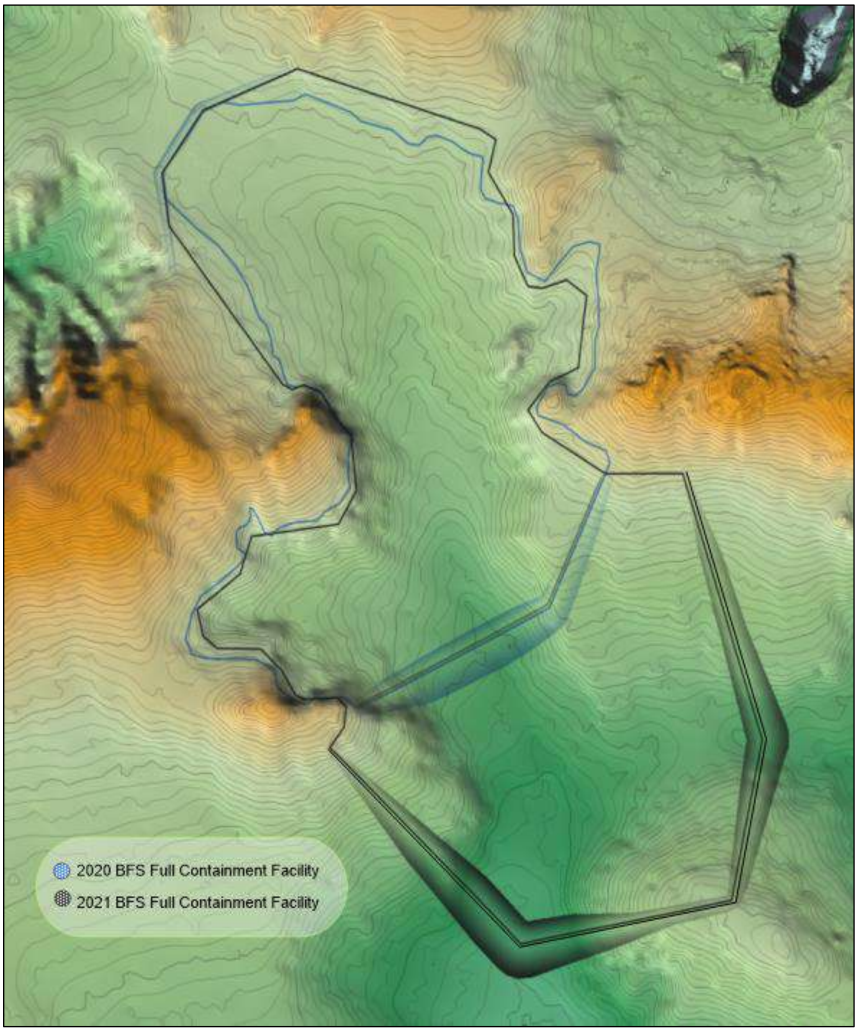


Figure 18.7: Pre-Feasibility, 2020 DFS and 2021 DFS TSF Footprints

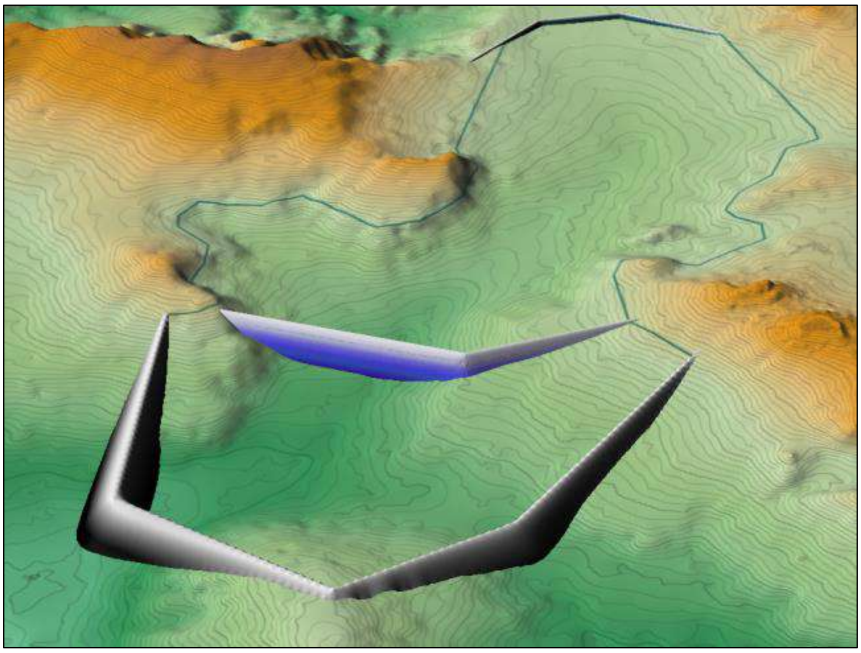


Figure 18.8: 2020 vs 2021 DFS TSF Embankment Walls

18.1.11.5.1 Impact of DFS Update on Site Selection

The site selection has been reassessed to confirm that the selected site location (Option 8) is still favourable for the larger facility (storing 45 Mt of tailings). Option 13 from the initial site selection was deemed unsuitable for consideration due to the direct proximity to the Kobada Village. Table 18.16 summarises the key site selection aspects taken into consideration for the new required storage capacity for the remaining three site options. The infrastructure, agricultural and environment impacts were provided by ABS Africa.

Table 18.16: Revised Site Selection Aspects

	Option 1	Option 3	Option 8
Structures / temporary structures within 500m buffer zone of TSF footprint	23	2	87
Failure ZOI	Eastern failure - 812 Structures Southern failure - Several (5-10) temporary isolated/ structures	South-eastern failure – 27 structures/temporary structures South-western failure – 0 obvious structures/ temporary structures	Southern failure- 3 temporary structures 315 Structures (Frontline Camp) Southwest - 2 temporary structures
Agricultural activities (approximate ha)	14.5	27.5	40.4
Biodiversity natural habitat (ha)	87	195	56
Biodiversity value (ha)	77	50	43

	Option 1	Option 3	Option 8
Plant distances away (approximate m)	1530	1030	2160
Pit distances away (approximate m)	2340	1500	415
Artisanal mining	Within footprint and within failure ZOI.		
Potential resources/reserve	Field work and drilling has confirmed mineralisation in near proximity to the east this option, in addition to a soil geochemical sampling anomaly. This option therefore presents potential for future resources.	Field work confirmed geochemical surface anomalies, indicating potential resource interest, but hasn't been assessed further and is earmarked for further assessment.	Drilling in this area confirmed this option is of low resource potential in comparison to other potential prospects.

Option 3 is preferable to Option 1 and the current Option 8 from a resettlement point of view, as the tailings zone of influence doesn't encroach on any villages. The assessment would require further confirmation, as these areas were not included in the social survey. Figure 18.9 depicts the three site options and their zone of influence. Evidently, the Unnamed Village and Frontline Camp are within the tailings zone of influence of Options 1 and 8, respectively, which would require relocation.

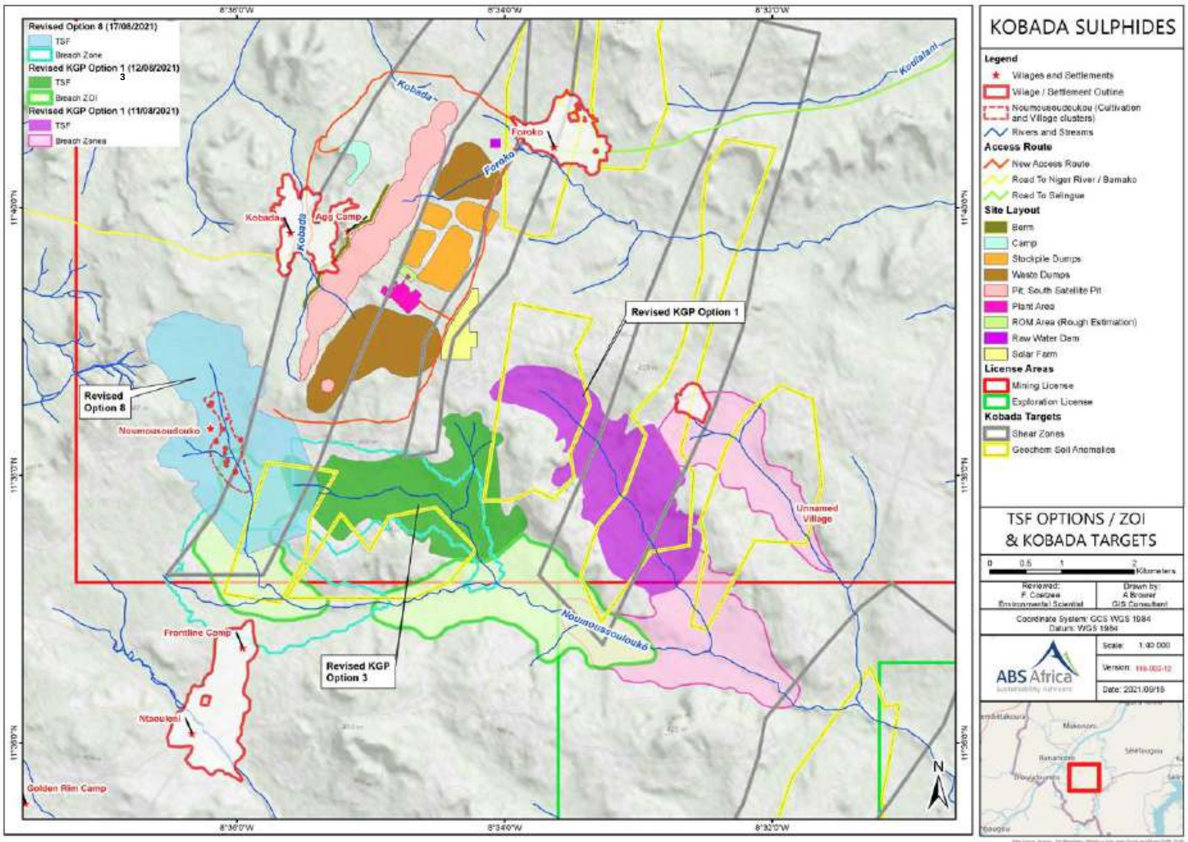


Figure 18.9: Shear Zones and Geochemical Anomalies Relative to TSF Footprint Options (ABS Africa)

Figure 18.9 also depicts the shear zones and geochemical anomalies, assessed thus far, across the site. The current Option 8 straddles over an area which, in the previous DFS, was identified as an area for potential future reserves. Some drilling since the previous DFS has been undertaken in this area and doesn't suggest any significant interest potential in comparison to other potential prospects. Option 3 has some geochemical surface anomalies indicating potential resource interest but hasn't been assessed further and is earmarked for further assessment. This option is therefore currently considered as a no-go area until the potential for future resource is assessed. Field work and drilling has confirmed mineralisation in near proximity to the east of Option 1, in addition to a soil geochemical sampling anomaly, therefore this option presents potential for future resources.

Based on the above, although Option 8 has more resettlement aspects compared to the other two sites, it is the only site which does not currently lay over potential future resources, making it the preferred candidate site. If further assessment in the future provides confirmation that Option 3 is not of interest from a potential resource perspective, it may be considered preferable to Option 8.

18.1.11.6 Characterisation of the Tailings

18.1.11.6.1 Tailings Geotechnical Testing

Table 18.17 summarises the tailings parameters based on test work conducted by Specialised Testing Laboratory (Pty) Ltd (ST-Lab). The TSF will receive oxides for the first nine years of the LOM, and will thereafter receive sulphides for the final six years.

Table 18.17: Material Parameters Adopted for the TSF Slope Stability Analyses

Material Description	Unit Weight (kN/m^3)	Effective Friction Angle, φ' (degrees)	Effective Cohesion, c' (kPa)	Hydraulic Conductivity (m/s)
Oxide Tailings	13.7	25	0	1×10^{-8}
Sulphide Tailings	13.3	33	0	4×10^{-8}

The particle size distribution (PSD) curves for the tailings based on test work conducted by Maelgwyn Mineral Services Africa (MMSA) (provided by SENET) and ST-Lab are depicted in Figure 18.10. The curves indicate that a minimum of 85 % of the particles pass the 75 µm for the oxides and 83 % for the sulphides. Typical gold tailings PSD curves from past projects are also depicted for comparison. It is evident that the Kobada tailings have a higher fine fraction compared to the other gold tailings samples shown. The sulphides are coarser than the oxides, as expected.

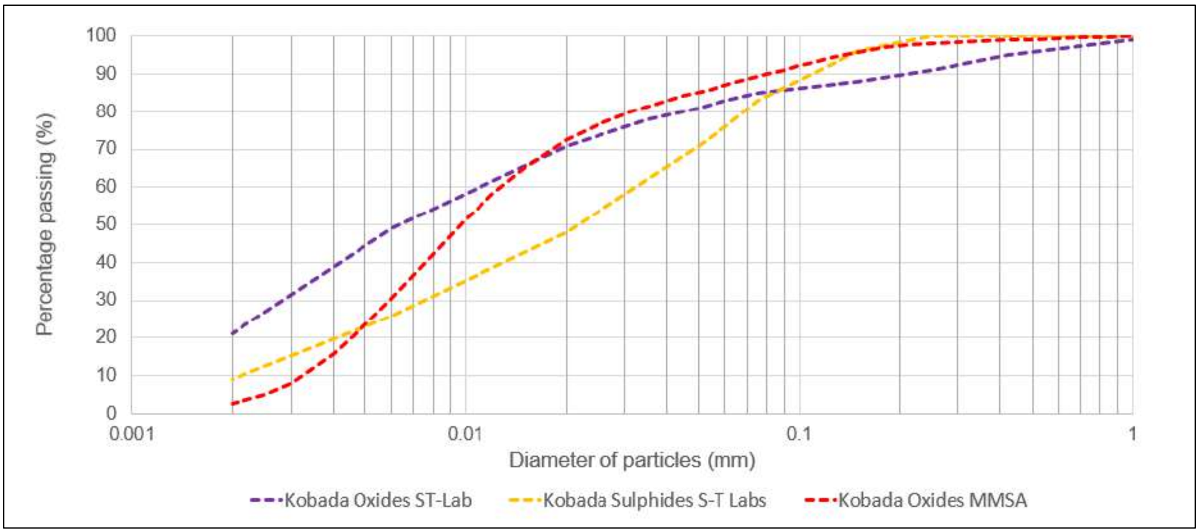


Figure 18.10: Tailings PSD for the Kobada Gold Tailings vs Gold Tailings from Various Projects

18.1.11.6.2 Tailings Geochemistry

The geochemistry analysis for Kobada was conducted on ore samples taken predominantly from the saprolite, with small portions from the laterite and transition zones to mimic the intended mining proportions. The geochemical testing and result analyses were done by Moss Laboratories and provided by ABS Africa in the Static Test Data for Mineral Samples Report (2020).

The tests conducted on the samples were as follows:

- Whole rock element analysis performed by X-ray fluorescence (XRF) (Thermo Scientific ICap). Trace element and total digestion quantification performed by Inductively Coupled Plasma Mass Spectrometry (ICP-MS) (Agilent 7700 ICP-MS)
- Acid-Base Accounting (ABA) conducted to determine the acid neutralising capacity using the Sobek and Modified Sobek (Lawrence) Methods
- Net Acid Generation (NAG) test performed as per the AMIRA P387A ARD Test Handbook
- Deionised water leach test to determine the leaching of cations from the samples by a reagent water leach, as per the Australian Standard Leaching Procedure (AS 4439.1)

The results and conclusions arising from the tests and subsequent analyses (ABS Africa, 2020) are as follows for the oxides:

- The XRF data for the nineteen samples identified barium and copper at concentrations that exceeded the South African total concentration threshold (TCT) for non-hazardous waste (TCT0), but only marginally, using the South African National Norms and Standards for the Assessment of Waste for Landfill Disposal under the National Environmental Management: Waste Act, 2008 (Act No. 59 of 2008). The primary concern was arsenic, which was present at concentrations that would cause the material to be classified as hazardous (> TCT2).

- The reagent leach data showed that the barium and copper concentrations in the leachate for all samples were below the WHO's drinking water guidelines, indicating that mobility is low.
- The ABA analysis showed that the sulphur grade was very low for all but one of the samples from the cyclone testing. Consequently, the maximum potential acidity for most of the samples was very low.
- The acid neutralising capacity was low (< 10 kg of H_2SO_4/t) for all but the one sample from the cyclone testing mentioned above. This is consistent with the XRF data that showed low concentrations of alkaline earth elements in the samples.
- The calculated net acid production potential (NAPP) values were negative for all samples, except for the one sample from the cyclone testing mentioned above and the North domain comminution test sample, where the value is positive, but below 7 kg H_2SO_4/t, which is considered low.
- The NAG tests confirmed the ABA data, with the NAG pH exceeding 4.5 in almost all cases, suggesting that any mineral acidity generated by the oxidation of reduced sulphur had been neutralised. Several samples had NAG pH values below 7, suggesting a small amount of residual acidity. This could be due to metal ions (which could undergo hydrolysis) or organic acids. The only exception was the cyclone sample, where the NAG pH is 3.05, suggesting a small amount of mineral acidity.
- The geochemical characterisation plot, which uses both ABA and NAG test data, indicated that eighteen of the nineteen samples may be classified as non-acid forming.
- The reagent water leach tests yielded leachate with a pH above 6.96 in all cases, suggesting that acid-generating salts were not present at significant levels in any sample. The exceptions were the three samples from the comminution tests, where the pH was between pH 5.2 and 5.4. For these samples the Electronic Conductivity (EC) value was only marginally above deionised water, so the magnitude of any acid released was inconsequential.
- The base metal and metalloid analysis of the leachate yielded concentrations that were below the health-based WHO guidelines for all elements, with the exception of arsenic, which was consistently above the guideline limits.
- The arsenic concentration in the leachate for the comminution test samples, representative of the waste material, did exceed the WHO drinking water guideline, but was well below the Malian and International Finance Corporation (IFC) standards.
- Of the twelve cyclone samples tested, only one registered an arsenic concentration in excess of the IFC Environmental Health and Safety (EHS) level for effluent while the arsenic concentrations for the rest of the samples were consistently less than 30 % of the IFC guideline limit.
- The arsenic concentrations in the reagent water leachate for the detoxified tailings samples and the liquid fraction of the tailings slurry exceeded the IFC EHS guideline, most likely due to increased arsenic mobility at high pH values. To prevent any impact on the receiving waters, the tailings should be confined to a secure, fit-for-purpose TSF.

The key points for the oxides were summarised by ABS Africa (2020) as follows:

> "The static tests performed on the nineteen samples indicate that there is very limited risk of acid generation, but the presence of elevated arsenic concentrations is a concern. While mobility during the water leach tests was low, the concentrations still exceeded the drinking water standard in most of the tests."

The results and conclusions arising from the tests and subsequent analyses for the sulphides are as follows:

- All samples tested contained significant amounts of arsenic, with concentrations for all thirty-five samples exceeding the South African TCT 0 value (5.8 mg/kg). The mobility of the arsenic was significant, with the arsenic concentration in the reagent water leachate of all thirty-five samples exceeding the Malian standard (50 µg/l) and thirty-two of the thirty-five samples exceeding the IFC EHS value (100 µg/l).
- The sulphur grades were low (< 0.5%) for thirty-three of the thirty-five samples tested, suggesting limited potential for acid generation. The two exceptions were one ore sample (1.08%) and the waste rock samples (0.63%).
- The measured acid neutralising capacity (ANC) values were relatively high (>20 kg H_2SO_4/t) for most of the samples and were, in all cases, higher than the calculated maximum potential acidity (MPA) values. This resulted in negative NAPP values in all cases.
- The NAG tests confirmed the tailings samples had insignificant acid generating potential and only three of the twenty-two waste rock samples resulted in a NAG pH below pH 4.5. For these three samples, the magnitude of the NAG was low, with the highest value (10.69 kg H_2SO_4/t).
- The NAPP and NAG pH values can be used to classify the material as potentially acid forming (PAF), uncertain or non-acid forming (NAF). Based on the test data, none of the thirty-five samples were classified as potentially acid forming, eight samples (four ore and 4 waste rock) were classified as uncertain and the remaining twenty-seven could be classified as non-acid forming.
- For most of the samples, the concentrations of potentially hazardous metals and metalloids in the leachate were acceptable, below the WHO drinking water, IFC EHS and Malian standards. The only exceptions were two tailings samples that had antimony concentrations marginally above the WHO standard and a waste rock sample, which has a manganese concentration marginally more than the Malian standard.

The key points for the sulphides were summarised by ABS Africa (2021) as follows:

> "The majority of the material tested exhibited limited acid generating potential and moderate acid neutralising potential. Therefore, the risk of acidic drainage is low.
>
> All thirty-five samples evaluated contained high to extremely high arsenic concentrations and mobilisation of the arsenic was significant in most cases, even used deionised water as a leach solution. There is a significant risk that leachate arising from the percolation of water through tailings, waste rock impoundments and ore stockpiles could accumulate arsenic to concentrations in excess of the Mali standards."

SENET has, however, incorporated measures to precipitate arsenic levels out in the process plant to achieve concentrations below 100 mg/L as per the WHO guidelines. These measures were not assessed as part of the original ABS Africa evaluation.

The analyses conducted on the three comminution test samples, representative of the oxide waste material, indicated very little reactivity, both from an acid generation and mineral liberation perspective. There was limited mobilisation of iron and arsenic, but this is likely to have been influenced by the small particle size. The arsenic concentration of the sulphides material, however, significantly exceed the test results of the oxide material tested during the 2020 DFS. The sulphide waste material would therefore not be used for construction of the TSF embankment.

Based on the above information and the fact that the process includes cyanide (with an associated cyanide treatment plant), the decision to HDPE line the TSF was accepted as a best practice approach.

18.1.11.7 Geotechnical Investigation

18.1.11.7.1 Geotechnical Site Investigation of the Preferred Site

InRoads Consulting (Pty) Ltd (InRoads), under the appointment of Epoch, undertook a geotechnical site investigation of the preferred TSF footprint identified for the design. The test pit locations from the geotechnical site investigation are shown in Figure 18.11. The number and position of test pits were determined according to the initial configuration for the self-raising facility with the associated return water dam and storm water dam. The site investigation and profiling of the test pits were carried out in accordance with the guidelines specified in the *Site Investigation Code of Practice* (SAICE, 2010). The previous DFS and updated DFS TSF footprints are also shown in Figure 18.11 and indicates that the geotechnical site investigation carried out did not include test pits near the critical section of the relocated embankment wall and basin.

The geology of the area investigated comprises primarily sedimentary rocks, and the terrain is intensely lateralised with large laterite plateaus covering the region to depths of up to 25 m. Saprolite, which underlies the laterite, is variable in colour, shows very slight variation from a clay to a fine silty clay, and is a residual from siltstone and mudstone. Groundwater was not encountered in any of the pits excavated within the footprint of the TSF; however, it was noted that water was standing in two water wells excavated by the villagers to depths of 4.5 m located within the vicinity of the proposed storm water dam.

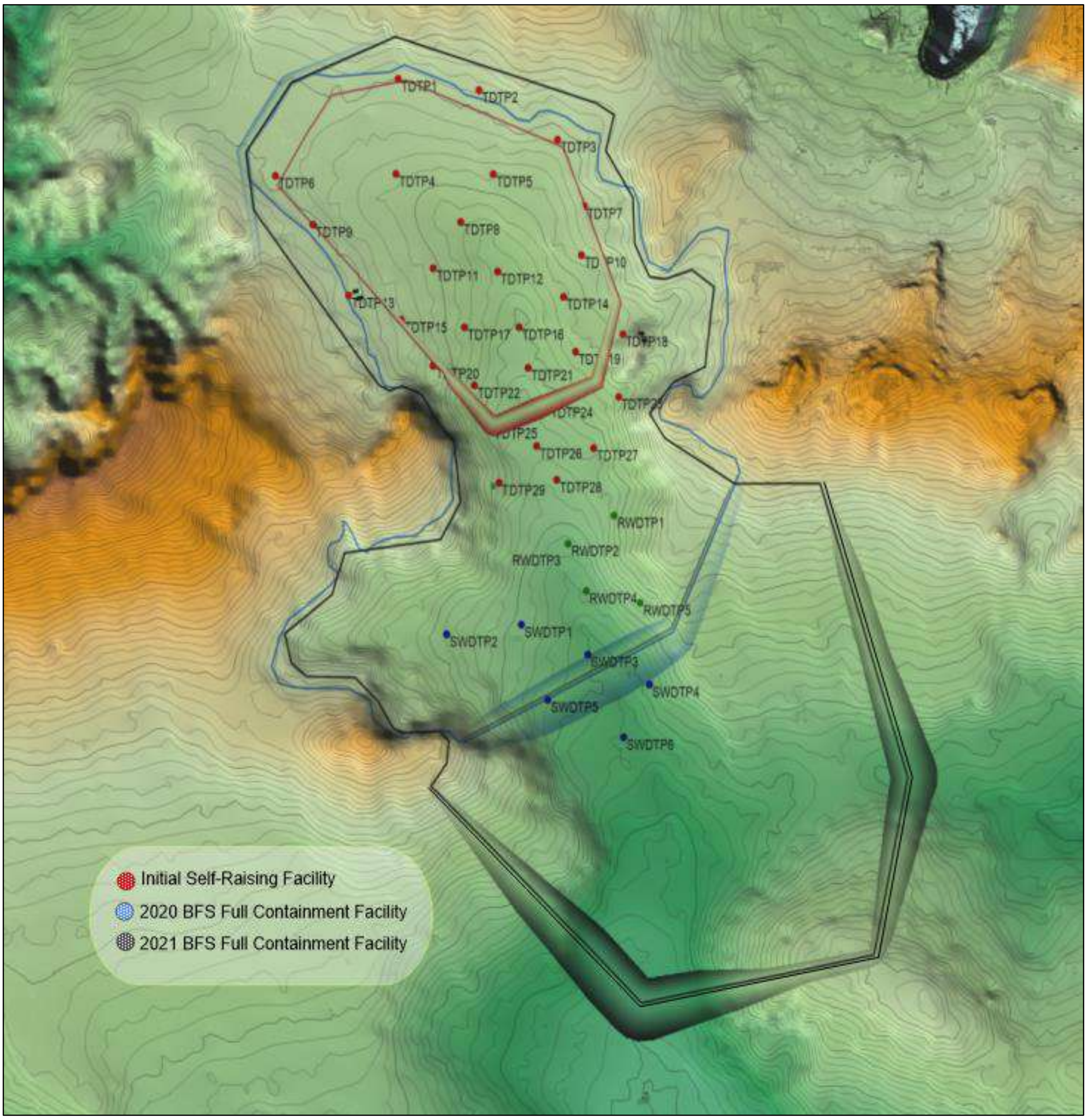


Figure 18.11: Test Pit Locations for the Kobada Pre-feasibility, 2020 DFS and 2021 DFS

A total of 40 test pits were excavated within the footprint of the TSF. The pits were excavated to refusal or partial refusal with a 330 excavator, with refusal averaging at a depth of 1.7 m below ground level. The typical subsoil conditions are described by Inroads as follows:

- Topsoil as a 0.1 m to 1.0 m thick layer of loose gravels in a clayey silt matrix, or a stiff clayey silt containing many roots, blankets the site.
- Transported gravels as loose and medium dense gravels in a clayey silt matrix underly the topsoil where they extend to an average depth of 1.1 m below the present ground surface.

- Transported clayey silt is generally present as a yellow-brown or grey, stiff, shattered, clayey silt, which is often pinholed and potentially collapsible. It extends to an average depth of 1.0 m below the present ground surface.
- Nodular laterite as medium dense and dense, weakly cemented, fine and medium laterite nodules in a clayey silt matrix underlies the transported soils or topsoil in places. It extends to an average depth of 1.2 m below the present ground surface.
- Laterite was encountered in all the test pits at an average depth of 1.7 m below surface and in almost all cases extended to the bottom of the test pits. It typically occurs as a strongly cemented, ferruginous, gravelly silt or silty gravel and is of very dense to soft rock consistency. The excavator refused within this horizon in all but two of the test pits.
- Residual siltstone as a stiff silty clay or stiff silt was encountered below the laterite in test pits TP1 and TP26 only.

Typical soil profiles are shown in Figure 18.12 and Figure 18.13 for test pits within the basin and near the location of the full containment wall, respectively.

Figure 18.12: Test Pit TD19 Profile within the TSF Basin

Figure 18.13: Test Pit SWD5 Near the Location of the Full Containment Wall

18.1.11.7.2 Geotechnical Laboratory Test Work of the In-Situ Soils

The soil parameters for the study are based on laboratory test results from samples taken during the geotechnical site investigation and are shown in Table 18.18. The in-situ clayey silt and gravel layers typically found across the site are shown in Figure 18.14 and Figure 18.15.

Table 18.18: Inroads Recommended Design Parameters

Test Pit No.	Depth (m)	Description	c' (kPa)	φ' (°)
TDTP12	0.0-2.0	Transported clayey silt	0	31
TDTP20	0.0-1.5	Transported gravels in a clayey sand matrix	0	33
TDTP21	0.0-1.3	Laterite	0	38
TDTP26	0.0-3.8	Laterite mix with silt and clay	0	34
TDTP27	0.5-1.9	Laterite	0	35
TDTP29	0.0-2.3	Nodular laterite	11	33
SWDTP6	0.7-1.7	Transported clayey silt	5	26
RWDTP2	0.0-2.0	Laterite	0	36
RWDTP4	0.0-1.5	Transported clayey silt with nodules	22	27

Figure 18.14: Close-Up of the Clayey Silt Near the Ground Surface Below Topsoil

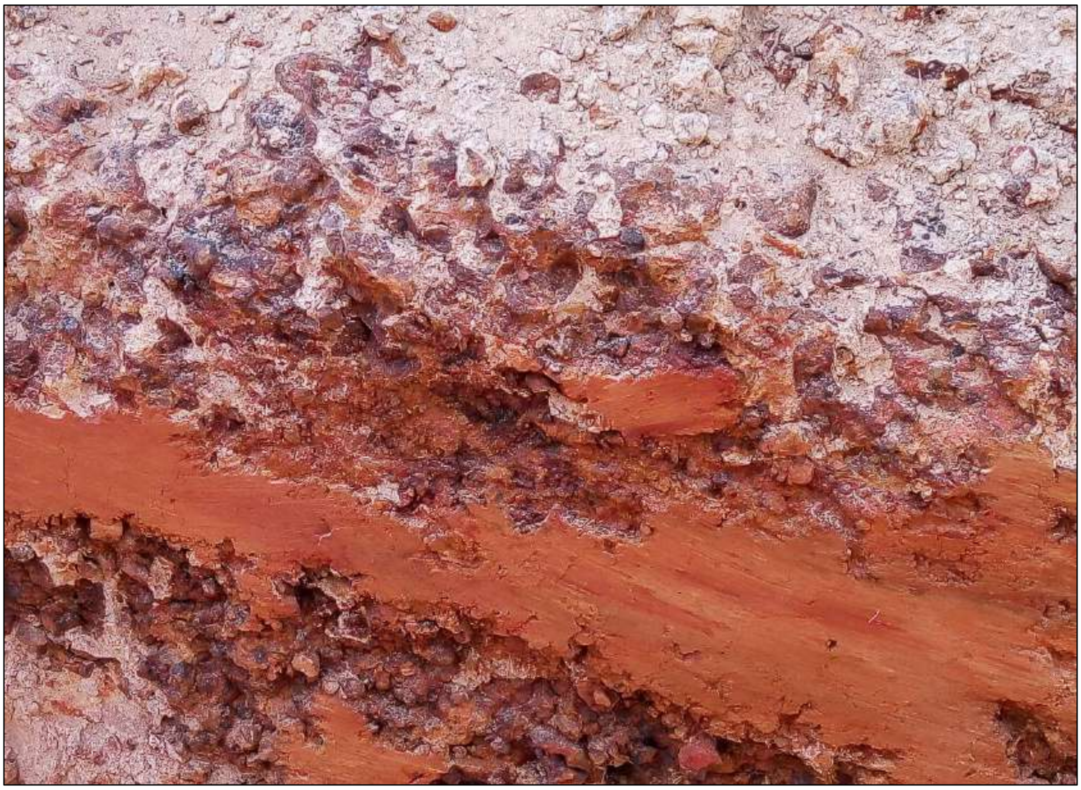


Figure 18.15: Close-Up of the Laterite Gravels in a Matrix of Clayey Silt

18.1.11.8 Tailings Storage Facility Design

The TSF comprises the following:

- A 1,500 µm HDPE-lined, full-containment valley tailings dam
- Associated tailings dam infrastructure, including the slurry distribution pipeline, catchment paddocks, toe drain system, underdrainage system, curtain drain system, blanket drain system, solution collection pipeline, collection sumps and manholes, seepage cut-off trench, storm water diversion trenches, emergency spillways, access roads and perimeter fence line
- A floating barge to decant supernatant tailings slurry water and storm water from the facility back to the plant

Figure 18.16 displays the general layout of the TSF within the mining site. Figure 18.17 shows a close-up of the TSF configuration.

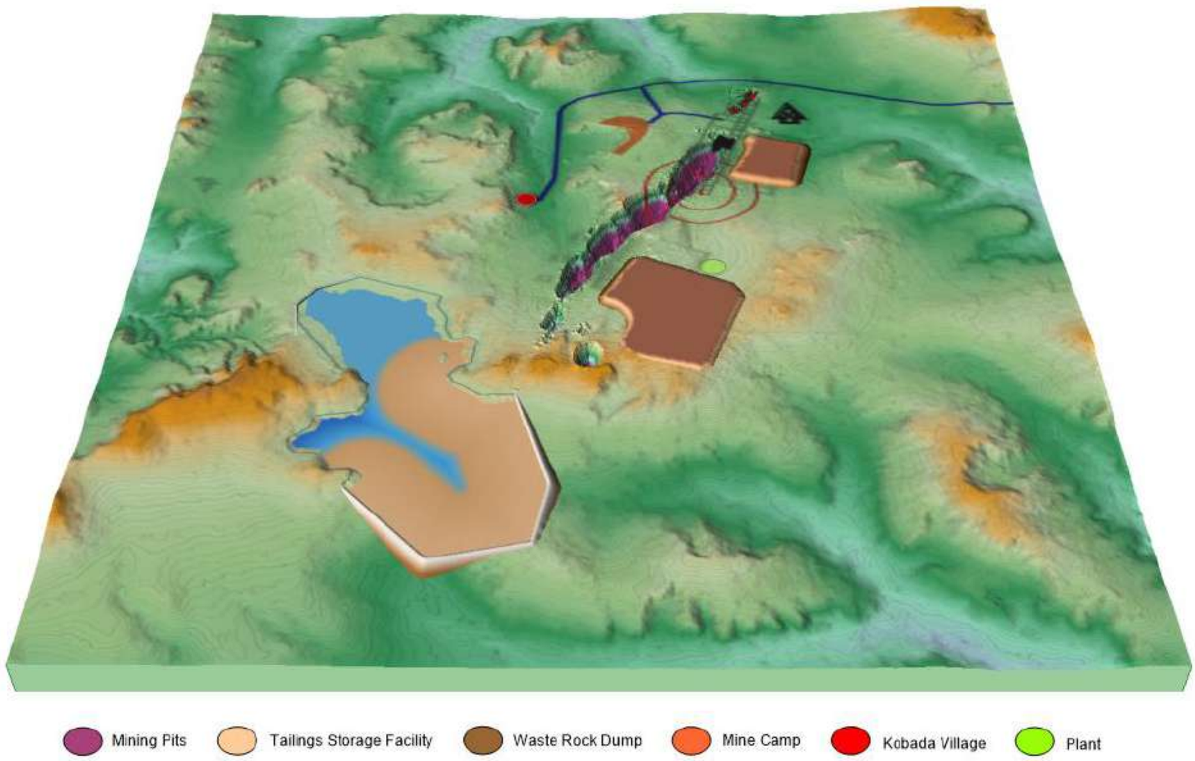


Figure 18.16: General Arrangement of the Kobada Mine Site

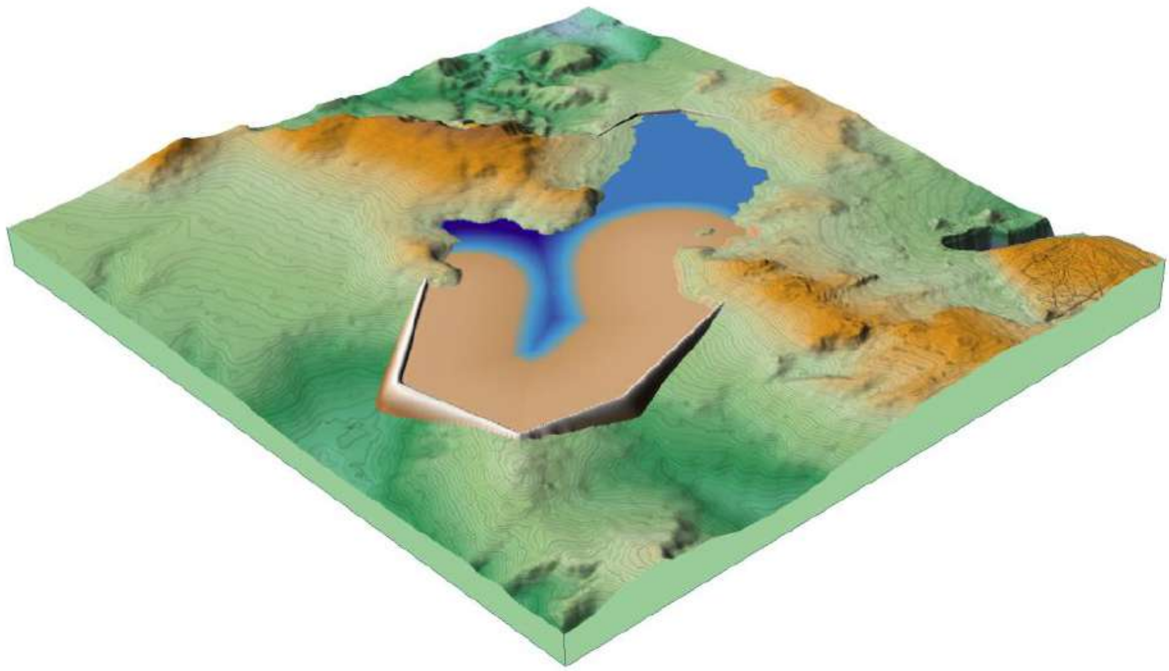


Figure 18.17: Close-Up of the Kobada TSF Site

18.1.11.8.1 Tailings Dam Safety Classification and Zone of Influence

The classification of the TSF has been carried out in accordance with the CDA *Dam Safety Guidelines* and is summarised in Table 18.19, with the appropriate levels applicable to Kobada's TSF highlighted in green and the selection of the dam class explained below. The safety classification system serves to provide a consistent means of differentiating between low, significant, high, very high and extreme hazard deposits based on their potential to cause harm to life, the environment and infrastructure. The classification scheme may be used to provide guidance on the standard of care expected of dam owners and designers. The consequence classification of the dam determines the target levels for the design criteria including flood events, earthquake loading and wind events, which may be applied for the construction, operation, and transition phases of a TSF.

Table 18.19: Dam Classification as per the CDA *Dam Safety Guidelines*

Dam Class	Population at Risk[a]	Loss of Life[b]	Environmental and Cultural Values	Infrastructure and Economics
Low	None	0	Minimal short-term loss No long-term loss	Low economic losses: area contains limited infrastructure or services
Significant	Temporary only	Unspecified	No significant loss or deterioration of fish or wildlife habitat Loss of marginal habitat only Restoration or compensation in kind highly possible	Losses to recreational facilities, seasonal workplaces, and infrequently used transportation routes
High	Permanent	10 or fewer	Significant loss or deterioration of important fish or wildlife habitat Restoration or compensation in kind highly possible	High economic losses affecting infrastructure, public transportation, and commercial facilities
Very high	Permanent	100 or fewer	Significant loss or deterioration of critical fish or wildlife habitat Restoration or compensation in kind possible but impractical	Very high economic losses affecting important infrastructure or services (e.g. highway, industrial facility, storage facilities for dangerous substances)
Extreme	Permanent	More than 100	Major loss of critical fish or wildlife habitat Restoration or compensation in kind impossible	Extreme losses affecting critical infrastructure or services (e.g. hospital, major industrial complex, major storage facilities for dangerous substances)

[a] Definitions for population at risk:

None – There is no identifiable population at risk, so there is no possibility of loss of life other than through unforeseeable misadventure.

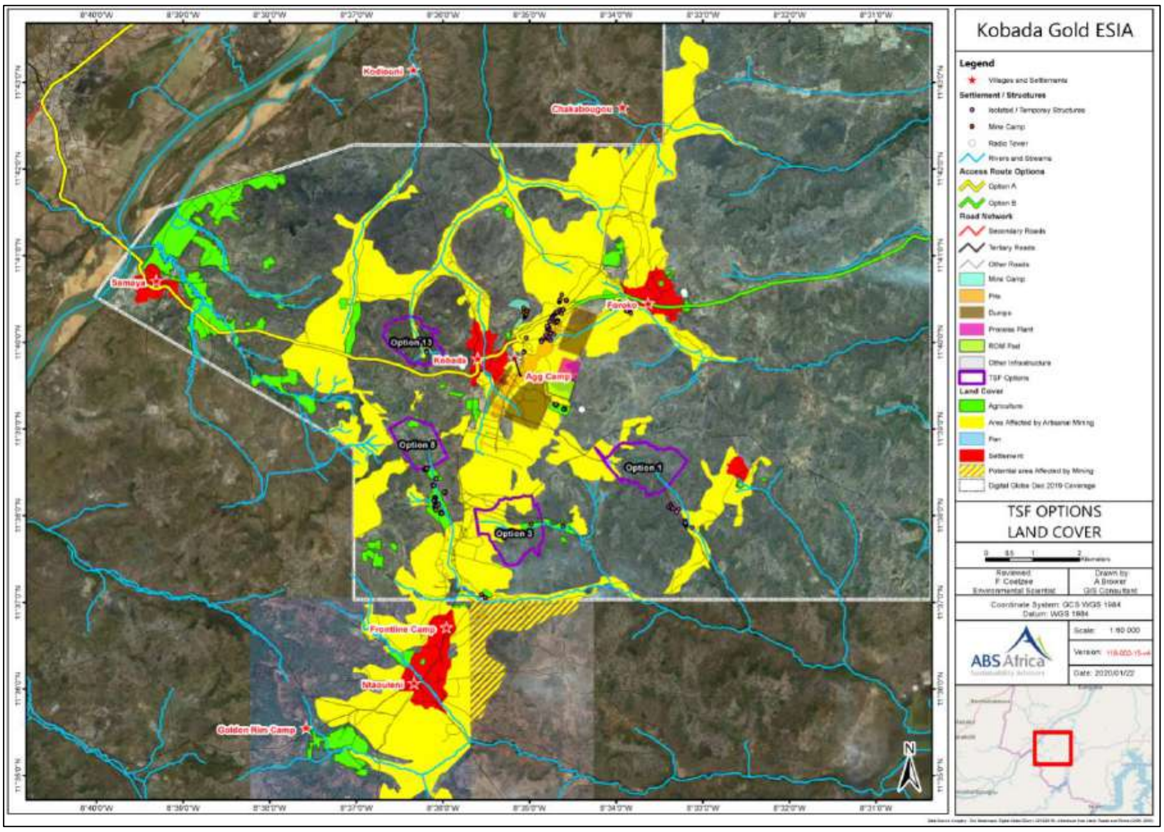
Temporary – People are only temporarily in the dam-breach inundation zone (e.g. seasonal cottage use, passing through on transportation routes, participating in recreational activities).

Permanent – The population at risk is ordinarily located in the dam-breach inundation zone (e.g. as permanent residents); three consequence classes (high, very high, extreme) are proposed to allow for more detailed estimates of the potential loss of life (to assist in decision-making if the appropriate analysis is carried out).

[b] Implications for loss of life:

Unspecified – The appropriate level of safety required at a dam where people are temporarily at risk depends on the number of people, the exposure time, the nature of their activity, and other conditions. A higher class could be appropriate, depending on the requirements. However, the design flood requirement, for example, might not be higher if the temporary population is not likely to be present during the flood season.

The classification system is based on the determination of a zone of influence for the facility corresponding to the area considered most likely to be affected by a flow slide emanating from the facility. Figure 18.18, provided by ABS Africa for the 2020 DFS, indicates the various settlements, road networks and land covers for the entire site within the study area.



Source: ABS Africa, 2020

Figure 18.18: Map of the Different Settlements, Roads and Land Covers

The TSF is to be constructed as a valley deposit behind a compacted earth embankment that is unlikely to liquefy should a seismic event take place. The delineation of the zone of influence of the TSF is, however, based on the premise that a failure of the containment wall occurs, resulting in the release of water and eroded/liquefied tailings from the facility.

The TSF zone of influence was analysed by means of Rift TD Advanced Tailings Deposition Modelling Software (Rift) TSF modelling of the full release of the stored tailings through a breach in the containment wall. A tailings resting slope of 1° was adopted, based on Lucia et al. (1981) and Blight et al. (1981), who state that liquified tailings have a low shear strength, which causes the saturated material to come to rest at an angle of between 1° and 4° where the ground slope is less than 4°.

Figure 18.19 shows the model, with the wall breach in the southern wall of the TSF, at the section of maximum height, with full release of the tailings stored downstream of the breach.

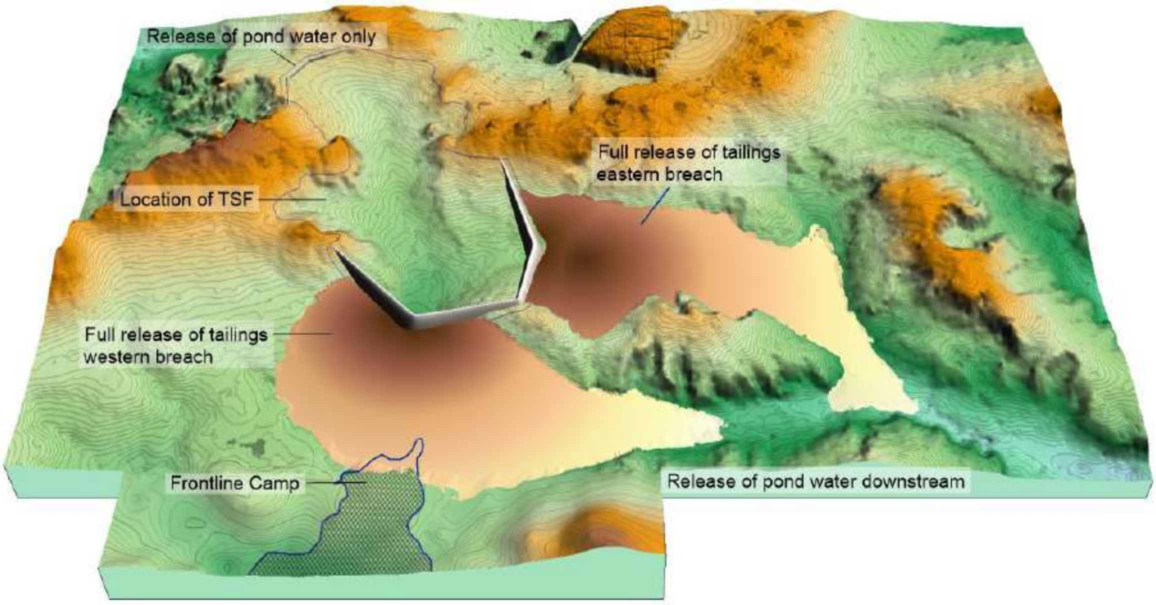


Figure 18.19: Breach in the Southern Walls of the TSF with Full Release of Tailings

Figure 18.20 shows the zone delineated using the SANS 10286 zone of influence method and the full release model method, depicted in red and blue, respectively. The various settlements, road networks and land covers shown in Figure 18.18 have been included to indicate the structures affected within the area delineated by the dam breach model and zone of influence.

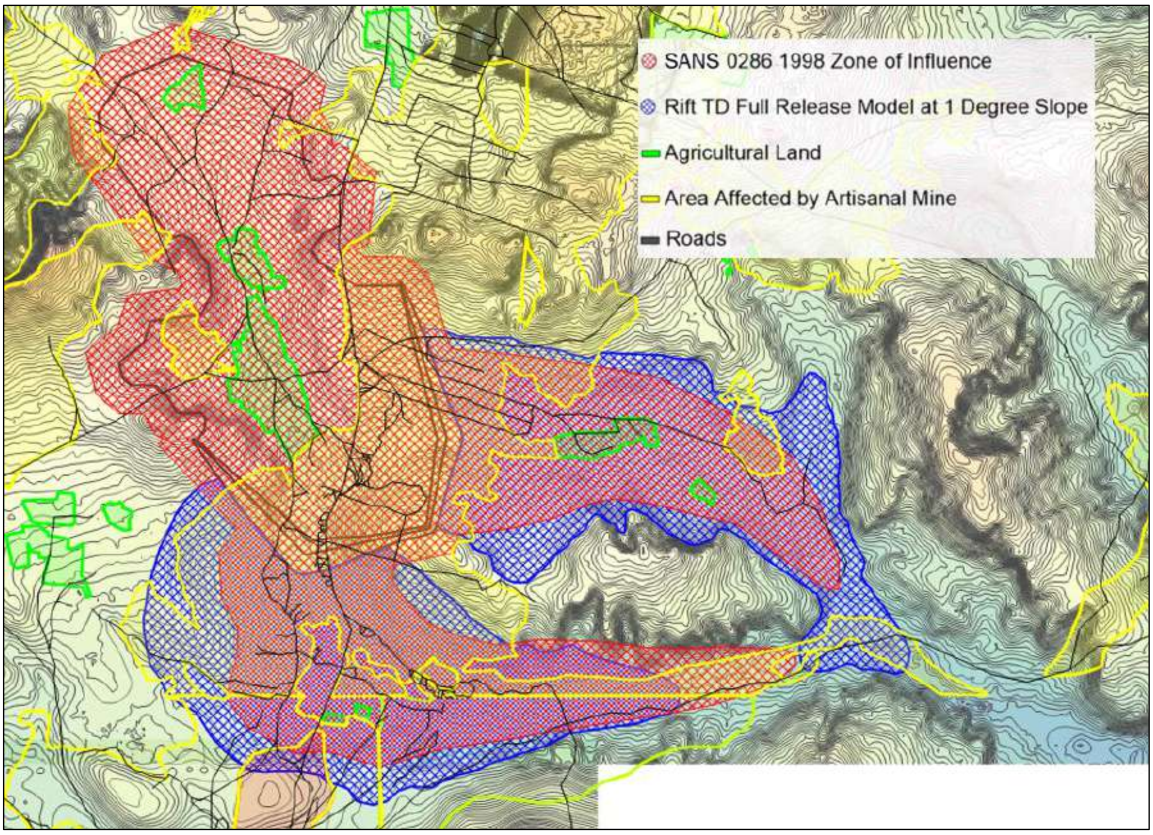


Figure 18.20: Kobada Zone of Influence and Full Release Model

The TSF classification is based on the understanding and undertaking by AGG that there will be relocation of the houses/private infrastructure and agricultural activities within the TSF zone of influence prior to the commencement of the TSF construction. Due to this, the population at risk is only temporary, with potential loss of life only including mine and construction workers downstream of the dam.

The discharged tailings shall flow into and along the main drainage lines, with the water from the tailings pond flowing along a similar path but to a greater extent. The likely flow path travelled by the water is shown in Figure 18.21, depicted by the blue lines. A detailed dam break analysis would need to be done to determine the extent of the TSF flood waters more accurately. A more extensive survey than that which is currently available is also required. Figure 18.22 shows the Google Earth image with the location of the Niger River indicated relative to the TSF.

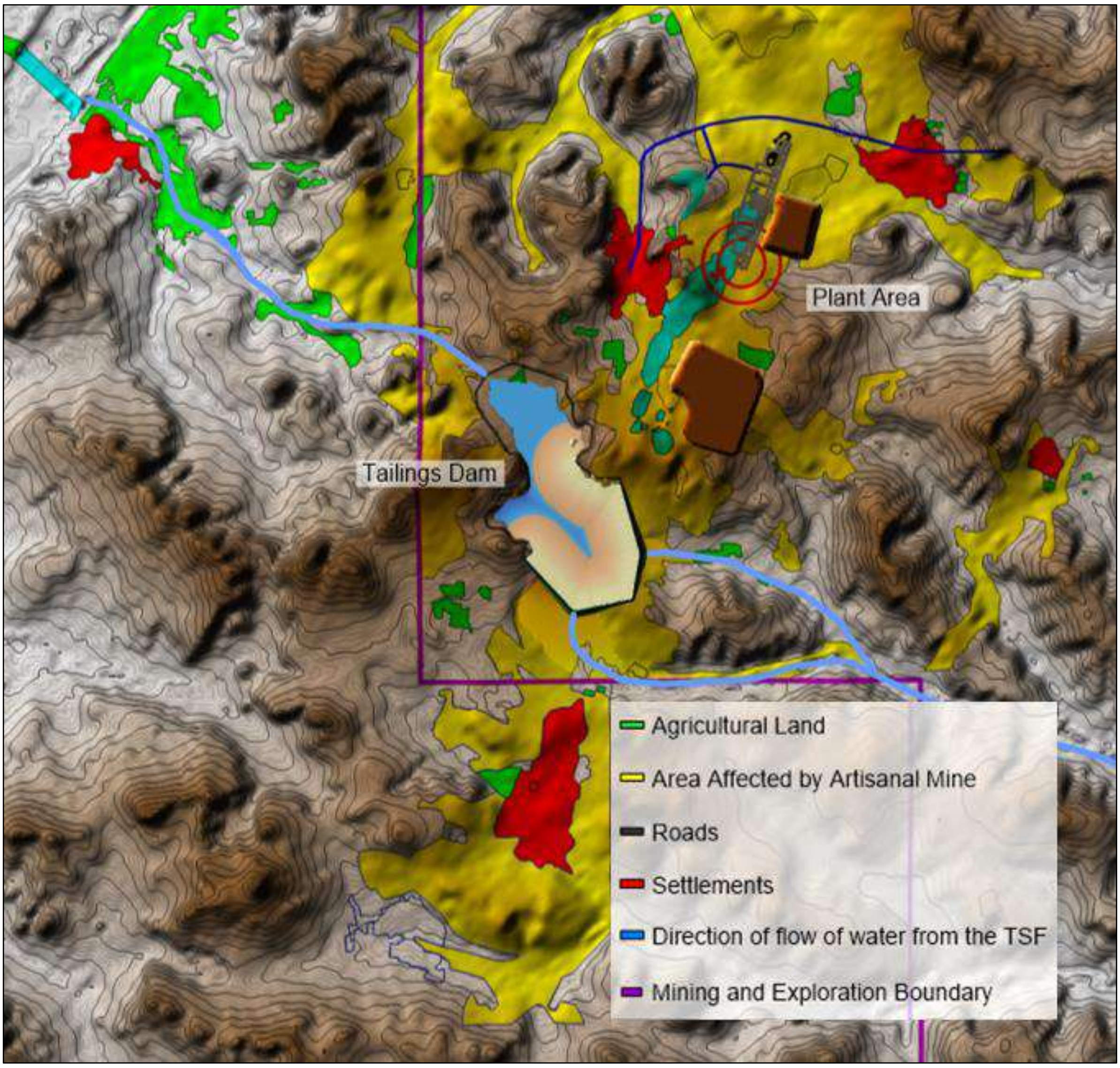


Figure 18.21: Valleys Surrounding the TSF and Possible Flow Path of Tailings Pond Water

Figure 18.22: Google Earth Image of the TSF relative to the Niger River

In the case of a breach in the northern side of the TSF, or in the unlikely event that an extreme flood occurs and is passed through the spillway, tailings water would flow towards the Niger River, located 6 km away from the TSF, as shown in Figure 18.22.

As stated in the CDA guidelines, a preliminary assessment using simple and conservative procedures should be done to obtain a first approximation of the level of consequences. Complexity and accuracy should be increased if there is a need for greater detail to confirm the dam classification. Specialist knowledge must be applied in the fields of ecosystems, land, water quality, fisheries, and cultural values. Because of the difficulty in predicting the environmental and ecosystem effects from accidental releases, it is often necessary to be on the conservative side when applying dam classifications.

A detailed dam break analysis is required to determine the extent of the tailings pond water travelled relative to the Niger River to aid in refining the classification of the dam according to the “environmental and cultural values” criterion. It is possible that the effect on terrestrial biodiversity and ecosystems may need to be assessed on an extended area according to the detailed dam break analysis and is recommended for the next stage of the study. As a result, the suggestion by the CDA guidelines to classify conservatively has been applied until the assessment is achieved with higher accuracy.

The effect on infrastructure and economics downstream of the TSF is considered to include losses to recreational facilities, seasonal workplaces, and infrequently used transportation routes.

Based on the assessment criteria outlined in the guidelines, the TSF is considered to be a Very High Hazard Facility, due to the potential impact of the release of the tailings pond water towards the Niger River. This classification is considered to err on the conservative side and is used as a bases to determine the TSF design loading and storm design criteria which are higher than compared to lower TSF Hazard Classification. The selection of the Very High Hazard option as opposed to High requires a more robust/conservative TSF design.

18.1.11.8.2 Design Aspects of the Tailings Storage Facility

The TSF has been designed taking cognisance of the following aspects:

- The topography, the immediate surroundings and mine infrastructure
- A total dry tailings storage capacity of 45.15 Mt at a deposition rate of 3 Mt/a over the LOM of 15.1 years
- Phased construction of the TSF over the LOM

Table 18.20 summarises the key parameters associated with the TSF. The TSF shall be constructed in five phases over the LOM. The construction of Phase 1 has been split across two years, forming Phase 1A in the first year of construction and Phase 1B in the second year of construction.

Table 18.20: Key Parameters Associated with the Kobada TSF

Tailings Dam Parameter	Phase 1A	Phase 1B	Phase 2	Phase 3	Phase 4	Phase 5
Total footprint area of the facility within the tailings surface and pond (ha)	160	200	260	320	350	370
Maximum design embankment wall elevation (mamsl)	388	390.5	396.5	401	403.5	406
Maximum design embankment wall height (m)	18	21	27	31	34	36
Outer side slope of embankment wall	1V:3H					
Inner side slope of embankment wall	1V:2H					
Embankment wall crest width (m)	8					
Embankment wall material	Waste material from mining operations (to be confirmed with geotechnical and geochemical test work)					
Years of tailings deposition	1.25	1.25	3	3.2	3.2	3.2
Cumulative years of tailings deposition	1.25	2.5	5.5	8.7	11.9	15.1
Tonnes of dry tailings stored in TSF (Mt)	3.6	3.7	8.2	10.5	9.6	9.5
Cumulative tonnes of dry tailings stored in TSF (Mt)	3.6	7.3	15.5	26.0	35.6	45.2

18.1.11.8.2.1 Stage Capacity Curves

The stage capacity curves show the development of the TSF over time and illustrate the relationship between tailings elevation, RoR, storage volume, footprint area, cumulative tonnage and time.

The compacted earth embankment of the TSF is constructed in phases over its operational life, reaching a final height of 36 m above Natural Ground Level (NGL), corresponding to an elevation of 406 mamsl. For each phase, tailings will be deposited behind the embankment wall until the maximum elevation is reached at the freeboard level below the crest elevation. By this time, the embankment wall of the subsequent phase will have been constructed and be ready to contain the deposited tailings. Table 18.21 summarises the tailings elevation, operational years, and dry tonnes of tailings per phase over the LOM.

Table 18.21: Staged Capacity of the TSF

Phase of Facility	Tailings Elevation (mamsl)	Cumulative Years	Cumulative Million Tonnes (Mt)
1A	386	1.25	3.6
1B	389.5	2.5	7.3
2	395.5	5.5	15.5
3	400	8.7	26.0
4	402.5	11.9	35.6
5	405	15.1	45.2

18.1.11.8.2.2 Preparatory Works

The preparatory works associated with the TSF comprise the following for Phase 1 of the TSF:

- Topsoil stripping to a depth of 0.3 m within the TSF footprint, including the embankment wall footprint area and associated TSF infrastructure area.
- A box cut over and above topsoil stripping to a depth of 0.5 m beneath the compacted earth embankment where the horizontal blanket drain system is absent.
- A compacted earth embankment with an 8 m wide crest, an outer slope of 1V:3H and an inner side slope of 1V:2H.
- A 3 m wide elevated toe drain located on an elevated platform at 381 mamsl, along a section of the embankment corresponding to the first three months of tailings deposition elevation. During this time period, the pool is pushed away from the drain, which comprises slotted 160ND HDPE pipe, suitably graded filter sand, and intermediate and coarse graded stone, all wrapped in non-woven geofabric. The drain serves to draw down the phreatic surface within the TSF.
- A 3 m wide toe drain, constructed where the elevated toe drain is absent, comprising suitably graded filter sand, intermediate and coarse graded stone, and 160ND slotted HDPE piping, all wrapped in non-woven geofabric. This serves to draw down the phreatic surface within the TSF.

- Five 3 m wide drains, constructed within the basin, parallel to the drainage line and towards the centre of the basin, comprising suitably graded filter sand, intermediate and coarse graded stone, and a 160ND slotted HDPE piping, all wrapped in non-woven geofabric. This serves to draw down the phreatic surface within the TSF.
- A 0.75 m wide vertical curtain drain within the Phase 1 main embankment, extending from the box cut to 6 m in height, comprising suitably graded filter sand, coarse graded stone, 160ND slotted HDPE piping, and non-woven geofabric. This serves to prevent the phreatic surface from migrating through the TSF embankment and exiting on the downstream side in the case of a liner leak.
- A 0.3 m thick horizontal blanket drain below the main embankment base in the low-lying areas comprising suitably graded filter sand, 160ND slotted HDPE piping encased in coarse graded stone, and non-woven geofabric. This serves to prevent any potential uplift pressures and sub-surface groundwater from entering the base of the TSF embankment.
- 160ND non-slotted HDPE pipes at specified intervals along the perimeter of the elevated toe drains, underdrains, vertical curtain drains and blanket drains, channelling the water collected by these drains into the solution pipeline
- A 355ND HDPE buried solution pipeline with 1 m backfill cover to channel the water from the various drain outlets to a water collection sump.
- Solution collection manholes spaced at intervals along the solution pipeline to collect seepage from the outlet pipes to be conveyed to the collection sump via the solution pipeline
- A seepage cut-off drain located ~100 m downstream of the Phase 1 embankment downstream toe. It is 1 m wide and 2 m deep and collects sub-surface seepage water. The drain comprises coarse drainage material, 19 mm stone, two 160ND slotted HDPE pipes, all wrapped in non-woven geofabric.
- A water collection sump for the collection of water from the solution pipeline and seepage cut-off manhole, from where it is pumped back onto the TSF and ultimately to the process plant for reuse.
- 1 m high catchment paddocks along the perimeter of the TSF.
- A 5 m wide gravel access road around the perimeter of the TSF area.
- A preliminarily sized, 2 m deep, 1V:1.5H side slope and 1.5 m wide base trapezoidal clean storm water diversion and associated cut to fill berm.
- A 400 OD HDPE, PE 100 PN 10 SDR 17 slurry distribution pipeline along the perimeter length of the TSF, with discharge outlets located at 36 m intervals.
- An earth fill barge access ramp.
- A preliminarily sized and positioned emergency spillway, 1V:2H side slopes and 8 m wide at the base and 0.3 m Reno mattress to prevent overtopping of the TSF embankment wall in the unlikely event that the pool size increases, resulting in emergency decanting off the TSF.

The preparatory works associated with Phases 2, 3, 4 and 5 of the TSF predominantly comprise the following:

- The downstream lifting of the earth compacted embankment wall with suitable open-pit overburden material.

- The construction of a 5 m high berm on the northern saddle of the TSF to provide freeboard to the operational pool and store floods under extreme events.
- A preliminarily sized, 2 m deep, 1V:1.5H side slope and 1.5 m wide base trapezoidal clean storm water diversion and associated cut to fill berm around the final TSF footprint.
- A 1 m high bund wall with an outer slope of 1V:3H and an inner side slope of 1V:2H along the final TSF perimeter.
- The augmentation or extension of the associated infrastructure where warranted, e.g. toe drains, solution trench, paddocks, and spillway

18.1.11.8.2.3 Tailings Dam Depositional and Operational Methodology

The depositional characteristics of any tailings product are based upon the physical characteristics of the material, the slurry density and the particle size distribution, which influence:

- Tailings behaviour upon deposition
- Beach formation and profile
- Particle segregation along the beach
- Pool control
- General tailings dam operations and deposition practices

The proposed depositional methodology for the TSF is by spigot/open-ended discharge behind a fully contained valley-type dam concept. This requires that each phase of the TSF embankment be built to its required height prior to commencing with that phase's associated deposition. During the initial commissioning stage of the Project, it remains crucial that the tailings not be deposited directly onto the various toe drains as this would lead to erosion and possible blinding of the toe drain system. Tailings should be deposited into the basin of the TSF by means of an open-ended deposition technique whereby flexible hosing, positioned at 36 m interval off-takes, is utilised. Prior to the tailings reaching the various toe drains, coarse tailings should be used to cover and further protect the drains. Open-ended deposition shall continue above the covered toe drains to the final elevation of each phase.

Surface water accumulating onto the TSF emanates from the following sources:

- Supernatant slurry water on the TSF
- Storm water runoff from the surface of the TSF

Supernatant water and storm water collected on the TSF shall be decanted by a floating barge arrangement and pumped back to the plant for reuse as process water. As the pond migrates up the valley, so too does the floating barge. Given that barge systems make use of electricity to pump slurry water back to the plant, it is often good practice to have standby pumps or a diesel generator to adequately cope with rapid ingress during an emergency. Furthermore, it is important that careful consideration be given to anchorage of the floating barge. The operational target limit for the pool volume is approximately 5 d of slurry water comprising roughly 70,000 m^3 of water or a sufficient pool depth (~1.5 m deep) to enable operating and management of the barge.

The development of the TSF and anticipated movement of the TSF pool over the LOM is shown in Figure 18.23 to Figure 18.27. The pool extent is a function of the required depth for the operation of the barge, the volume of water required by the plant and a safe distance from the containment walls. From a safety standpoint, the ideal pool extent to be maintained throughout the LOM has been depicted as 15 % of the tailings footprint as shown in the figures below.

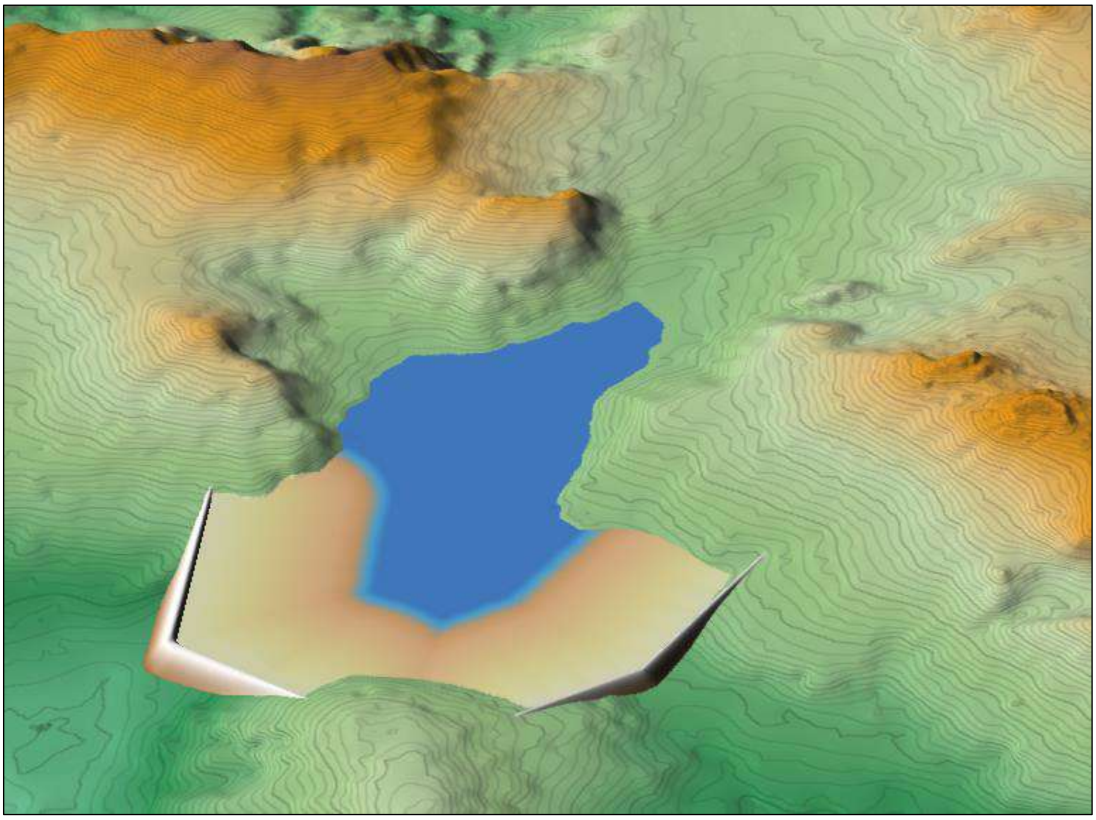


Figure 18.23: TSF Development at Phase 1 in Year 2.5 of the LOM

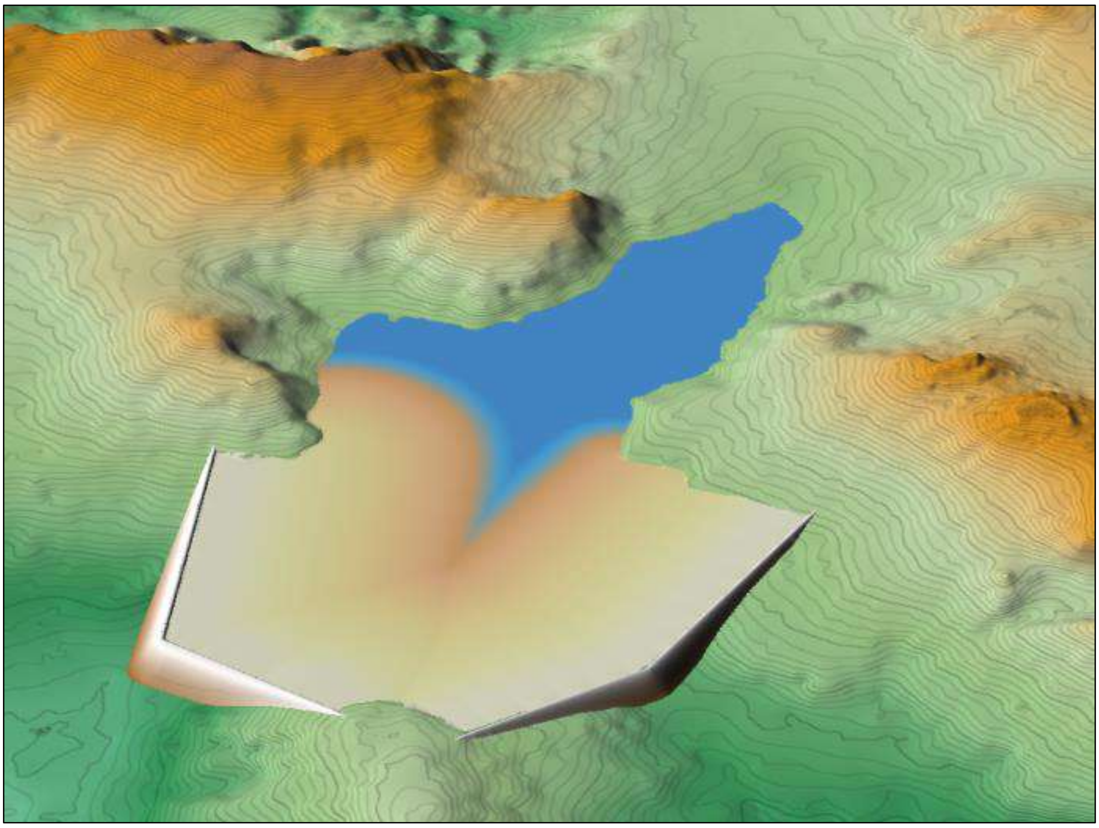


Figure 18.24: TSF Development at Phase 2 in Year 5.5 of the LOM

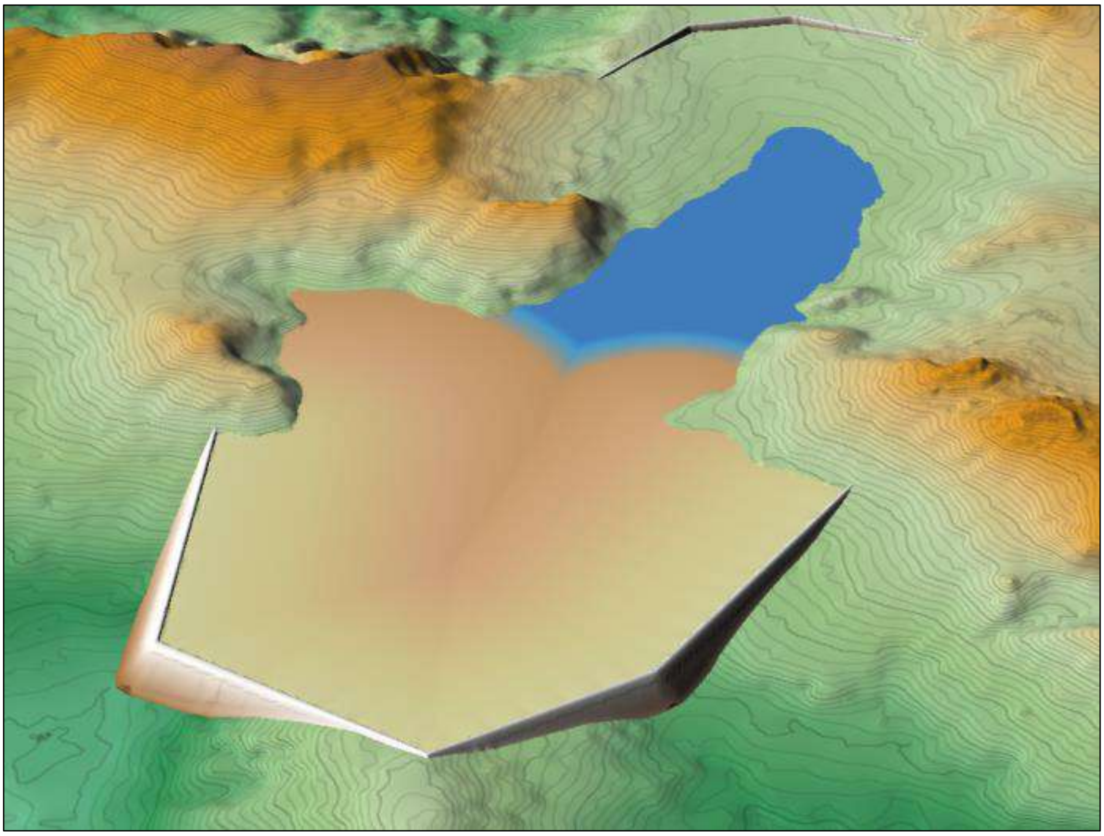


Figure 18.25: TSF Development at Phase 3 in Year 8.7 of the LOM

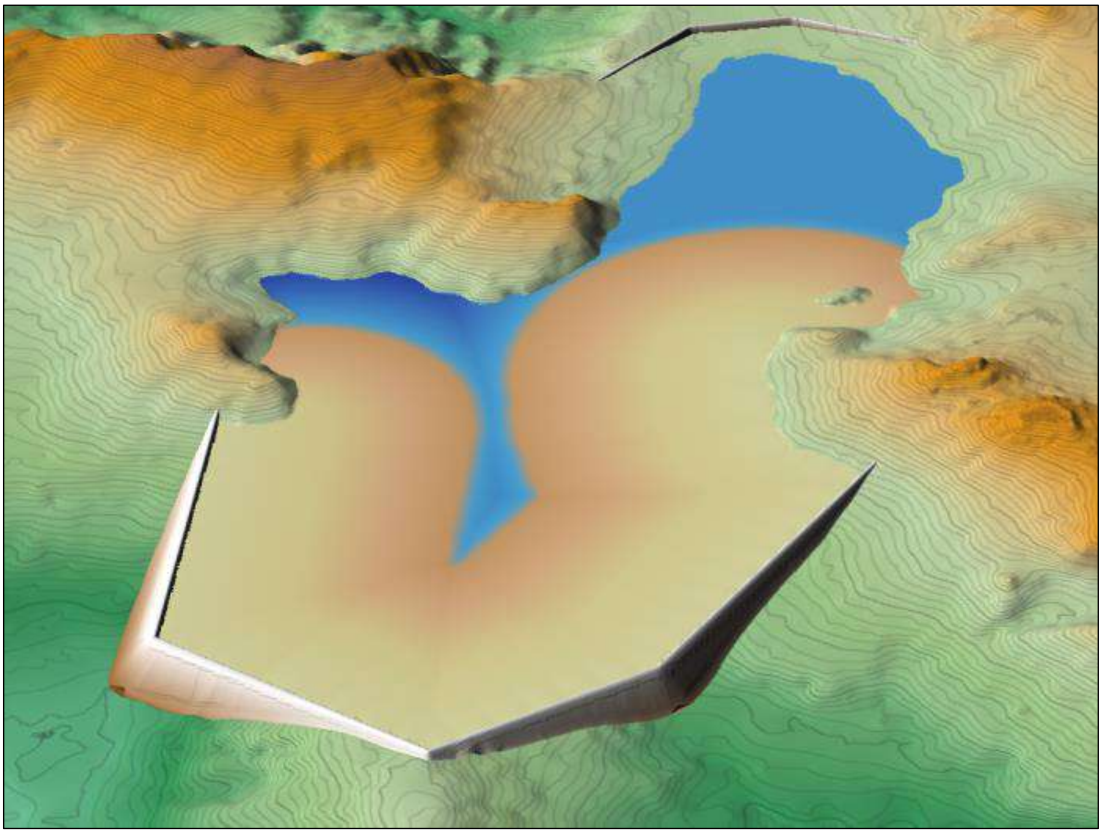


Figure 18.26: TSF Development at Phase 4 in Year 11.9 of the LOM

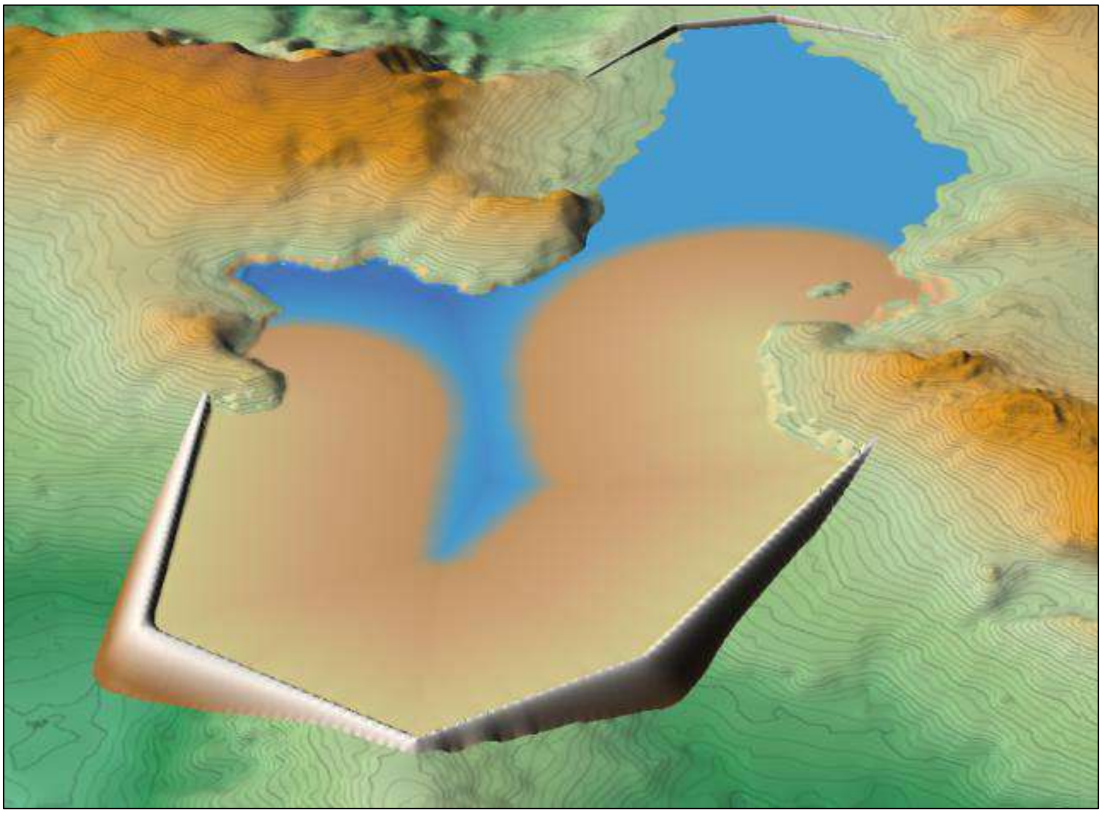


Figure 18.27: TSF Development at Phase 5 in Year 15.1 of the LOM

Seepage water collected from the various drains shall collect in a solution pipeline leading into a collection sump from where it shall be pumped back onto the TSF.

An emergency spillway has also been allowed for in each phase in the unlikely event that the pool size increases, resulting in emergency decanting off the TSF. Strict management and control of the location and size of the pool that develops over time and the freeboard during the later stages of the TSF development should, however, negate the occurrence of such an event.

18.1.11.8.2.4 TSF Pool Water Management Philosophy

As highlighted by the CDA guidelines, which are in line with the ICOLD *Tailings Dam Safety* draft document, there are several key functions of the management of water held on TSFs:

- Temporary storage of seasonal flows and sufficient water to allow settling of fines
- Temporary storage of the Environmental Design Flood (EDF)
- Storage and/or safe passage of the Inflow Design Flood (IDF) runoff to ensure the integrity of the containment dams

Figure 18.28 shows a section through a typical TSF extracted from the CDA guidelines and shows the defined levels of water stored and conveyed with an emergency spillway for passage of the IDF.

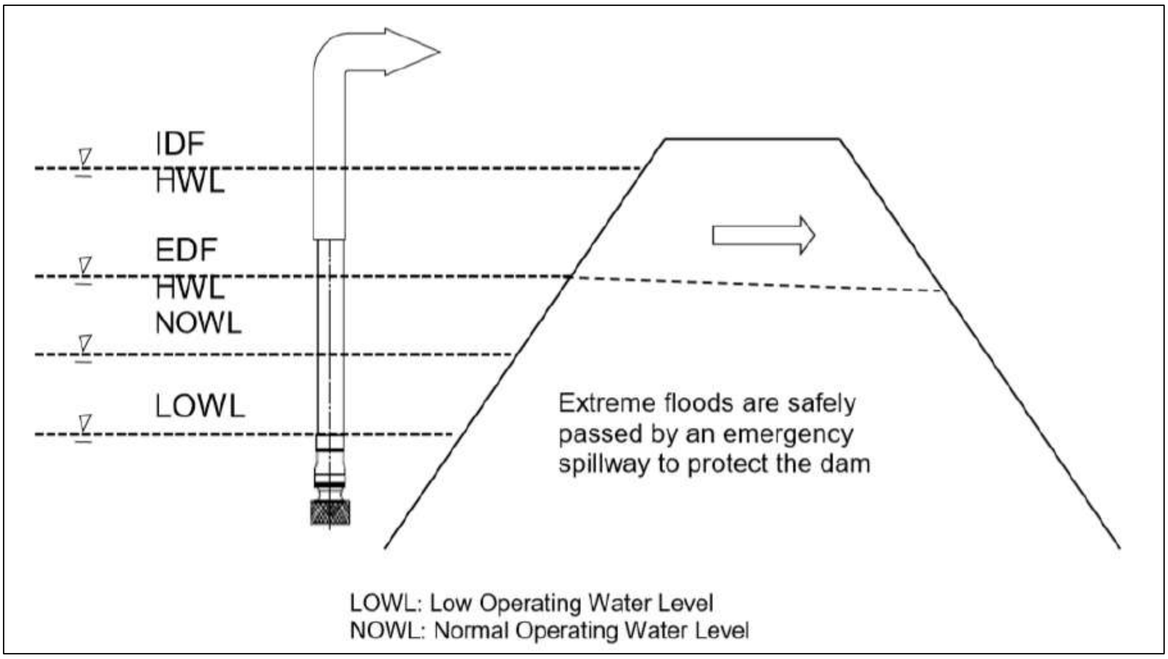


Figure 18.28: Typical Appurtenances Required for EDF Storage and IDF Conveyance

The CDA guidelines define the EDF as “the most severe flood that is to be managed without release of untreated water to the environment.” The EDF is, therefore, the retention of contaminated water without release to the environment or controlled release through a treatment system. Storage capacity of the EDF is required above the normal operating water level (NOWL) during the period when the EDF is retained. The TSF has been sized to store the EDF of the 1 in 100, 7 d storm event of 530 mm.

According to the CDA guidelines, the IDF is “the most severe inflow flood for which a dam and its associated facilities are designed.”

Table 18.22 presents recommendations for target levels for the IDF. These are considered applicable for the construction, operation, and transition phases. The IDF for the Very High consequence class dam is derived by interpolating 2:3 between the 1:1 000-year flood and the (Probable Maximum Flood (PMF). Based on a 24 h duration of the 1:1 000-year flood of 765 mm and a 24 h duration of the PMP of 1 703 mm provided by Peens & Associates (2020), the IDF depth was calculated to be 1 390 mm.

Table 18.22: Target Levels for Flood Hazards and Standards-Based Assessments for Construction, Operation, and Transition Phases

Consequence Class	IDF
Low	1:100 year
Significant	Between 1:100 and 1:1 000 year[a]
High	1/3 between 1:1 000 year and PMF[b]
Very High	2/3 between 1:1 000 year and PMF[b]
Extreme	PMF

[a] Selected on the basis of incremental flood analysis, exposure, and consequence of failure

[b] Extrapolation of flood statistics beyond 1:1 000-year flood (10-3 annual exceedance probability (AEP)) is generally discouraged. The PMF has no associated AEP. The flood defined as “1:3 between 1:1 000 year and PMF” or “2:3 between 1:1,000 year and PMF” has no defined AEP.

The emergency spillway passes the IDF from the TSF pool to draw down the TSF to at least the EDF level following the storm event. Management of the TSF pool to maintain levels between the low operating water level (LOWL) and the NOWL is required in cases where the TSF can be subject to seasonal and environmental constraints that result in retention of water in the TSF.

The following freeboard guidelines, as outlined in the CDA guidelines, apply to most TSFs. For an embankment structure, the crest level should be set so that the structure is protected against the most critical of the following cases:

- No overtopping by 95 % of the waves caused by the most critical wind with a 1:1 000-year frequency when the reservoir is at its maximum normal elevation
- No overtopping by 95 % of the waves caused by the most critical wind when the reservoir is at its maximum extreme level during the passage of the IDF

The consequence class of the TSF is used to determine the most critical wind. Suggested annual exceedance probability (AEP) values of wind frequency used to calculate the freeboard during the IDF are as follows:

- Low consequence dam: AEP = 1:100
- Significant consequence dam: AEP = 1:10
- High, very high, or extreme consequence dam: AEP = 1:2

The wind rose for Kobada is shown in Figure 18.29, with an average wind speed of 10.5 km/h. The maximum recorded wind speed of 32 km/h was used in the freeboard calculations in the absence of data for the 1 in 1 000-year wind event. The wind rose was provided by SENET in the General Site Conditions and Design Criteria Report (2020).

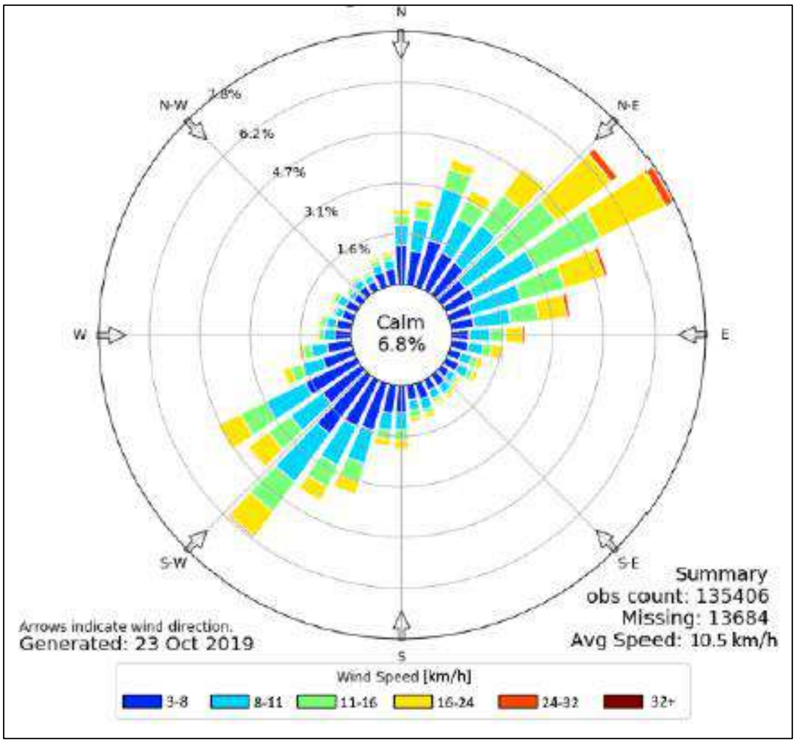


Figure 18.29: Kobada Mine Wind Rose

The LOWL and NOWL were estimated to be a 1.5 m deep pool sufficient for barge operation and the seasonal rainfall of ~1.2 Mm^3 based on the water balance discussed in Section 18.1.11.9, respectively.

Figure 18.30 shows the pool extents for the LOWL, NOWL, EDF and IDF to determine the minimum required freeboard for the TSF at final height. The boundary of the IDF pool is depicted in red, the EDF in yellow, the NOWL in green, and the LOWL in blue. The freeboard for the TSF meets the minimum criteria of 1 m specified by the CDA guidelines and the ICOLD *Tailings Dam Safety* draft document for each phase of construction.

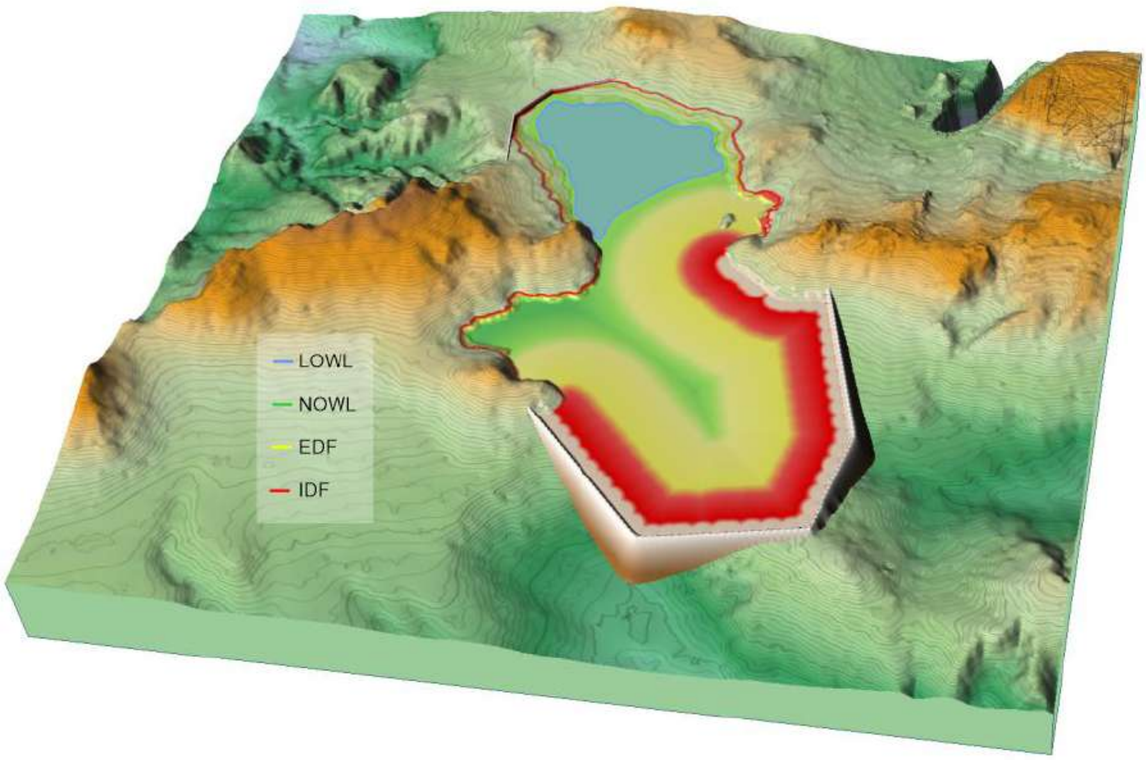


Figure 18.30: Design Water Levels and Floods for Freeboard Determination

Flood operating rules should be observed according to the CDA guidelines. The minimum freeboard at the dam should only be reached under extreme conditions where the pool of the facility reaches the level of the IDF. Regardless of the magnitude of the occurring flood or what was forecast, the resulting facility level after the passage of the flood should be in the range of levels normally observed for that time of the year. Operating rules should be available for all floods up to the IDF and should be well understood by the operating staff.

Storm water runoff outside of the TSF footprint area is classified as clean water and is diverted away from the TSF via storm water diversion trenches along the southern face of the TSF. Runoff water emanating from the embankment wall side is collected in the catchment paddocks, located along the toe of the TSF starter wall embankment. The captured water is drained into the solution pipeline or evaporated over time.

18.1.11.9 Water Balance

A TSF deterministic monthly water balance has been developed in EXCEL, based on normal year monthly, wettest year monthly, driest year monthly rainfall and evaporation figures, and simulates the flow of water between the TSF and plant over the LOM.

18.1.11.9.1 Water Balance Objectives, Constraints and Criteria

The objectives of the water balance are to determine

- The flow philosophy and management of water across the TSF and mine
- The volume of surplus or shortfall of water arising within the TSF water balance and thus the volume of make-up water required by the process plant, or water discharged (treated) into the receiving environment

The following key constraints were applied to the water balance:

- The process plant is able to receive a maximum of 80 % of the water volume contained in the tailings slurry as process plant dirty water make-up
- The TSF water balance design criteria as outlined in Table 18.23.

Table 18.23: Modelling Criteria Associated with the Kobada TSF Water Balance

Item	Parameter	Value	Source/Comment
1	LOM	15.1 years	SENET
2	Particle specific gravity of tailings product	2.75	Laboratory Test Work/Epoch
3	Placement dry density of tailings	1.25 t/m^3	Epoch – calculated from assumed void ratio and specific gravity
4	Slurry Density	1.29 t/m^3 (35 % solids dry mass)	SENET
5	Interstitial water lock-up within deposited tailings	44 % by dry mass	Epoch Experience
6	Seepage losses into underlying founding material	0 % as facilities are HDPE lined	SENET/Epoch
7	Normal operational pool size	15 % to 20 % of the total depositional area and 1.5 m to 2.0 m depth for a barge operation	Epoch Experience

The overall TSF water balance comprises the following key aspects:

- An HDPE-lined TSF for the storage of the tailings
- All the storm water arising on the TSF basin shall be held on the TSF and reused in the process plant. Discharging into the environment via the TSF is to be avoided, unless under extreme storm rainfall events.
- Any shortfall in the plant process make-up water, not sourced from the TSF, shall be sourced from other mine sources, e.g. open-pit dewatering and river sources.

18.1.11.9.2 Models and Modelling Philosophy

The monthly Excel mine-wide water balances developed for Kobada covered the following scenarios:

- Normal year monthly climatic conditions and steady-state modelling parameters, referred to as a “Normal Monthly Water Balance”

- Wettest year monthly rainfall figures, wettest year monthly evaporation figures and steady-state modelling parameters, referred to as the “Wettest Monthly Water Balance”
- Driest year monthly evaporation figures and steady-state modelling parameters, referred to as the “Driest Monthly Water Balance”.

18.1.11.9.3 Overall Water Balance Modelling Philosophy

The Kobada mine-wide water balance is presented in Figure 18.31 and summarised as follows:

- A process plant from which tailings are pumped in slurry form to an HDPE-lined TSF.
- Slurry supernatant and storm water from the TSF, which is
  - Pumped to the process plant as make-up water
  - Stored on the TSF for utilisation in the subsequent month(s)
  - Discharged into the downstream environment if the storage capacity is exceeded
- Water from other sources, such as the open pit, are used as process plant make-up water sources when warranted, otherwise discharged into the downstream environment.

The pertinent modelling parameters/assumptions associated with the Kobada TSF water balances are summarised in Table 18.24.

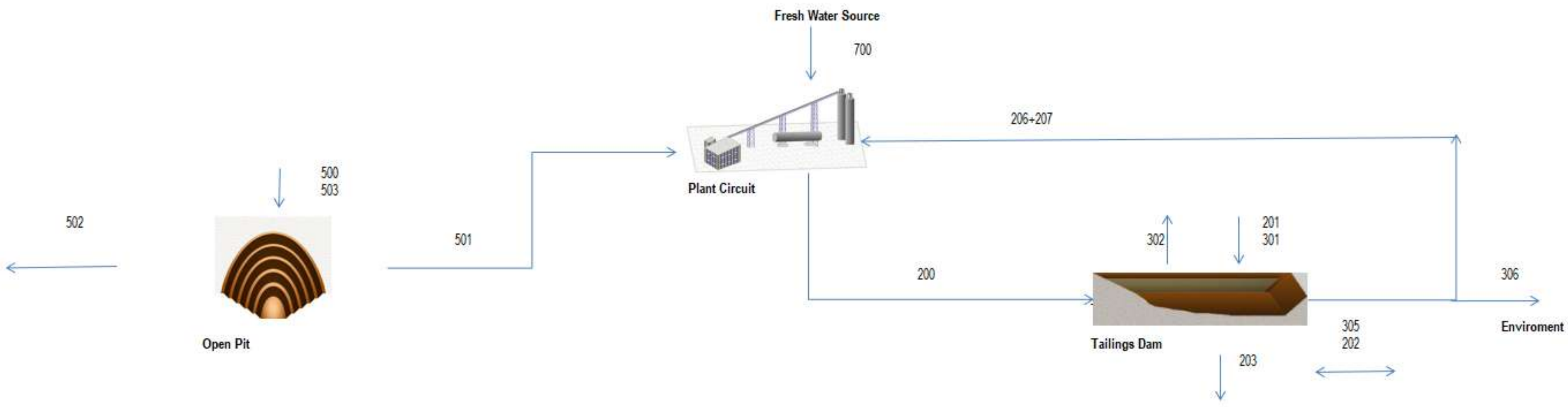


Figure 18.31: Kobada Mine-Wide Water Balance Circuit

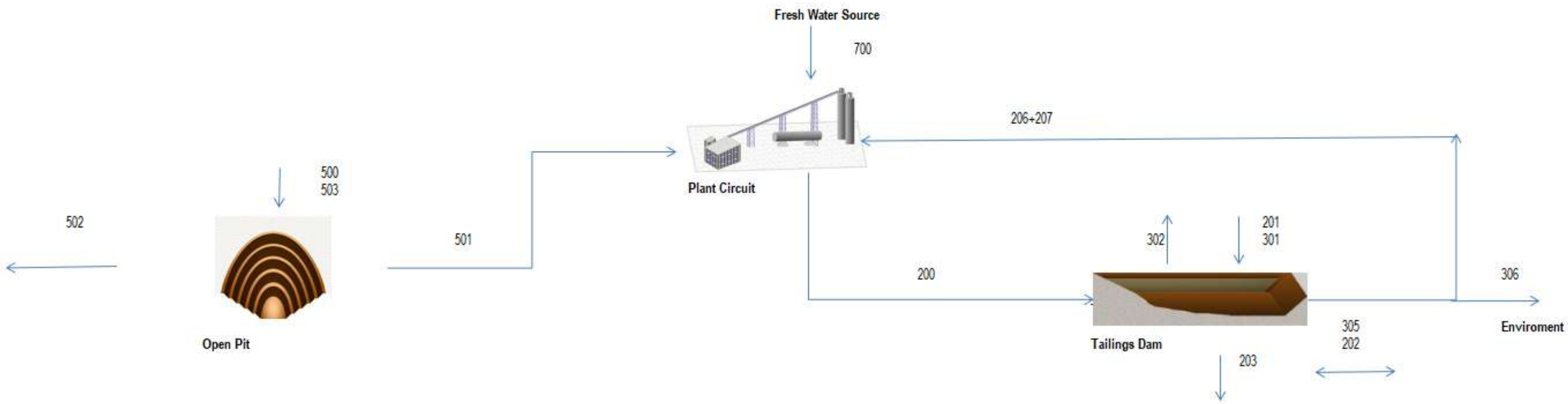
Table 18.24: Kobada Mine-Wide Water Balance Modelling Parameters/Assumptions

Item No.	Description/Parameter	Value/Comments
1	Surface area of TSF	360 ha
2	Pool size	Unrestricted, not allowed to discharge from TSF
3	Maximum process plant reuse volume	80 % of tailings slurry water content
4	Clean process plant water requirement	20 % of tailings slurry water content
5	Interstitial lock-up water content (mass water/dry mass tails)	44 % of tailings dry mass
6	Runoff coefficient applied to water bodies and lined surfaces	1.0
7	Runoff coefficients per month applied to unlined surfaces:	
	January	0
	February	0
	March	0
	April	0
	May	0.3
	June	0.5
	July	0.65
	August	0.75
	September	0.75
	October	0.40
	November	0
	December	0

18.1.11.9.4 Water Balance Results

The overall outputs for the various water balances are presented in Figure 18.32. The key outcomes associated with the Kobada mine-wide water balance are as follows:

- The mine wide water balance is positive, i.e., a surplus of water is generated within the mine wide water circuit, requiring discharge. Surplus water from the open-pit dewatering boreholes and open-pits themselves, due to storm water accumulation during the wet season, needs to be discharged into the downstream environment when not consumed by the process plant. This occurs intermittently from Year 1 to Year 3 and all year round from Year 3 to end of LOM.
- The process plant can accept 80% of the TSF slurry water volume sent to the TSF back, with the remaining 20% being sourced from clean water sources. As such, the TSF meet the 80% process plant water demand from May to October. For the remaining months additional water, above the 20% clean water sources, needs to be augmented to the TSF water from the open pits dewatering boreholes, etc.
- The TSF pool size reaches a maximum storage volume of 1,35 million m^3 in the month of August based on the wettest year rainfall water balance simulations.



Water Balance		Float Slurry Water	TSF Area Runoff	Interstitial Lock Up	Water Return to Plant - Stage 01	Rainfall on TSF Pool	Evaporation from TSF Pool	Seepage from TSF Pool	Water Return to Plant from Pool - Stage 02	Actual Water Stored on TSF Pool after Stage 02	Discharge from TSF	Return to Plant from Pits	Plant Shortfall after Stage 05	Open Pit Discharge	Open Pit Ground Inflows	Open Pit Rainfall Inflows
	Stream No	200	201	202	206	301	302	303	304	305	306	501	700	502	503	500
Average	Maximum	463,415	1,129,913	109,091	268,237	153,972	102,024	-	370,732	1,215,997	-	107,302	92,714	624,388	233,783	391,494
Average	Minimum	463,415	-	109,091	-	-	51,480	-	102,495	-	-	-	-	-	13,420	-
Average	Average	463,415	311,382	109,091	16,863	48,867	73,632	-	313,598	310,481	-	37,322	2,950	230,853	175,337	92,837
Wet	Maximum	463,415	1,253,551	109,091	247,874	170,820	102,024	-	370,732	1,356,483	-	107,302	92,714	667,129	233,783	434,332
Wet	Minimum	463,415	-	109,091	-	-	51,480	-	122,857	-	-	-	-	-	13,420	-
Wet	Average	463,415	362,785	109,091	15,710	56,979	73,632	-	315,636	369,110	-	36,367	3,019	246,940	175,337	107,970
Dry	Maximum	463,415	1,013,144	109,091	151,467	138,060	102,024	-	370,732	1,083,316	-	118,432	105,012	578,081	352,283	531,785
Dry	Minimum	463,415	-	109,091	-	-	51,480	-	219,264	-	-	-	-	-	52,870	-
Dry	Average	463,415	242,258	109,091	7,257	37,479	73,632	-	313,504	239,668	-	45,716	4,255	202,793	283,740	113,904

Figure 18.32: Kobada Mine-Wide Water Balance Circuit and Outputs

18.1.11.10 Seepage Analysis

A steady-state seepage analysis of the TSF has been undertaken, utilising the finite element program, SEEP/W (GeoStudio 2019) to determine the following:

- The seepage regime, as well as the pore water pressure distribution within and below the facility at the end of the various phases of its development over the LOM
- The effectiveness of the drain systems in managing the phreatic surface within the facility
- The slope stability assessments

The TSF has been modelled under steady-state conditions corresponding to the following stages:

- Final elevation of Phase 1
- Final elevation of Phase 2
- Final elevation of Phase 3
- Final elevation of Phase 4
- Final elevation of Phase 5

The scenarios modelled consider operational conditions as per design, as well as unfavourable conditions (drain failure, an extreme storm event, etc.). The critical section analysed for the seepage assessment is shown in Figure 18.33.

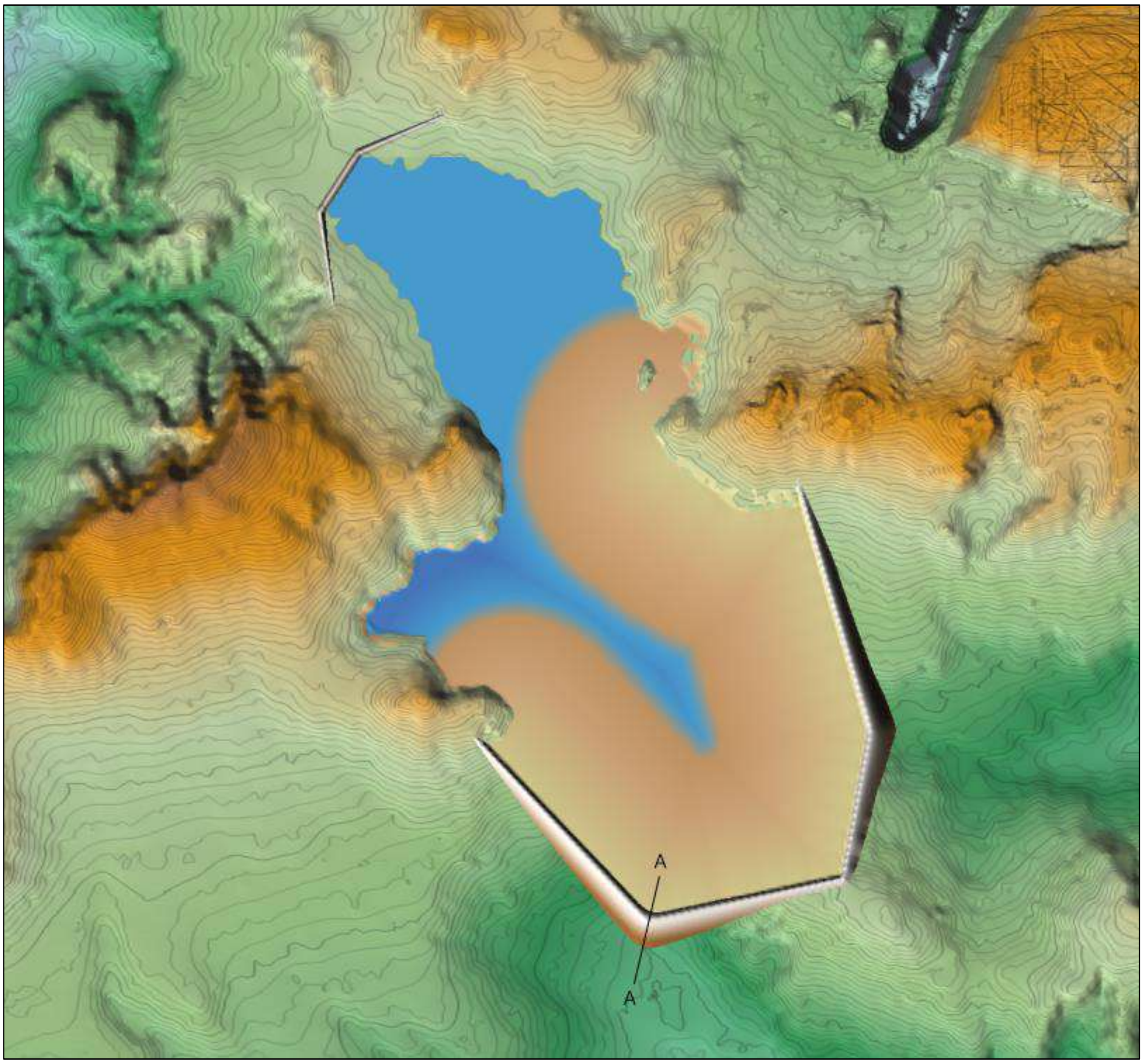


Figure 18.33: Critical Section through the TSF Wall for Seepage Analysis

The following three seepage scenarios were analysed:

- Operational drains, liner and pool as per design
- Non-operational liner with operational pool
- Non-operational drains with operational pool

The pool extent under extreme conditions remains at a safe distance of 350 m away from the embankment wall for Phases 2 and onwards of construction. During Phase 1, however, the tailings do not yet extend far enough to push the pool away from the wall during extreme flood events. Therefore, the additional scenario included for analysis of Phases 1 is for a pool extent up to crest level with operational drains and liner.

The geometry of the TSF upon which the analyses were conducted is shown in Figure 18.34.

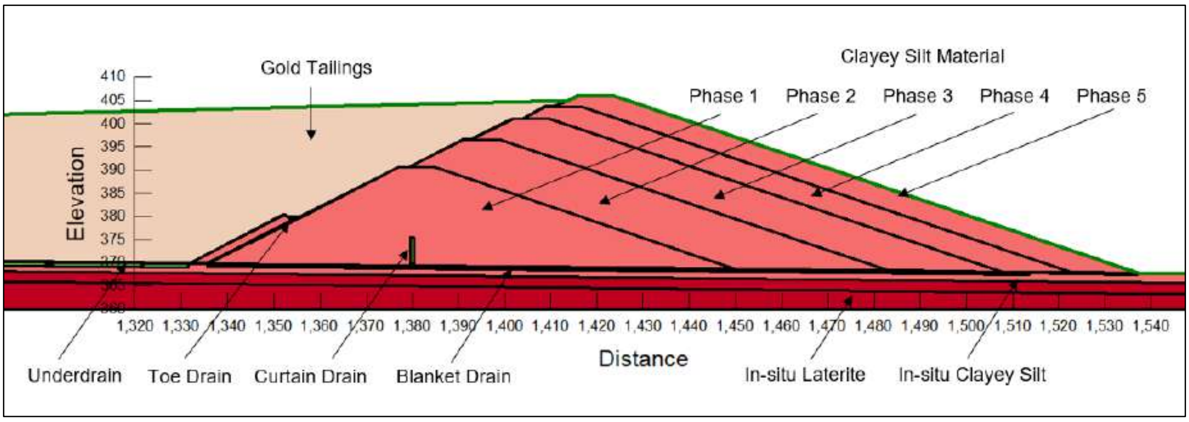


Figure 18.34: Tailings Dam Geometry

The material parameters used in the seepage analysis are summarised in Table 18.25. The Kx:Ky ratio indicates the hydraulic conductivity of the material in the horizontal direction to the vertical direction.

Table 18.25: Material Parameters Adopted for the Seepage Analysis

Material	Kx:Ky Ratio	Saturated Permeability (m/s)
Gold Tailings	1	3×10^{-8}
Clayey Silt with Laterite Gravels (TSF Embankment)	1	1×10^{-6}
In-Situ Laterite	1	1×10^{-8}
In-Situ Clayey Silt with Laterite Gravels	1	1×10^{-6}

Figure 18.35 shows the seepage regime for the TSF.

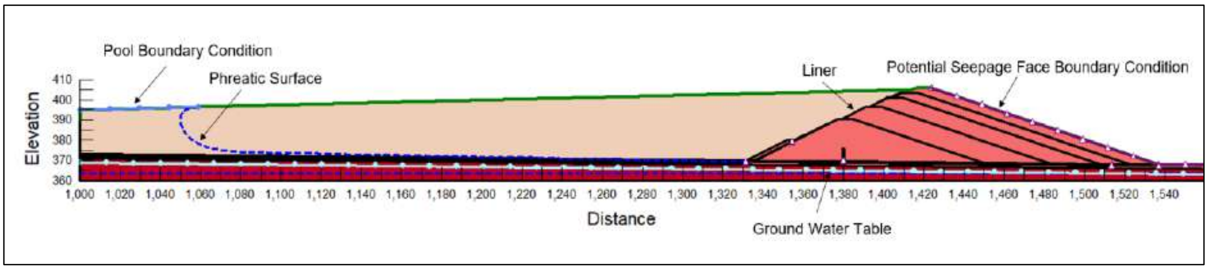


Figure 18.35: Steady-State TSF Seepage Regime with Functional Drains and Liner

The results of the seepage analysis indicate that

- Operational drains and an operational liner have a significant influence on the position of the phreatic surface.

- Non-operational scenarios indicate that, due to redundancy in the design, the seepage regime does not show variation between the non-operational blanket drain, curtain drain, toe drain and liner.
- The highest phreatic surface is indicated for the non-operational underdrainage system and toe drain scenario for each phase, and extreme flood event scenario for Phase 1.
- The extent of the pool influences the phreatic surface. A larger pool increases the phreatic surface, as indicated for the extreme flood event scenario.

18.1.11.11 Slope Analyses

Slope stability analyses have been carried out on a variety of possible operational and upset conditions.

18.1.11.11.1 Methodology

Both deterministic and probabilistic slope stability analyses, using Slope/W (GeoStudio 2019), were undertaken on the following scenarios for the TSF:

- Final elevation of Phase 1
- Final elevation of Phase 2
- Final elevation of Phase 3
- Final elevation of Phase 4
- Final elevation of Phase 5

The above scenarios were assessed incorporating the phreatic surfaces determined by the seepage analyses described in Section 18.1.11.10.

The deterministic analysis calculates the factor of safety (FoS) of a slope, based on fixed material parameters and specified conditions. The application of the deterministic FoS approach on its own fails to recognise the level of variability associated with the input parameters used in a slope stability analysis and, therefore, the potential variability in the FoS. This variability is assessed in terms of the probability of failure index and reliability index. The probability of failure index is computed by relating the number of trials that fall below an FoS of 1.0 (failure) and expressing that number as a percentage of the number of Monte Carlo trials that were completed. The reliability index describes the stability of a slope by the number of standard deviations separating the mean FoS from the defined FoS failure value of 1.0, i.e. the reliability index is an indication of how far away the mean FoS is from the defined failure value of 1.

The minimum FoS requirements have been summarised in Table 18.26 based on the CDA guidelines. The acceptable probability of failure and reliability index values have been defined as < 0.0007 % (< 1:143 000) and > 4.35, respectively (U.S Army Corps of Engineers, 1997, (EN 1990:2002, 2005) and (Cole, 1993).

Table 18.26: Minimum FoS Requirements

Loading Condition	Required Minimum FoS	Slope
During or at the end of construction	1.3 depending on the risk assessment during construction	Typically downstream
Long term (steady-state seepage, normal reservoir level)	1.5	Downstream
Pseudo static	1.0	
Post-earthquake	1.2	

18.1.11.11.2 Static Slope Stability Analysis

Soil parameters shown in Table 18.27 have been selected based on test work from the geotechnical site investigation. These have been updated from the previous DFS, which was based on assumptions due to the lack of test work results available at the time. The updated parameters are similar to the parameters estimated in the previous DFS.

The waste material mined from the pit and used for the various embankment phases is assumed to have similar properties as the saprolite soil from the basin, which shall be used for the construction of the TSF. It is recommended that this assumption be confirmed with future geotechnical test work.

The oxide tailings parameters were more conservative than that of the sulphide tailings and were therefore used for the models. The critical slip circles, however, occur within the confines of the main embankment, thus negating the influence of the tailings geotechnical parameters and the phreatic surface within them.

Table 18.27: Material Parameters Adopted for the TSF Slope Stability Analyses

Material Description	Unit Weight (kN/m^3)	Effective Friction Angle (φ’) (degrees)	Effective Cohesion (c’) (kPa)
Gold Tailings (not within Critical Slip Circle)	13.7	25	0
Clay Silt with Laterite Gravels	20 (±5)	27 (±5)	0 (−3; +2)
In-Situ Laterite	22.0 (±5)	31 (±5)	20 (−4; +2)

The slope stability assessment presented was conducted on the cross section and scenarios presented in the seepage analysis and represents the critical section for embankment stability.

18.1.11.11.3 Pseudo-Static Slope Stability Analysis

Earthquakes impose additional loads on tailings dams and embankment dams over and above those experienced under static conditions. The earthquake loading is of short duration, cyclic and involves motion in the horizontal and vertical directions. To assess whether a dam can safely absorb these additional earthquake-induced loads, a pseudo-static analysis is performed.

The approach involves a conventional limit equilibrium stability analysis and incorporates a horizontal acceleration to represent the effects of the earthquake loading. The vertical force is usually ignored in a pseudo-static analysis since most earthquakes produce a peak vertical acceleration less than the peak horizontal acceleration, and the vertical pseudo-static force usually has much less of an effect on the stability of a slope. The horizontal inertia force is expressed as a product of a seismic coefficient (k) and the weight of the sliding mass.

The seismic coefficient selected for such analyses is based on the hazard classification of the dam. An annual probability of exceedance of 1:5 000 for a Very High dam consequence category as outlined in Table 18.28 from the ICOLD Guidelines has been adopted for the current analysis and TSF design.

Table 18.28: Recommended Annual Exceedance Probability for Earthquakes

Dam Classification	Annual Exceedance Probability – Earthquakes[a]
Low	1:100
Significant	1:1 000
High	1:2 475
Very High	1:5 000
Extreme	1:10 000 or Maximum Credible Earthquake (MCE)[b]

[a] The criteria presented are a guidance for the suggested minimum criteria. The Accountable Executive Officer (AEO) and the Engineer of Record (EOR) should be used to assess each facility for the potential to increase the design criteria as far as reasonably practical.

[b] The selection of the probabilistic or deterministic design earthquake should consider the seismic setting and the reliability and applicability of each method.

As no specific probabilistic seismic study was carried out for Kobada, the probabilistic seismic hazard analysis conducted for an existing mine site situated in south-western Mali was used. The PGA is the maximum acceleration of the ground shaking during an earthquake. The PGA is converted into a seismic coefficient that is used in pseudo-static limit-equilibrium slope stability models and accounts for the force exerted by an earthquake onto the model. For a 1:5 000-year return period, a PGA of 0.06 g was adopted for the pseudo-static limit-equilibrium slope stability analysis.

The Modified Mercalli Scale Earthquake Intensity Map of Africa from the Office for the Coordination of Humanitarian Affairs (OCHA) Regional Office for Central and East Africa, as shown in Figure 18.36, has been used to compare the degree of the earthquake intensity. As indicated, Kobada is not located in a seismically active zone; therefore, a severe earthquake is not anticipated, and the seismic coefficient is expected to be below 0.1, according to the Geotechnical Earthquake Engineering Handbook (Robert W. Day, 2002).

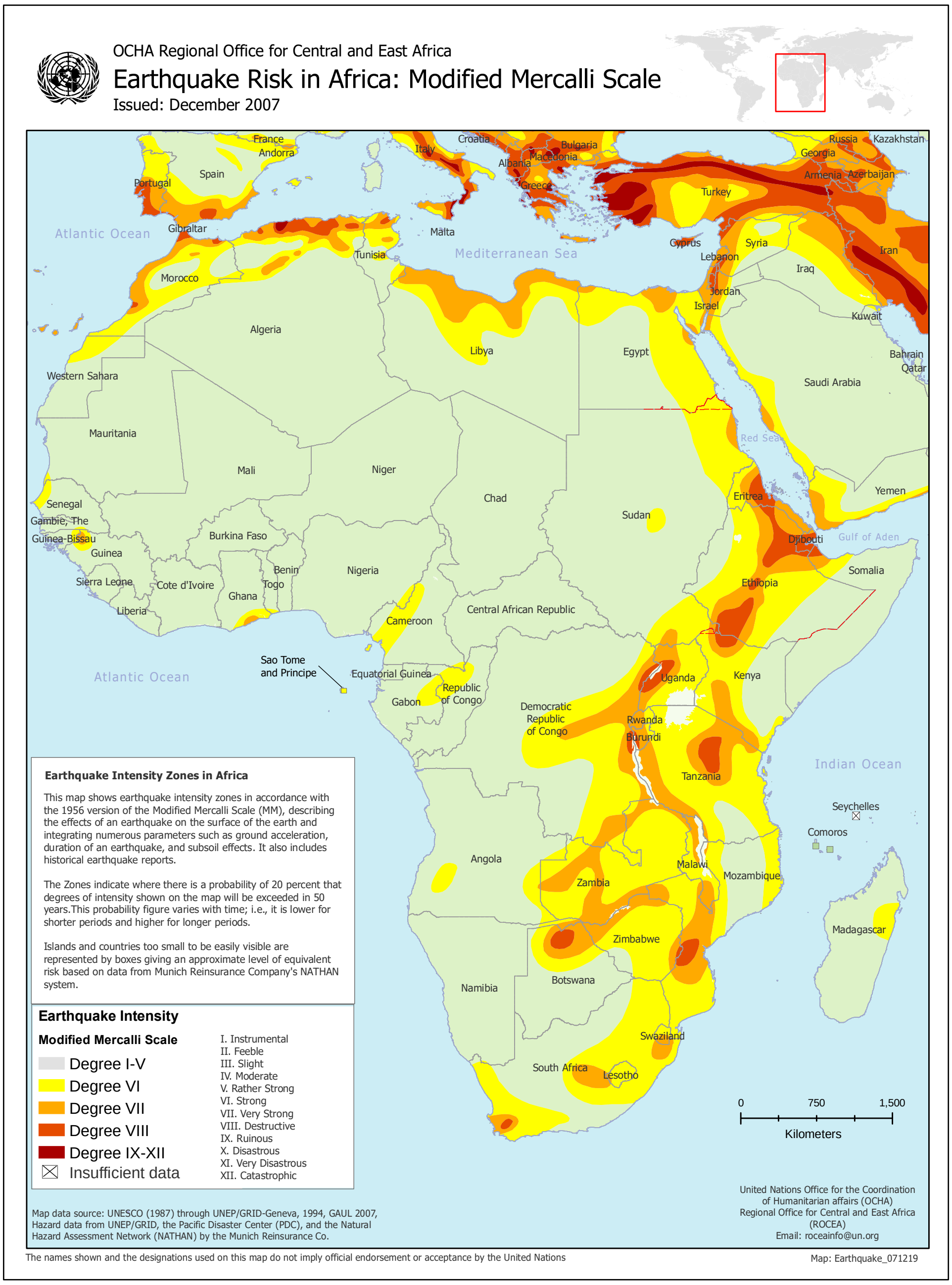


Figure 18.36: Modified Mercalli Scale Earthquake Intensity Map of Africa

18.1.11.11.4 Slope Stability Analyses Outcomes

The locations of the critical slip circles for the TSF are shown in Figure 18.37 to Figure 18.39. Table 18.29 summarises the results obtained for the various static slope stability assessments.

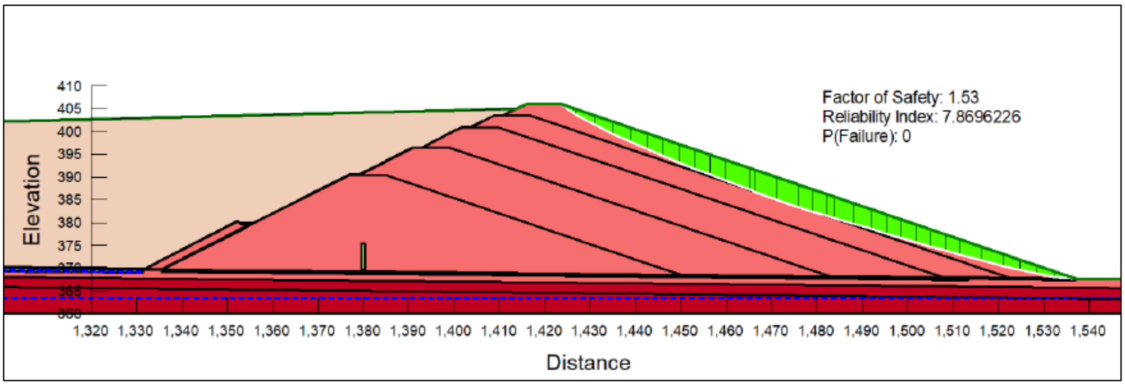


Figure 18.37: Slope Failure Through Operational Final Phase 5 (as per Design)

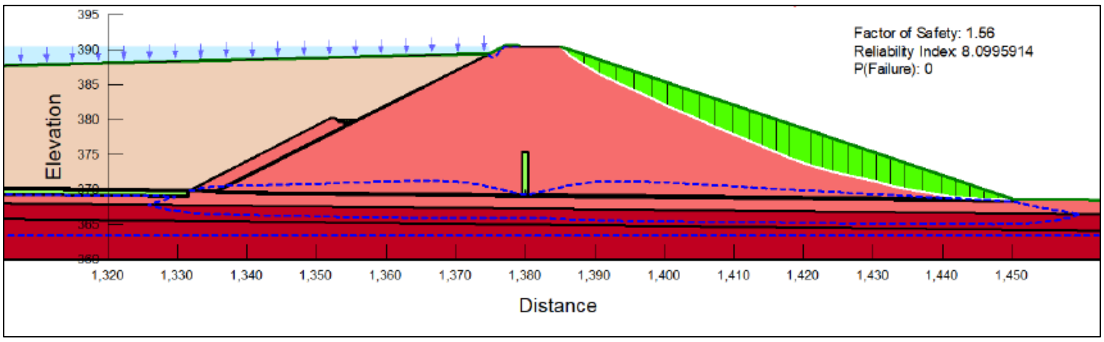


Figure 18.38: Slope Failure Through Phase 1B with Extreme Flood Event

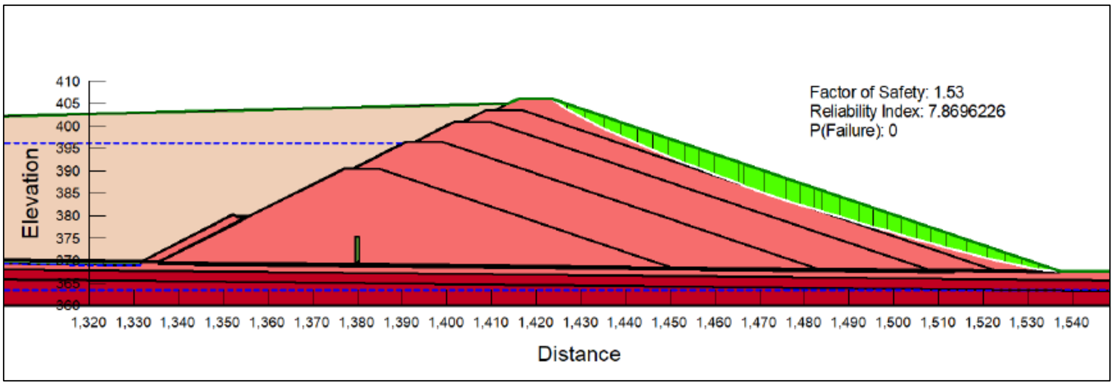


Figure 18.39: Slope Failure Through Final Phase 5 with Toe Drain and Underdrainage Not Operational

Table 18.29: TSF Static Slope Stability Analysis Results

Slope Stability Analysis and Failure Type	Minimum Required FoS	Deterministic FoS	Probability of Failure	Reliability Index
Phase 1				
Operational TSF (as per Design)	1.5	1.56	< 1/143 000	8.10
Non-Operational Toe Drain and Underdrainage	1.3	1.56	< 1/143 000	8.10
Extreme Flood Event to Crest Level	1.3	1.56	< 1/143 000	8.10
Phase 2				
Operational TSF (As Per Design)	1.5	1.54	< 1/143 000	8.02
Toe Drain and Underdrainage Non-Operational	1.3	1.54	< 1/143 000	8.02
Phase 3				
Operational TSF (As Per Design)	1.5	1.58	< 1/143 000	7.95
Toe Drain and Underdrainage Non-Operational	1.3	1.58	< 1/143 000	7.95
Phase 4				
Operational TSF (As Per Design)	1.5	1.53	< 1/143 000	7.87
Toe Drain and Underdrainage Non-Operational	1.3	1.53	< 1/143 000	7.87
Final Phase 5				
Operational TSF (As Per Design)	1.5	1.53	< 1/143 000	7.87
Toe Drain and Underdrainage Non-Operational	1.3	1.53	< 1/143 000	7.87

The critical slip circles for the pseudo-static analysis are shown in Figure 18.40 to Figure 18.44. Table 18.30 summarises the pseudo-static results for the TSF under normal operational drain and pool conditions.

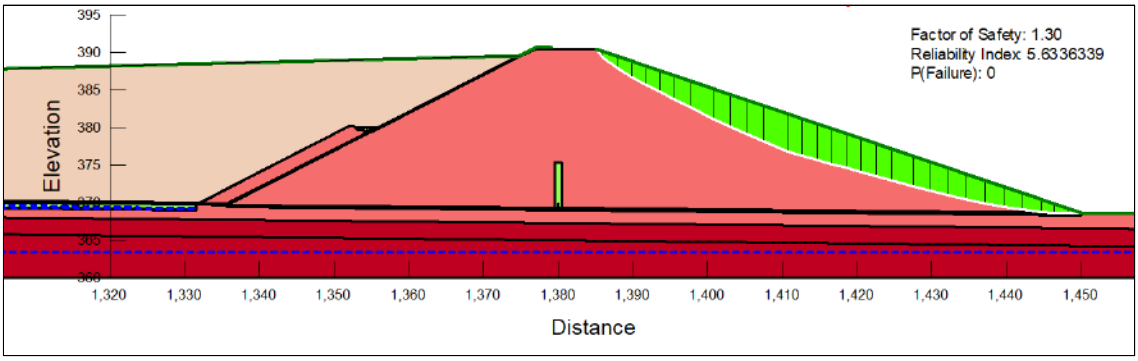


Figure 18.40: Slope Failure Through Operational Phase 1 for the Pseudo-Static Analysis

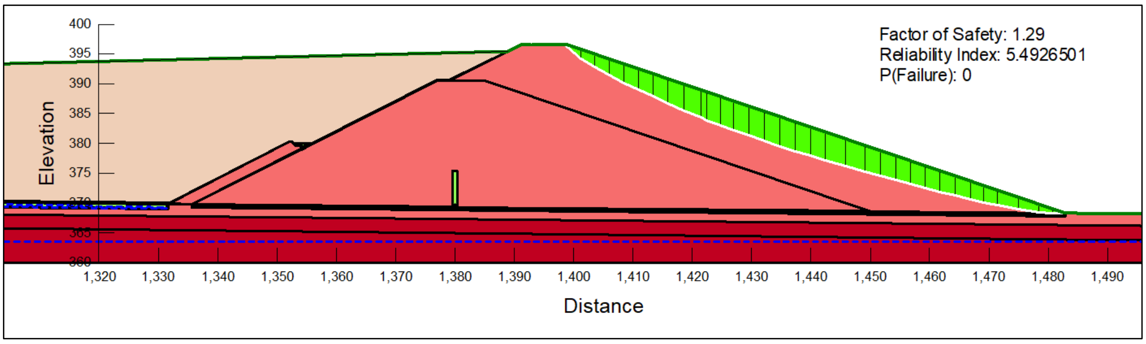


Figure 18.41: Slope Failure Through Operational Phase 2 for the Pseudo-Static Analysis

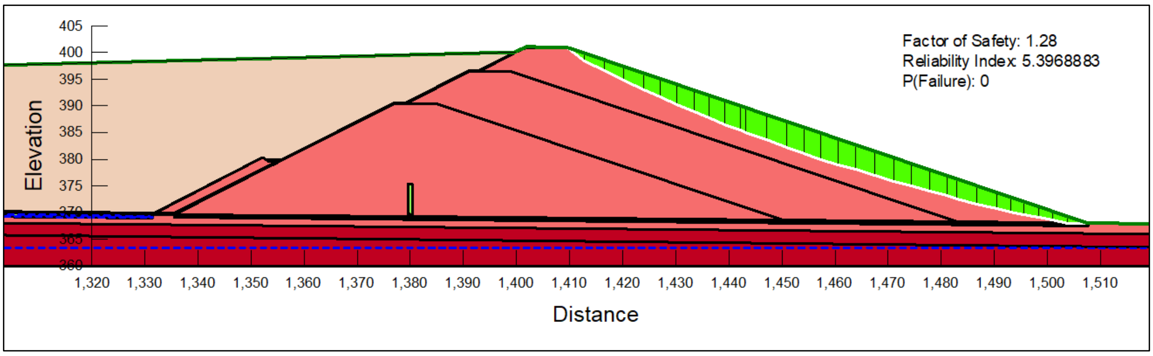


Figure 18.42: Slope Failure Through Operational Phase 3 for the Pseudo-Static Analysis

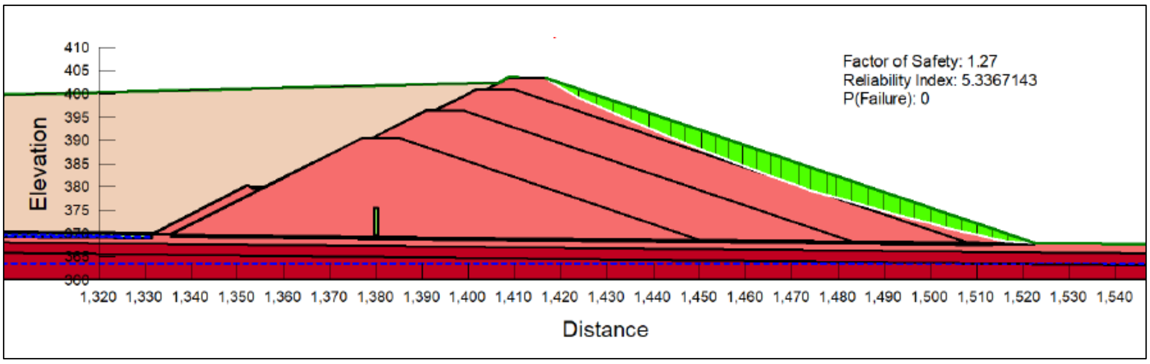


Figure 18.43: Slope Failure Through Operational Phase 4 for the Pseudo-Static Analysis

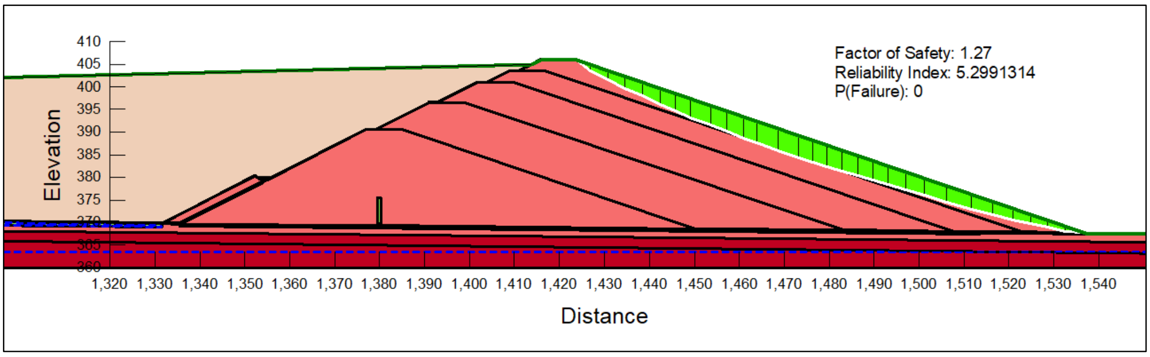


Figure 18.44: Slope Failure Through Operational Phase 5 for the Pseudo-Static Analysis

Table 18.30: Pseudo-Static Analysis Results

Slope Stability Analysis and Failure Type	Minimum Required FoS	Deterministic FoS	Probability of Failure	Reliability Index
Phase 1				
Operational TSF (as per Design)	1.0	1.40	< 1/143 000	6.06
Phase 2				
Operational TSF (As Per Design)	1.0	1.29	< 1/143 000	5.49
Phase 3				
Operational TSF (As Per Design)	1.0	1.28	< 1/143 000	5.40
Phase 4				
Operational TSF (As Per Design)	1.0	1.27	< 1/143 000	5.34
Final Phase 5				
Operational TSF (As Per Design)	1.0	1.27	< 1/143 000	5.30

The outcomes from the slope stability analyses are summarised as follows:

- The critical slip circles occur within the confines of the main embankment, thus negating the influence of the tailings geotechnical parameters and the phreatic surface within them.
- The factor of safety, reliability index and probability of failure of the TSF for the various scenarios considered meet the minimum prescribed values.

18.1.11.12 Capital and Operating Cost Estimates

18.1.11.12.1 Initial Capital and Sustaining Capital Cost Estimate

The CAPEX associated with the TSF has been determined to a Class 2 AACE accuracy of (+20%, -15%) based on quantities measured by Epoch and rates sourced from earthworks and liner tender enquiries undertaken and provided by SENET. Costs are in US$.

The CAPEX estimate has been apportioned across the LOM for each phase of the TSF, with the first Phase 1A as the initial CAPEX, and the sustaining CAPEX comprising the remaining phases. Phasing of the facility and the associated CAPEX over the LOM allows for the deferral of costs into sustained CAPEX, thus improving the initial, LOM cash flow requirements, internal rate of return and net present value of the project.

Table 18.31 presents the initial and sustaining TSF CAPEX that occurs over the LOM. The P&Gs are set to 26.7 % of the measured works based on the original 2020 DFS quantities. No contingencies have been allowed for and this is catered for under an overall project contingency elsewhere.

Table 18.31: Initial and Sustained CAPEX for the Kobada TSF

Description	Amount (US$ Million)						Percentage of Total Measured Works
Schedule	Phase 1A	Phase 1B	Phase 2	Phase 3	Phase 4	Phase 5	
Site Clearance	1.64	0.00	0.46	0.44	0.35	0.28	4.8%
Earthworks and Excavations	4.81	1.36	4.62	6.94	4.94	2.71	38.4%
Drainage	11.94	2.71	3.58	3.17	2.49	1.88	39.0%
Concrete Structures	0.06	1.61	0.33	0.32	0.32	0.32	4.5%
Pipework	1.59	0.00	2.08	1.41	0.09	0.00	7.8%
Gabions	0.00	0.14	0.14	0.14	0.14	0.14	1.1%
Catwalk	0.01	0.00	0.00	0.00	0.00	0.00	0.0%
Warning Signage and Safety	0.30	0.00	0.00	0.00	0.00	0.00	0.5%
Miscellaneous	0.12	0.15	0.37	0.52	0.65	0.72	3.8%
Total Measured Works	20.47	5.98	11.58	12.95	8.98	6.06	100.0%
P&G Costs	5.47	1.60	3.09	3.46	2.40	1.62	
Total CAPEX per Phase	25.94	7.58	14.68	16.41	11.38	7.67	
Total CAPEX	**83.65**						

Figure 18.45 provides the percentage split of the initial CAPEX associated with TSF Phase 1A, with the earthworks (24%) and drainage (including HDPE lining and geotextile) (58%) making up the predominant costs, with the remaining items collectively amounting to 18% of the total initial TSF CAPEX.

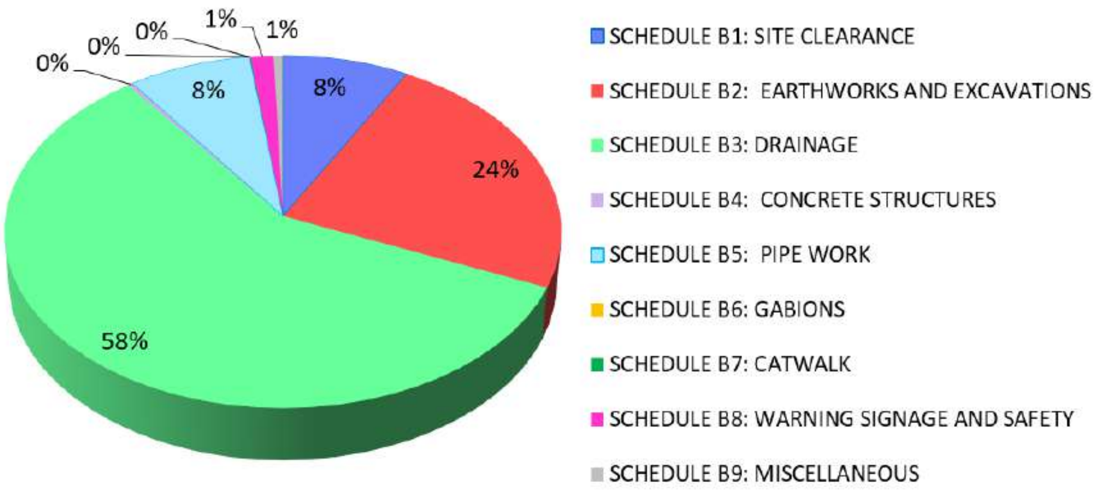


Figure 18.45: Phase 1A CAPEX Percentage Split

18.1.11.12.2 Operating Cost Estimate

The operating costs (OPEX) associated with the TSF have been estimated at US$12.58 million over the LOM, based on typical full containment operational costs sourced from tailings dam management companies. The OPEX of US$0.74 to US$ 0.9 million per annum comprises the following:

- US$0.6 million per annum for operational management, comprising a team leader/manager, a supervisor, unskilled labour, and a spares workshop. This team shall be responsible for:
  - Day to day depositional management
  - Maintaining the pool wall
  - Maintenance and repairs to the slurry delivery pipeline and valves
  - Monitoring and cleaning of the drains
  - Seepage collection sump pump monitoring
  - General maintenance (cleaning trenches)
  - Monitoring various components (freeboard, drain flows, water returns, rainfall, tonnes deposited, etc.)
- US$0.08 to US$0.31 million per annum for pipeline and valve replacement costs and maintenance, depending on the pipeline length per phase of construction; and
- US$0.06 million per annum for quarterly inspections, monitoring and quarterly reports by the design engineer.

18.1.11.13 Reclamation, Closure and Aftercare

A conceptual concurrent reclamation and closure plan for the TSF has been developed and an estimate of the required financial provisions to ensure that the plan can be implemented has been determined. It is expected that the closure plan shall be updated periodically during the development and operational phase of the mine to reflect progress in the reclamation and closure process, as well as to identify any additional closure liability issues arising through operations.

The reclamation of the TSF shall include activities that take place concurrently with its development and operation, and the closure plan includes those activities associated with its final decommissioning as well as closure. At closure, the TSF shall remain a permanent feature on the landscape but will be completely enclosed by the surrounding hills and constructed embankments.

The following costs will be incurred as part of the CAPEX required for the construction of the facility and are accounted for in the associated CAPEX budgets:

- During the development of the TSF, topsoil shall be stripped from the footprint of the containment wall and TSF basin and stockpiled for use in the reclamation process.
- Upon completion of each phase of the embankment wall lift, the outer slopes of the wall shall be covered with a layer of topsoil in preparation for the establishment of a sustainable indigenous vegetative cover. This is to be completed during the operational phase of the mine.
- Upon completion of each phase, an emergency spillway will be constructed/augmented from the previous phase to ensure that the minimum requirements for freeboard above the IDF are met. The spillway invert has been designed to the level of the EDF to protect against the release of potentially contaminated water from the crest of the facility.

Runoff from the basin of the TSF shall report to the lowest area within the basin in perpetuity. The volume of water held in the decant pool area shall fluctuate in size depending on the season and occurrence of rainfall. Only the exposed beach surface areas, which shall be permanently above the final water level, are to be reclaimed.

SENET has specified that the arsenic levels shall be precipitated out in the process plant to achieve concentrations below 100 mg/L as per the WHO guidelines. The tailing area shall be topped with a 150 mm layer of topsoil, prepared for revegetation and sown with indigenous vegetation species.

Interstitial water shall, during the operational phase of the facility, be drained via the toe drain underdrainage system installed throughout the tailings basin to reduce the phreatic surface within the TSF. The TSF toe drain and underdrainage system is expected to continue to operate for several years after completion of revegetation and active closure as interstitial water shall continue to drain from the TSF. These drains shall report to the collection sump, which shall be dewatered by pumping and discharged onto the TSF surface area. The discharged water may also require additional treatment.

Personnel from the environmental and closure teams shall direct reclamation and post-closure monitoring. A detailed closure/decommissioning plan is to be prepared prior to decommissioning the TSF.

Table 18.32 shows the estimated costs for the concurrent reclamation, decommissioning, end of LOM reclamation and aftercare maintenance amounting to US$11.06 million within an accuracy of ± 35 %. P&Gs have been included at 26.7 % of measured works, expect for aftercare and maintenance in the years after the operational LOM, amounting to a total of US$14.50 million (see Table 18.33).

Table 18.32: Closure Plan Estimate across the LOM

Item	Description	Amount (US$)
1	Load from stockpile, haul, place and spread 150 mm selected topsoil in preparation for the establishment of vegetation to:	
1.1	Embankment wall outside slope	Included in CAPEX
2	Supply and hand plant a selection of indigenous grasses, shrubs and trees, together with any fertilisers and ameliorants required to ensure establishment of vegetative cover to steep and erosion prone areas of: (Rate includes testing of soils, weed control and reseeding of areas where germination is unsuccessful)	
2.1	Embankment wall outside slope	Included in CAPEX
3	Repairs to earthworks and topsoil placement as 3 % of cost of original work	840,000.00
4	Active care of vegetation as 8 % of cost of topsoil and vegetation establishment	200,000.00
5	Passive care of vegetation as 8 % of cost of topsoil and vegetation establishment	200,000.00
6	Construct emergency spillway and associated energy dissipation structures	Included in CAPEX
7	Dismantle and remove pipework and fittings as 5 % of cost of pipework	255,000.00
8	Construct emergency spillway and associated energy dissipation structures	Included in CAPEX
9	Load from stockpile, haul, place and spread 150 mm selected topsoil in preparation for the establishment of vegetation to:	
9.1	Exposed tailings and erosion prone areas (300 000 m^3 at a rate of US$4.94/ m^3	2,223,000.00
9.2	Embankment wall outside slope	Included in CAPEX
10	Supply and hand plant a selection of indigenous grasses, shrubs and trees, together with any fertilisers and ameliorants required to ensure establishment of vegetative cover to steep and erosion prone areas of: (Rate includes testing of soils, weed control and reseeding of areas where germination is unsuccessful)	
10.1	Exposed tailings and erosion prone areas (2 000 000 m^2 at a rate of US$1.80/ m^2)	5,400,000.00
10.2	Embankment wall outside slope	Included in CAPEX

Item	Description	Amount (US$)
11	Repairs to earthworks and topsoil placement as 3 % of cost of original work	105,000.00
12	Active care of vegetation as 8 % of cost of topsoil and vegetation establishment	432,000.00
13	Passive care of vegetation as 8 % of cost of topsoil and vegetation establishment	432,000.00
14	Repairs to earthworks and topsoil placement as 3 % of cost of original work	105,000.00
15	Active care of vegetation as 8 % of cost of topsoil and vegetation establishment	432,000.00
16	Passive care of vegetation as 8 % of cost of topsoil and vegetation establishment	432,000.00
Total		**11,056,000.00**

The total LOM cost associated with the TSF is estimated to be US$114.66 million (Net Present Value). This includes detail design of the TSF and construction supervision during the construction of the various phases (a total of US$3.93 million). The reclamation, closure, and post closure monitoring of the TSF is estimated at US$14.50. The annualised cash flow for the LOM is summarised in Table 18.33.

Table 18.33: Annualised LOM Cash Flow associated with the Kobada TSF

Kobada Gold Mine TSF
Annualised Cash Flow
(Feasibility Design to End of LoM)

			Pre-operational Phase		TSF Operational Phase															Closure and post closure					
			Design of Phase 1			Design of Phase 2			Design of Phase 3			Design of Phase 4			Design of Phase 5					Decommission and reclamation	Aftercare and Maintenance				
				Construction of Phase 1A	Construction of Phase 1B		Construction of Phase 2			Construction of Phase 3			Construction of Phase 4			Construction of Phase 5									
Number	**Item**	**NPV**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**10**	**11**	**12**	**13**	**14**	**Total**
1.0	**TSF Capital Cost (USD)**																								
1.1	Tailings Storage Facility	66.02		20.47	5.98		11.58			12.95			8.98			6.06									66.02
1.2	Contingencies (Included in Overall Design)	0.00		0.00	0.00		0.00			0.00			0.00			0.00									0.00
1.3	Tailings Storage Facility P & G's	17.63		5.47	1.60		3.09			3.46			2.40			1.62									17.63
	Sub-Total	83.65	0.00	25.94	7.58	0.00	14.68	0.00	0.00	16.41	0.00	0.00	11.38	0.00	0.00	7.67	0.00	0.00	0.00	0.00					83.65
2.0	**Engineering Costs (USD)**																								
1.1	Feasibility Design	0.00	0.00																						0.00
1.2	Detailed Design	0.85	0.25			0.15			0.15			0.15			0.15										0.85
1.3	Construction Supervision	3.08		0.56	0.28		0.56			0.56			0.56			0.56									3.08
	Sub-Total	3.93	0.25	0.56	0.28	0.15	0.56	0.00	0.15	0.56	0.00	0.15	0.56	0.00	0.15	0.56	0.00	0.00	0.00	0.00					3.93
3.0	**TSF Operating Costs (USD)**																								
3.1	TSF Deposition Management Costs	8.97			0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60						8.97
3.2	TSF Operational Costs (i.e. pipeline and valve replacement costs, maintenance, etc.)	2.71			0.24	0.24	0.24	0.31	0.31	0.31	0.21	0.21	0.21	0.06	0.06	0.06	0.08	0.08	0.08						2.71
3.3	Consulting Services (i.e. quarterly inspections, quarterly reports, slope stability assessments, monitoring, etc.)	0.90			0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06						0.90
	Sub-Total	12.58	0.00	0.00	0.90	0.90	0.90	0.97	0.97	0.97	0.87	0.87	0.87	0.72	0.72	0.72	0.74	0.74	0.74	0.00					12.58
4.0	**TSF Closure and Rehabilitation Costs (USD)**																								
4.1	Engineering Costs (i.e. design and supervision of closure and rehabilitation measures)	0.75					0.10			0.10			0.10			0.10			0.35						0.75
4.2	Closure and Rehabilitation Measures Undertaken during Operational and Decommissioning Phase (i.e. re-vegetation, sealing of pipes, etc.)	13.75			0.31			0.31			0.31			0.31			0.31			5.77	2.96	2.96	0.24	0.24	13.75
	Sub-Total	14.50	0.00	0.00	0.31	0.00	0.10	0.31	0.00	0.10	0.31	0.00	0.10	0.31	0.00	0.10	0.31	0.00	0.35	5.77	2.96	2.96	0.24	0.24	14.50
	Total	**114.66**	**0.25**	**26.50**	**9.07**	**1.05**	**16.23**	**1.28**	**1.12**	**18.04**	**1.18**	**1.02**	**12.91**	**1.03**	**0.87**	**9.05**	**1.05**	**0.74**	**1.09**	**5.77**	**2.96**	**2.96**	**0.24**	**0.24**	**114.66**

18.1.11.14 Conclusions

The following conclusions may be drawn from the work documented in this section:

- The initial TSF sites have been reassessed to affirm that the current site location is still preferred for the revised TSF storage capacity.
- The TSF has been designed to accommodate a storage capacity of 45.15 Mt over a 15.1-year LOM and comprises an HDPE-lined full-containment valley dam with a maximum height of 36 m and a footprint area of 370.0 ha.
- The zone of influence for the TSF has been determined, and the facility has been classified conservatively as a Very High Hazard Facility in accordance with the CDA dam classification.
- The mine wide water balance is positive, i.e., a surplus of water in generated within the mine wide water circuit, requiring discharge. Surplus water from the open-pit dewatering boreholes and open-pits themselves due to storm water accumulation during the wet season, needs to be discharged into the downstream environment when not consumed by the process plant.
- The design of the TSF has been undertaken to sufficient detail to determine CAPEX and OPEX to a Class 2 AACE accuracy (+20 %, −15 %), with the following results:
  - An initial CAPEX of US$25.94 million and remaining sustained CAPEX of US$57.71 million have been estimated.
  - The engineering costs associated with the TSF have been estimated to be US$3.39 million over the LOM and include design and construction supervision costs.
  - The OPEX associated with the TSF has been estimated to be US$12.58 million over the LOM. This is a total of US$0.74 to US$0.97 million per annum and varies depending on the pipeline length per phase of construction.
  - The TSF closure, reclamation and aftercare estimated costs for the concurrent reclamation, decommissioning, end of LOM reclamation and aftercare maintenance were determined to be US$14.50 million within an accuracy of ± 35 %.
  - The total LOM cost associated with the TSF over the duration of the project life has been estimated with an NPV of US$114.66 million at a zero-discount rate.

18.2 PROJECT OFF SITE INFRASTRUCTURE

The proposed infrastructure will support the mining and plant operations. Camp accommodation will also be provided at the site for senior AGG mine and plant site personnel. The main off-site infrastructure required for the development of the project will be the following:

- Mining infrastructure and buildings
- Camp and catering facilities
- Medical facilities
- Power supply and distribution
- Fuel storage
- Communication
- Water supply system

- Sewage disposal

18.2.1 Mining Infrastructure and Buildings

18.2.1.1 In-Pit Haul Roads

A system of mining haul roads will be constructed across the site to connect the pits with waste rock facilities, the plant area, and tailings storage facilities.

The network of haul roads comprises a main haul road and 11 pit haul roads of the typical cross section shown in Figure 18.46.

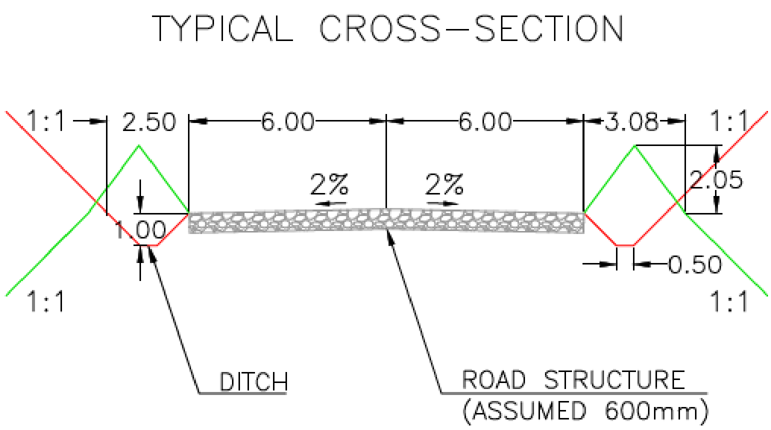


Figure 18.46: Typical Cross Section of Haul Roads

18.2.1.2 Haul Road Culverts

The haul road culverts will comprise the following:

- Excavate for pipe trenches.
- Lay bedding material.
- Lay Class 100D precast pipe culverts of 600 mm diameter.
- Construct reinforced concrete culvert bases, headwalls and wingwalls.

18.2.1.3 Mining Administration Building

The mining administration building will be shared with the process plant administration staff and will be a prefabricated, Chromadek panel building (37 m × 26 m) located adjacent to the process plant entrance and en route to the mining infrastructure.

The mining administration building will comprise the following:

- A minimum of 20 offices on the perimeter of the building
- 3 storerooms
- 1 central open-plan office area
- 1 kitchen

- 2 boardrooms
- 1 female ablution arrangement
- 1 male ablution arrangement

The prefabricated structure will be founded on a mesh reinforced concrete slab. A 2 m concrete apron slab, which will be covered by an awning, will form a walkway at the entrance to the building.

Security fencing will be provided around the main administration building perimeter and parking area.

18.2.1.4 Mining Contractor’s Equipment Workshop

The Mining Contractor is responsible for the installation of their own equipment workshop and will provide the appropriate tooling required. This facility will also include areas dedicated to equipment maintenance, a wash station, and a warehouse facility.

The Mining Contractor is responsible for providing their own level, prepared platform, complete with storm water management and drainage, in a suitable location on which to build their workshops.

18.2.1.5 Light Vehicle Workshop

The light vehicle workshop will be located in the same vicinity as the Mining Contractor’s equipment workshop.

18.2.1.6 Mining Equipment Refuelling Facility

The Mining Contractor is responsible for refuelling the mining equipment. A refuelling facility, which will be constructed by the fuel supply subcontractor will be available in the area. The average daily fuel consumptions for the mining fleet are a minimum of 7,500 L with a maximum peak demand of 17,330 L.

Under normal operating conditions, at least 14 d of diesel fuel will be stored on site with a minimum stockholding of 240 000 L.

The mining fleet dispensing station shall comprise:

- 2 high-volume mine truck filling pumps (type Volvo A60 at 55 t capacity)
- 2 standard vehicle pumps

18.2.1.7 Explosives Magazine Storage

The Mining Contractor is responsible for the supply and storage of the explosives and blasting accessories required for the liberation of ore. The Mining Contractor is responsible for providing their own level, prepared platform, complete with storm water management and drainage, in a suitable location on which to build their explosives magazine storage facility.

The Mining Contractor shall ensure that the facility is properly secured and that it complies with the requirements of the Mali Standard for the construction of explosives storage.

18.2.2 Camp and Catering Facilities

18.2.2.1 Existing Camps

AGG has currently established certain administrative and support infrastructure off site:

- Exploration camp, complete with sample storage buildings and accommodation for exploration and security staff. The camp has its own kitchen and ablution facilities.
- AGG management and visitor camp, i.e. 40-man expatriates' camp (termed New Camp), complete with raw water borehole and storage tank arrangement, and reticulated services including electrical power diesel generators. The New Camp also has a kitchen/dining hall arrangement.

18.2.2.2 Upgraded New Camp

During the construction phase, AGG will utilise the New Camp (existing 40-man expatriates' camp) for the EPCM construction management personnel and the AGG management personnel. During construction, this camp will be upgraded to a 100-man camp, complete with additional kitchen and laundry equipment, recreation hall, containerised sewage plant and new service connections, i.e. an electrical supply connection from the new main HFO/PV power generation facility and a raw water supply connection from the Niger River feed system to the process plant.

The New Camp is an existing expatriates' camp built in 2019 that comprises the following:

- Single and double quarters (sized for 40 people – sharing)
- Dining hall and kitchen (sized for 100 people)
- Borehole, raw water storage and septic tank arrangements (sized for 40 people)
- Services reticulation (sized for 40 people)
- Fencing, gate, and gatehouse for security control
- 10 kVA and 30 kVA generators, complete with reticulation to existing building units

The New Camp shall be upgraded as follows:

- Additional prefabricated single and double quarters (sized for 60 people; this will result in a total capacity of 100 people)
- Prefabricated laundry (sized for 100 people)
- Raw water take-off from the process plant raw water supply from the Niger River to the camp, complete with a localised potable water treatment plant and additional potable water storage tank
- Containerised sewage treatment plant (sized for 100 people)
- Additional services reticulation (sized for 60 people; this will result in a total capacity of 100 people)
- Earthworks for terraces and civil works for new building slabs
- Additional fencing
- Additional equipment and furniture, i.e. catering, laundry and accommodation
- Electrical supply from the new HFO power plant, complete with reticulation to the new prefabricated buildings (The 10 kVA and 30 kVA generators shall serve as backup.)

Figure 18.47 shows the new camp upgrades.

Figure 18.47: New Camp Upgrades

The upgrades shall enable operational senior management and technical staff to be accommodated at the New Camp, with catering and housekeeping being provided by an outsourced catering contractor.

18.2.2.3 Catering Facilities

A Catering and Housekeeping Services Contractor shall fulfil the catering and camp management services, including food and consumables management, room cleaning, personnel laundry services, and pesticide spraying.

In addition to catering for the staff at the camps, food will also be prepared for the lunch time meals at the process plant.

18.2.3 Medical Facilities

A clinic will be established and be manned by a fully qualified General Practitioner (minimum seven-year trained), licensed by the Mali Medical Council and experienced in the delivery of the full range of general practitioner services. To meet the site requirements, the Medical Services Vendor will provide a two-person general practitioner team on rotation, which is expected to be three weeks on site and one week on leave.

The Medical Services Vendor will coordinate emergency medical evacuation from site with their own ambulance to a Bamako hospital of AGG's choice and onward evacuation of expatriates from Bamako International Airport by working with the Bamako medical facility and utilising AGG's existing medical insurance evacuation policy.

The clinic shall comprise a facility complete with power supply, lights, air conditioning, and a sink with taps provided with potable water. As a minimum, the clinic shall include the following:

- Secure medical stores cabinet
- Secure pharmacy items cabinet
- Secure documents cabinet
- Suitable office desk and chair
- Examination table
- Examination lamp

The Medical Services Vendor shall also provide a fully equipped Land Cruiser Ambulance capable of transporting two patients (one prone and one sitting).

The use of malaria prophylactics will be mandatory, as this is the single biggest cause of lost time. The AGG Safety Officer will conduct regular campaigns of spraying the vicinity of the camp site with insecticides. Stagnant water ponds will be kept to a minimum by allowing for proper terraces and drainage around the camp site

18.2.4 Power Supply and Distribution

The power for the off-site infrastructure will be supplied by an IPP, which will construct a hybrid thermal, solar and energy storage system power plant near the process plant facility.

It was calculated that the average annual power demand for the off-site infrastructure will be 705.85 kW (not including the TSF and the plant infrastructure load). The loads are given in Table 18.34.

Table 18.34: Average Annual Power Demand

Project Load	Average Annual Power Demand (kW)	Source of Supply
Mine Infrastructure (including raw water distribution pumps and dewatering pumps)	227.85 kW	Overhead power line (3.2 km)
New Camp	320 kW	Overhead power line (8 km)
Old Camp	40 kW	Overhead power line (7 km)
Other Infrastructure (including Mining Contractor and fuel farm facilities)	108 kW	Overhead power line (1 km)

18.2.4.1 Electrical Standards and Specifications

The electrical installations are based on International Electrotechnical Commission (IEC) standards.

Low-voltage switchboards (main switchboard (MSB), MCC) shall comply with the following NF (Norme Française) standards:

- NF C15-100: French standard relating to low-voltage electrical installations
- NF C30-31-32: French standard relating to wires and cables
- NF C13-100: French standard relating to high-voltage installations
- NF C46: French standard relating to measures and control of industrial processes
- NF C60-61-63: French standard relating to protective equipment and low-voltage control

18.2.4.2 Voltage Levels

The project will be realised with a 50 Hz power system frequency and the following voltage levels:

- Power generation from plant: 11 kV
- Electricity production, big motors and electrifying of remote areas: 6.6 kV
- Big motors (for pit dewatering): 0.4 kV
- Lighting and auxiliary circuits: 230 V/400 V

18.2.4.3 Neutral System

18.2.4.3.1 Low Voltage

The neutral system for low-voltage distribution is solidly grounded.

18.2.4.3.2 High Voltage

The neutral system for high-voltage distribution is low resistance grounding.

18.2.4.3.3 Control Voltage for MCCs

The control voltage for the MCCs will be 110 V 50 Hz. This voltage will also supply the contactors of the motor starters.

18.2.4.3.4 Lighting Network

The neutral system for lighting is a neutral-to-ground system, which is provided by a local connection to earth.

The voltage classification for lighting shall be either a 3-phase 400 V or a single-phase 220 V AC system, with a neutral-to-ground distribution.

18.2.4.4 Air-Conditioning System

The air-conditioning system is connected to lighting transformers.

18.2.4.5 MV Switchgear and Distribution

Type-tested switchgear installed in the consumer substation building will serve as the main distribution point.

Provision will be made for feeder panels to provide energy to each of the plant area step-down transformers.

18.2.4.6 Transformers and Substations

Step-down transformers will be installed in close proximity to the areas of use and will reduce the voltages from 11 kV to 400 V.

The distribution and power transformers will generally be oil-insulated, double-wounded, 3-phase, 50 Hz, Dyn11 transformers that comply with the relevant standards.

18.2.4.7 Lightning Protection

The Project is located in an area that is very prone to lightning; therefore, all the units will be protected by anti-lightning systems.

18.2.4.8 MCC Panels

MCC panels consist of electric fixed starters, a motor circuit breaker, and a contactor. The panels are single-sided, waterproof IP55, even if they are of the indoor type.

The short-circuit current of all equipment will be a maximum of 75 kA. The MCC panels have the following general specifications:

- Arrival: Withdrawable circuit breaker with a current rating at least equal to the rated current of the transformers that supply it
- Rated main bus bar: Not less than the rated current of the incoming circuit
- Short-circuit current: Between 20 kA and 75 kA
- Insulation: 1,000 V
- Ingress protection rating: IP55

18.2.5 Fuel Storage

18.2.5.1 HFO

The HFO for the power generation plant will be supplied and operated by a reputable supplier in Guinea/Mali, who is currently supplying neighbouring mines.

The Fuel Supply Contractor is responsible for providing their own MCC, office building, ablutions, storage, inner roads and storm water management, oily water drainage by gravity to depot separator, and connection to plant potable water and sewerage systems.

The storage of the HFO will be at the HFO power plant, where allowance has been made for 21 days' storage of HFO for the power generation plant usage.

The HFO 180 storage tank farm comprises two 496 m^3 usable volume vertical HFO 180 tanks (8.594 m diameter × 10.0 m height), complete with HFO pump house and two 80 m^3 fire water reservoirs, fire pump house complete with two diesel fire pumps servicing both the HFO and diesel storage tank farms.

The HFO fire system consists of tank cooling by means of fixed monitors with local foam concentrate tanks and bund foam by means of fixed monitors with local foam concentrate tanks. The HFO receipt fire system consists of a receipt spill slab covered by a fixed water/foam monitor with local foam concentrate tanks.

18.2.5.2 Diesel

The diesel for the mining fleet and the process plant will be supplied and operated by the fuel supplier.

The storage of the diesel will be at the mining fleet dispensing station, located adjacent to the HFO storage facility and power plant, where allowance has been made for 21 days' storage of diesel for mining, plant and HFO power plant start-up usage.

The diesel storage tank farm comprises five 80 m^3 single-skin above-ground horizontal diesel tanks on metallic skids, complete with staircases at either end with steel walkways linking all tanks.

The diesel pumping system will be located in the local pump house and comprise duty and standby loading pumps, complete with diesel filters to 5 µm with a coalescer for heavy vehicle/truck filling at 2 × 300 L/min each, and dispensing for light vehicles at 2 × 80 L/min, all situated on dedicated islands for heavy and light vehicles.

The diesel fire system consists of tank cooling by means of fixed monitors with local foam concentrate tanks and bund foam by means of fixed monitors with local foam concentrate tanks. The diesel receipt fire system consists of a receipt spill slab covered by a fixed water/foam monitor with local foam concentrate tanks.

The diesel for plant usage will be supplied by means of a diesel bowser to a self-bunded diesel storage tank. The self-bunded storage tank can store 12 m^3 of diesel.

18.2.6 Communication

An integrated information system will be provided taking full advantage of the latest operating software, hardware and information technology enabling effective telephonic and digital communications.

18.2.7 Water Supply System

A continuous supply of water to the Kobada Gold Plant will be achieved by drawing water from the Niger River and depositing it into a raw water storage dam (via a single 400 OD HDPE pipeline), which is located closer to the Kobada Gold Plant.

Raw water extraction will be conducted by means of a floating barge pump system, rated to meet a design flow rate capacity of 435 m^3/h. Due to the isolated location of the barge system, there is insufficient electrical infrastructure to provide the necessary current to feed the electrical drives. As such, the decision was made to utilise diesel-driven pumps, as this is a better financial solution than running electrical cables from the plant to the barge location.

The floating barge pump system comprises the following components/infrastructure:

- Two 280 kW diesel-driven pumps, designed to have one pump running and one on standby
- Floating barge platform to house the pump/motor assemblies with all associated discharge piping and valves (The platform is fitted with the required handrailing and a floating walkway for maintenance access as well as pipe support.)
- A 2 m^3 diesel storage tank, providing 25 h of continuous pumping operations (consumption rate of 80 L/h for a single unit)
- Control panels to regulate the diesel drives (start/stop and speed control)

The raw water storage dam holds approximately 20,000 m^3 of water (received from the Niger River) and is solely dedicated to the Kobada Gold Plant. Water is abstracted from the raw water storage dam via a puddle pipe, which is installed into the dam wall and connected to the suction line of the centrifugal pumps installed externally to the dam. Two 90 kW electrically driven centrifugal pumps supply raw water to the Kobada Gold Plant at a design flow rate of 435 m^3/h via a single 400 OD HDPE pipeline. This water is deposited directly into the raw water pond located on site.

18.2.8 Sewage Disposal

18.2.8.1 Main Administration Building Sewage Disposal

The 55 m^3/h containerised sewage treatment plant that will be provided for the treatment and disposal of the sewage generated on the process plant site will also be connected to the off-site infrastructure sewage network.

The basis for the sizing of the sewage treatment plant includes the process plant buildings and the off-site infrastructure main administration building.

Sewage reticulation piping and manholes will be provided to facilitate the flow of sewage under gravity to a collection manhole located adjacent to the sewage treatment plant. The sewage will be pumped via a submersible pump into the containerised treatment plant.

The technology selected is compact, simple and robust, and is based on a standard activated sludge system, where the Biochemical Oxygen Demand is broken down using air and bacteria that grow in this medium. This system provides optimised nitrification and an effluent quality to a standard that complies with the requirements of the Department of Water Affairs and Forestry for the release of treated effluent back into the environment, in accordance with the General Limit Values in terms of Section 39 of the National Water Act, 1998 (Act No. 36 of 1998).

18.2.8.2 Mine Workshops and HFO Power Plant Sewage Disposal

The Mining Contractor and the HFO Power Generation Contractor will be responsible for their own sewage reticulation and disposal in accordance with regulations.

18.3 LOGISTICS

Logistics and transport studies were conducted to

- Define the possible access routes to site.
- Identify port facilities and capabilities at the point of discharge.
- Determine the most efficient routing and method of transport to site.
- Determine road/bridge upgrade requirements to ensure the safe delivery of all shipments.
- Investigate project insurance requirements.
- Determine total logistics budget to complete the movement to site of all project cargo.
- Complete a methodology to enable control of all movements.
- Establish a shipping procedure specific to the Project.
- Identify staff resource requirements along the supply chain and at the Project site.
- Determine customs and excise requirements in Mali and their effect on the Project programme/budget.

Bolloré Logistics was nominated as the freighter forwarder to conduct the route survey due to their proven track record and experience in West Africa. Furthermore, Bolloré Logistics has been recommended for the execution of the Project.

18.3.1 Routing

Three routing options were considered:

- Durban (South Africa) to Abidjan (Côte D'Ivoire) by sea, and Abidjan to site by road freight (for containers)
- Durban (South Africa) to Dakar (Senegal) by sea, and Dakar to site by road freight (for abnormal loads/break bulk)
- Johannesburg (South Africa) to Bamako Airport (Mali) via commercial airlines (for airfreight)

Based on the route and climate conditions, as well as the size of cargo to be transported, either of the first two routes (Abidjan or Dakar) will be used for the Project.

18.3.1.1 Durban/Abidjan/Dakar Route

A full route survey was initiated from Durban via Abidjan to site to determine the requirements for the movement of all cargo. This included the survey of all low water bridges and river crossings. This survey also determined the transit times en route and at border crossings.

Due to bridge restrictions, however, the Abidjan route is not recommended for abnormal loads. Prior to project execution, the roads will require further assessment to ascertain any changes in road conditions.

It was concluded from this Route Survey Report that it would take approximately 45 d to transport containerised cargo from Durban to the mine site via Abidjan (see Figure 18.48) and approximately 51 d via Dakar (see Figure 18.49), including the 21 d required from the Dakar Port to the mine site for abnormal loads (see Figure 18.50).

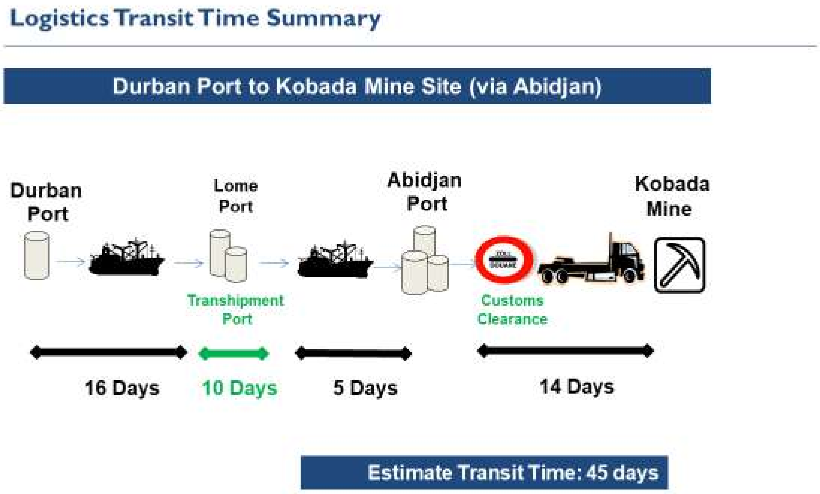


Figure 18.48: Logistics Transit Time Summary (Durban to Site via Abidjan)

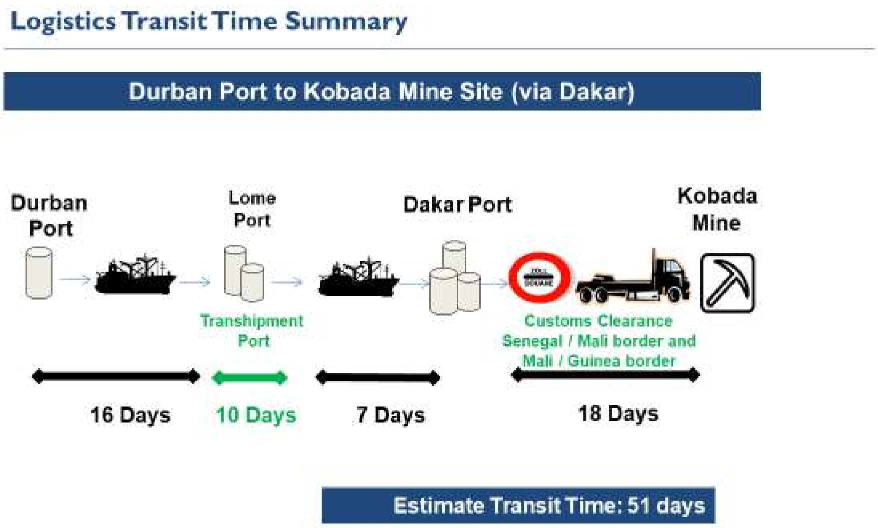


Figure 18.49: Logistics Transit Time Summary (Durban to Site via Dakar)

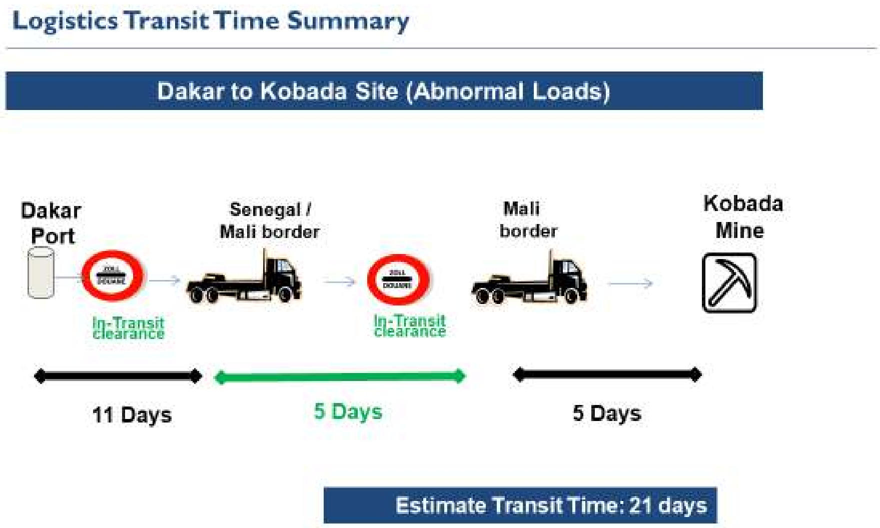


Figure 18.50: Logistics Transit Time Summary (Abnormal Loads from Dakar to Site)

18.3.1.2 Charter Aircraft and Airfreight

In the event that the project programme requires chartering an aircraft to swiftly transport goods into the country, the aircraft can be deployed into Bamako International Airport (see Figure 18.51). These costs would have to be negotiated at the time of shipment.

Figure 18.51: Bamako Airport, Mali

The landing runway at Bamako International Airport has the following specifications:

- Runway: 10.444 ft /3.180 m
- Direction: 06/24
- Elevation: 1.247 ft/380 m
- Surface: Asphalt

The landing runway, therefore, meets the landing runway distances required for the types of airplanes and specified payloads stated in Table 18.35.

Table 18.35: Typical Planes and Payloads

Aircraft	Maximum Payload	Runway Requirement	Main Hold Size	Main Door Size	Volume
	kg	m	cm (l × w × h)	cm (w × h)	m³
Antonov An-124	120 000	3 000	3 648 × 640 × 440	640 × 440	750
Boeing B747-400F	120 200	3 170	4 800 × 486 × 304	340 × 312	735
Ilyushin IL-96-400T	88 000	2 500	4 444 × 571 × 286	485 × 287	580
Ilyushin IL-76 TF	60 000	1 800	3 114 × 345 × 325	345 × 325	400
Ilyushin IL-62	40 000	3 150	2 798 × 317 × 212	345 × 200	230

Cargo can also be airfreighted to Bamako on scheduled daily flights, provided that the cargo weighs less than 3 t and its dimensions are less than 3.0 m (length), 2.0 m (width) and 1.50 m (height).

18.3.2 Port Facilities

18.3.2.1 Port of Durban

The Port of Durban (see Figure 18.52) is Africa's biggest container port in terms of capacity. Located on some of the world's busiest shipping routes, it is South Africa's main port for general cargo and containers. It handles an average of 83 000 containers each month at the Port of Durban Container Terminal.

The Port of Durban covers over 1 800 ha of land and water, and it is protected by two breakwaters that are 335 m and 700 m (1 099 ft and 2 296 ft) long. It contains 302 km of rail tracks, and the harbour is only one block away from Durban's Central Business District. Pilotage is required for all vessels entering the port from a point 4.8 km northeast of the entrance to the port, and tug assistance is also required. The port also offers extensive safe anchorage outside the port.

Figure 18.52: Port of Durban, South Africa

18.3.2.2 Port of Abidjan

The Port of Abidjan is a commercial port at Treichville, located on a lagoon and connected to the sea by a buoyed channel. Abidjan is the economic capital and the main port of Côte d'Ivoire. It is a major contributor to the economy of Côte d'Ivoire and the greater part of the external trade of landlocked countries such as Burkina Faso, Mali, Niger, Chad, and Guinea also passes through it.

The Port of Abidjan (see Figure 18.53) is modern and well-equipped, and it offers the services of a container terminal, ro-ro terminal, fishing terminal, mineral terminal, grain terminal, oil terminal, wood terminal, fruit terminal, harbour master's office, ship repair and warehouse area.

Figure 18.53: Port of Abidjan, Côte D’Ivoire

18.3.2.3 Port of Dakar

Dakar is a major seaport and the capital city of Senegal (Western Africa).

Its exceptional geographical position (the most advanced point of the West African coast and at the intersection of the routes linking Europe to South America, and North America to South Africa) allows vessels coming from the north to have a navigation gain of 2 d to 3 d compared to other ports on the West African coast.

The container terminal in the north zone of the Port of Dakar covers an area of 24 ha. It has a quay line of approximately 700 m with three berths between 12 m and 13 m, and modern handling equipment. It consists of four docks (including two Panamax ports), four 100 t Gottwald cranes on wheels, ten gantry cranes, 15 reach stackers, and 400 refrigerated outlets. The Port of Dakar (see Figure 18.54) is one of the few ports on the West African coast that vessels of all types can access at all times due to the exceptional nautical conditions of the site (maximum tide levels vary between 0.20 m and 1.80 m).

Figure 18.54: Port of Dakar, Senegal

18.3.3 Documentation

In order to ensure effective management of logistics and that all parties' expectations of the Project are met, a written project logistics guide and execution plan will be required. This logistics execution plan will outline the responsibilities of all the stakeholders (contractor/company/suppliers/other interested parties) and will indicate how cargo management and control from time of receipt by the contractor to time of delivery will be achieved. Comprehensive project logistics documents and an execution plan applicable to the Kobada Project were developed during the study.

18.3.4 Project Cargo

Table 18.36 provides a summary of the project cargo, outlining tonnages, number of containers and shipping method.

Table 18.36: Summary of Project Cargo

Item	Description	Freight Mass (t)	Quantity Containers	Shipping Method
1	Structural Steelwork	1 176	108	Containerised
	Plate Work	375	19	Containerised
2	Mechanical:			
	Excluding Mill	595	58	Containerised
	Tower Crane/Conveyor Belting	-	18	Containerised
3	Mills and Components:			
	Ball Mill	338	4	Containerised/Break Bulk

Item	Description	Freight Mass (t)	Quantity Containers	Shipping Method
4	Plant Piping and Valves:			
	Piping	519	65	Containerised
	Valves	28	4	Containerised
	Piping - Ball Mill	20	3	Containerised
5	Electrical	-	45	Containerised
6	C&I	-	6	Containerised
7	Civil/Earthworks	-	2	Containerised
8	Plant First Fills:			
	Ball First Fill – Grinding Media	348	18	Containerised
9	Water Treatment Plant	-	2	Containerised
10	Sewage Treatment Plant	-	2	Containerised
11	Tanks	668	70	Containerised
12	Infrastructure:			
	Pre-Engineered Steel Buildings (Steel)	149	13	Containerised
	Pre-Engineered Steel Buildings (Sheeting)	40	4	Containerised
	Prefabricated Buildings and Furniture	-	26	Containerised
	Workshop Tools/Gantry Crane/Fencing	-	12	Containerised
	Weighbridge	17	4	Containerised
13	TSF, Raw Water Dam and Ponds:			
	HDPE Lining and Geotextile – To Site	-	114	Containerised
14	Spares	-	13	Containerised
TOTAL		**4 273**	**610**	

19 MARKET STUDIES AND CONTRACTS

19.1 INTRODUCTION

The Kobada Gold Project is planned to produce approximately 100,000 oz per annum of gold contained in doré for the first 10 years or production, and then <50 000oz per annum for the remainder of the mine life. The combined gold and silver content is expected to be 98 % (the remaining 2 % is likely to consist of impurities such as copper, iron and zinc). The gold content is expected to account for approximately 95 % to 97 % of the precious metals content with the remaining amount being silver content. No deleterious elements are indicated in the ore head grade assayed, and as such these are not expected to be in the doré.

Doré bars are planned to be cast in 804 oz (~25 kg) bricks with approximate dimensions of 190 mm length, 120 mm width and 80 mm height. Twelve to fifteen pours per month are planned with the doré being transported weekly from the mine site to the export facilities in Bamako. After weighing, sampling and assaying, the doré will be packed and secured in high-security tamper-evident carry boxes (see Figure 19.1) in the gold room in the presence of mine production and security personnel.

Industry standard gold room and strong room facilities will be constructed on site to hold the doré until it is ready for export. Appropriate checks and balances, security cameras, alarms, insurance and security procedures will be implemented to cover the production and storage at the mine. Maximum storage on site will be no more than two weeks' production (i.e. 4,000 oz doré) unless exceptional circumstances dictate otherwise.

MEGA FORTRIS
GROUP

Secure Tamper-Evident Carry Box

Manufactured with high-impact ABS and fitted with plastic coated reinforced metal handles to carry heavy contents, the GB Box has a tamper-resistant slide-on security lid, with one secure sealing point.

The sealing point on the GB Box can be used with either metal barrier seals or plastic indicative seals, making the GB Box easy and cost effective to seal.

The GB Box can be used to carry a wide range of high security goods, including bulk coin, cash handling, gold bullion, jewellery, high value retail goods, duty-free goods, medical and pharmaceutical drugs.

Features

1. The slide-on lid is tamper-resistant and can be sealed.
2. Plastic coated metal fold-down handles.
3. High-impact ABS plastic box and lid.
4. Lid design allows for stacking boxes on top of one another and nesting to save on space in transit.
5. One secure sealing point which can be used with either metal barrier seals or plastic indicative seals.

Technical Specifications

PRODUCT

Product	Material	Dimensions	Box Weight	Marking Area	Max Marking Digits
GB Box	High-impact ABS	265 x 140 x 95 mm	0.675 kgs	**Cover:** 250 x 130 mm **Side:** 95 x 155 mm	*According to customers' requests*

OUTER CARTON

Product	Carton Quantity	Dimensions / mm	Weight kg/carton	Volume m³/carton	Standard Colours
GB Box	10	615 x 280 x 300 mm	7.1 kgs	0.05166	*Customisable colour – MOQ 2000 pcs*

Source: https://mfgroupmedia.blob.core.windows.net/newdatasheets/datasheet_GB_Box.pdf

Figure 19.1: High-Security Tamper-Evident Carry Box

19.2 GOLD REFINERS ACCESSIBILITY

The gold bullion bars, typically no greater than 25 kg each, produced at the Kobada Project site may be sent to any of the active gold refiners in the world for toll refining. There are several refineries suitable for transforming the doré into refined gold bullion, including facilities in South Africa, Europe, North America and the Middle East. The key determinants in choosing the refinery will be credit standing, refining experience, pricing and refining terms, transport and insurance costs, and ease of logistics. There are several refiners in the world whose bars are accepted as "good delivery" by the following associations:

- London Bullion Market Association (LBMA)
- Istanbul Gold Exchange (IGE)
- Shanghai Gold Exchange (SGE)
- The Chinese Gold and Silver Exchange Society in Hong Kong (CGSE)

Table 19.1 provides a list of some of the active refiners in Europe published by the LBMA and automatically accepted by other associations. The rest of the LBMA Good Delivery list can be accessed from www.goldbarsworldwide.com.

Table 19.1: LBMA-Accredited Active Refiners in Europe

Country	Refinery Location	Active Refiner	Year of LBMA Accreditation
Belgium	Hoboken	Umicore SA	1930
Germany	Hanau	WC Heraeus GmbH	1958
Germany	Hamburg	Norddeutsche Affinerie AG	1934
Italy	Badia al Pino Arezzo	Chimet SpA	1996
Italy	Milan	Metalli Preziosi SpA	1962
Netherlands	Amsterdam	Schone Edelmetaalberivjen NV	1934
Spain	Madrid	SEMPSA Joyeria Plateria SA	1984
Sweden	Ronnskar	Boliden Mineral AB	1984
Switzerland	Mendrisio, Ticino	Argor-Heraeus SA	1961
Switzerland	Berne	Cendres & Métaux SA	1981
Switzerland	Neuchatel	Metalor Technologies SA	1934
Switzerland	Castel San Pietro	PAMP SA	1987
Switzerland	Balerna, Ticino	Valcambi SA	1968

19.3 REFINING CHARGES, GOLD PRICING AND REVENUE

The refining process will be completed within two to three working days after receipt of the bullion. The refiner will charge refining costs, which cover melting, assaying, refining and the provision of bars accredited by LBMA.

Generally, 99.92 % of the gold contained in the bullion will be returned to AGG, and the sale price will be fixed on the day of the refinery outrun with a settlement of two to three working days.

Enquiries made with Rand Refinery, located near Johannesburg in South Africa, have indicated a treatment cost of US$0.80 per troy ounce (with a minimum refining fee of US$250 per deposit), with payment of 99.92 % for gold and 98.00 % for silver. Rand Refinery will refine the product to the LBMA Good Delivery standard and credit the payable gold and silver to the Project's account three business days after arrival at the refinery.

Gold and silver will be priced as follows:

- Gold: London AM, PM or spot price
- Silver: Prevailing London market spot price

Transport and insurance costs from site via Bamako to Johannesburg will add an additional US$2.90 per troy ounce, which equates to a total refining and transport cost of US$3.70 per troy ounce. Shipping time should not typically exceed three days.

19.4 BULLION WEIGHING, MELTING AND ASSAYING

Weighing, melting and assaying will be performed at the refinery within one business day after receipt of the doré. Samples will be taken, and some will be used for assays and others retained and sealed for an umpire assay if required.

The material will be analysed using X-ray fluorescence (XRF) analysis as part of the refinery's standard procedure. The material will be evaluated using the weight after the melt procedure, with the following splitting limits:

- Weight: 0.10 %
- Gold: 0.5 %
- Silver: 1 %

When the assay result falls within the splitting limits, the refinery's value will be used for the final settlement. When the assay result exceeds the limits, samples may be submitted to an independent umpire at AGG's request. A written notice period of at least 72 h should be given to the refinery. Should the umpire's result be within the limits of AGG's result and the refinery's assay results or coincide with either, then the arithmetic mean of the umpire assay and the party closest to the umpire assay shall be used for the final settlement. Otherwise the median of the three shall be taken as the final settlement.

Several elements, when present in feed materials, may cause health or environmental problems or damage to the refining process. There are, therefore, certain limits for the deleterious element quantities in the feed material to the refinery. There is zero tolerance for any radioactive isotopes and mercury, and the refinery specification for the other elements is given in Table 19.2.

Table 19.2: Bullion Specification

Deleterious Element	Symbol	Maximum Permitted Level (%)
Iron	Fe	2.00
Copper	Cu	10.00
Zinc	Zn	5.00
Lead	Pb	5.00
Nickel	Ni	2.00
Arsenic	As	0.20
Mercury	Hg	Not accepted

Should the maximum limits of a deleterious element be exceeded, a charge/penalty is applied. This is typically US$0.05 for every 0.10 % or part thereof in excess of these levels per dry kilogram of the material. Should mercury or radioactive isotopes be present, the bullion will not be accepted and will be returned at the mine's cost.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 INTRODUCTION

The Kobada Project is held under one contiguous mining permit (Kobada) and two exploration permits (Kobada Est and Faraba), issued in terms of the 2012 Mining Code (see Figure 4.3):

- A mining permit (Permis d'Exploitation), No. PE 15/22 (Décret No. 2015-0528/PM-RM), for Group 2 minerals was issued on 31 July 2015 to African Gold Group Mali SARL (AGG Mali SARL) and is valid for 30 years expiring on 30 July 2045. The permit must be renewed every ten years and is currently valid. It encompasses an area of 135.7 km².
- The Kobada-Est exploration permit (Permis de Recherche), No. PR 18/957 (Arrêté No. 2018-3016/MMP-SG), for gold and Group 2 minerals was issued on 16 August 2018 to AGG Mali SARL. The permit encompasses an area of 77 km² and is valid for three years. The Kobada-Est permit was recently renewed and is expected to be finalised by October 2021.
- The Faraba exploration permit, No. PR 17/921 (Arrêté No. 2018-0992/MMP-SG), for gold and Group 2 minerals was issued on 6 April 2018 to AGG Mali SARL. The permit encompasses an area of 45 km² and is valid for three years. The Faraba permit was recently renewed and is valid for a further 3 years until April 2024.

An ESIA process was undertaken between 2019 and 2021 with the project description based on the oxides DFS study completed in 2020. This assessment was based on the Carbon-In-Leach (CIL) process allowing for the processing of oxide ore and includes opencast pits, waste rock dumps, a TSF facility and supporting facilities and infrastructure. The oxides ESIA was submitted during August 2021 and is currently with the Direction Nationale de l'Assainissement et du Contrôle des Pollutions et des Nuisances (DNACPN) awaiting approval. The approval is expected during Q4 2021.

An ESIA is currently under way for the 2021 sulphide DFS study with a view to aligning the oxides ESIA with the 2021 DFS study. The ESIA will be submitted to the DNACPN for approval once completed. Key changes from the oxides study include:

- Expansion of the pit and increase in depth of the pit.
- Mining of sulphide material and the associated environmental risk that the material poses.
- Establishment of four separate stockpiles, namely:
    - High-grade oxide stockpile
    - Low-grade oxide stockpile
    - High-grade sulphide stockpile
    - Low-grade sulphide stockpile
- Increased TSF capacity and footprint.
- Increased life of mine.

20.2 LEGISLATIVE REQUIREMENTS FOR AN ESIA

In Mali, the legal framework for environmental management as well as the undertaking of mining projects and the Environmental and Social Impact Assessment Statement (ESIA) required in support of such developments is based on a series of Laws, decrees and ordinances.

The Minister of Mines has the final responsibility for the administration of mining activities, though he is assisted and delegates certain powers to the Direction Nationale de Geologie et des Mines (DNGM).

In undertaking the ESIA study, several guidelines, standards and principles, both local and international, have been taken into consideration for the Kobada Project.

The Equator Principles; which are the “financial industry benchmark for determining, assessing and managing social and environmental risk in project financing” during the ESIA, was one of the key guidelines considered. The principles are adopted to ensure that any projects financed by member institutions are developed in a manner that is socially responsible and reflect sound environmental management practices and conform to internationally recognised good practices. Other guidelines considered in conducting this ESIA include:

- International Council on Mining and Metals, Good Practice Guidance for Mining and Biodiversity, 2006.
- International Cyanide Management Code for the Manufacture, Transport, and Use of Cyanide in the Production of Gold Code. The Code is an industry voluntary programme for gold mining companies. It focuses exclusively on the safe management of cyanide and cyanidation mill tailings and leach solutions. Companies that adopt the Code must have their mining operations that use cyanide to recover gold audited by an independent third party to determine the status of Code implementation.
- The Responsible Gold Mining Principles (RGMPs) are a new framework that set out clear expectations for consumers, investors and the downstream gold supply chain as to what constitutes responsible gold mining.

The requirements of the International Finance Corporation (IFC) Performance Standards on Social and Environmental Sustainability are used to manage social and environmental risks and impacts, and to enhance development opportunities in its private sector financing in its member countries eligible for financing. Accordingly, the Performance Standards are applied to projects it finances, consistent with the provisions of the IFC’s Policy on Social and Environmental Sustainability. There are eight Performance Standards that are to be considered throughout the life of the Project:

- Performance Standard 1: Social and Environmental Assessment and Management System
- Performance Standard 2: Labour and Working Conditions
- Performance Standard 3: Pollution Prevention and Abatement
- Performance Standard 4: Community Health, Safety and Security
- Performance Standard 5: Land Acquisition and Involuntary Resettlement

- Performance Standard 6: Biodiversity Conservation and Sustainable Natural Resource Management
- Performance Standard 7: Indigenous Peoples
- Performance Standard 8: Cultural Heritage

The remaining general EHS guidelines (IFC, 2007) address occupational health and safety and community health and safety. The sector-specific EHS guidelines for mining address mining-specific environmental, occupational and community health and safety, closure and post-closure impacts and management. Specific performance indicators and monitoring guidelines are given for water use and quality, wastes, land use and biodiversity, air quality, noise and vibrations, energy use, visual impacts, and occupational health and safety.

The IFC performance and sustainability requirements, Equator Principles and EHS guidelines call for several environmental plans and management programmes. These have been summarised in Table 20.1.

Table 20.1: Requirements for Management and Monitoring Plans in Accordance with International Best Practice Guidelines

Requirement	Guideline
Social and Environmental Management System with Monitoring	IFC Performance Sustainability Standard 1 Equator Principle 2
Social and Environmental Assessment	IFC Performance Sustainability Standard 1
Training Programmes for All Staff involved in the Project	IFC Performance Sustainability Standard 1
Working Conditions and Management of Worker Relationships	IFC Performance Sustainability Standard 2
Health and Safety Programme	IFC Performance Sustainability Standard 2 Occupational Health and Safety Guidelines
Supply Chain Labour Management	IFC Performance Sustainability Standard 2
Waste and Hazardous Materials Management and Recovery Programme	IFC Performance Sustainability Standard 3
Emergency Preparedness and Response Action Plans	IFC Performance Sustainability Standard 3 and 4 EHS Guidelines
Greenhouse Gas Emissions Management	IFC Performance Sustainability Standard 3
Pesticide Use and Management	IFC Performance Sustainability Standard 3
Involuntary Resettlement Action Plan	IFC Performance Sustainability Standard 5
Biodiversity and Sustainability Action Plan	IFC Performance Sustainability Standard 6 EHS Guidelines – Mining
Indigenous People Impact Assessment and Management Plan	IFC Performance Sustainability Standard 7
Cultural Heritage Action Plan	IFC Performance Sustainability Standard 8
Public Participation and Consultation	Equator Principle 6 EHS Guidelines – General and Mining

Requirement	Guideline
Decommissioning and Closure Plan	Equator Principle 8
Monitoring Programme	Equator Principle 9
Annual Public Reporting on Performance	Equator Principle 10
Air Quality Impact Assessment Management Plan and Monitoring Programme	EHS Guidelines – General and Mining
Energy Management Programme	EHS Guidelines – General and Mining
Waste Water Management and Monitoring Programme	EHS Guidelines – General and Mining
Storm Water Management System	EHS Guidelines – General and Mining
Water Use Efficiency and Monitoring Programme	EHS Guidelines – General
Hazardous Materials (HAZMAT) Programme	EHS Guidelines – General and Mining Occupational Health and Safety Guidelines
Waste Management and Monitoring Programme	EHS Guidelines – General
Noise Management and Monitoring Programme	EHS Guidelines – General and Mining
Water Balance	EHS Guidelines – Mining
Sustainable Water Supply Management Plan	EHS Guidelines – Mining
Reuse, Recycling and Water Treatment Management Programme	EHS Guidelines – Mining
Acid Mine Drainage (AMD) Management System	EHS Guidelines – Mining
Design of TSFs and WRDs	EHS Guidelines – Mining

20.3 PROJECT AND ESIA STUDY OBJECTIVES

In terms of Decree No. 2018-0991/P-RM of 31 December 2018 on ESIAs, projects are categorised according to their impacts on the biophysical and social environment. According to this Law, the Kobada Gold Project is considered to be a Category A project, which refers to projects with “potential significant adverse environmental or social risks and/or impacts that are diverse, irreversible or unprecedented”. In terms of the Equator Principles Association, (2020), a detailed ESIA is required as the project is also considered a Category A project.

20.4 ENVIRONMENTAL BASELINE SETTING

20.4.1 Physical Environment

20.4.1.1 Climate and Meteorology

The wind field in the Project Area is bi-modal with winds from the northeast during the dry season from November to May and winds from the southwest during the wet season from June to September. The wind field during the transitions between the wet and dry seasons (April and October) is variable with wind from all directions, but with high calms and very little strong winds during these periods. The highest wind speeds from the southwest occur during the height of the wet season in July and August while the highest windspeeds from the northeast, the Harmattan winds, occur during the height of the dry season in December and January.

The Kobada Project is located in the tropical southern part of Mali with a Köppen climate classification of Aw (Tropical Savannah). Historical monthly rainfall data was also obtained for weather station in Odienne, Bamako, Kalana, Siguiri and Kedougo[2].

The monthly rainfall is shown in Table 20.2 for a normal year.

Table 20.2: Mean Monthly and Annual Rainfall for A Normal Year

Month	OCT	NOV	DEC	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP	ANNUAL
Rainfall (mm)	102	10	2	0	1	6	32	87	179	242	329	263	1,252

20.4.1.2 Geology

Gold mineralisation at Kobada is present in the laterite, saprolite and quartz veins that comprise the Project and is coeval with the hydrothermal events that introduced the quartz veins that are common throughout the area. The mineralisation extends for a minimum strike of 4 km, associated with narrow, irregular, high-angle quartz veins and with disseminated sulphides in the wall rock and vein selvages. The predominant sulphides are arsenopyrite and pyrite, while chalcopyrite is rarer. Arsenopyrite (up to 5 mm) is localised near the vein selvages and as fine-grained disseminated patches within the host rock. Pyrite was noted in finely disseminated patches within the host rocks and as euhedral crystals in the black shale, generally as trace to 3 % by volume, with up to 10 % locally in the wall rock over centimetre scale intervals adjacent to the quartz veins.

20.4.1.3 Geochemistry

Material samples used for geochemical characterisation were done in two phases. The first phase was undertaken in 2020 as part of the oxides DFS and the second phase was undertaken in 2021 as part of the current sulphides DFS. The findings are discussed below.

A total of 19 material samples were collected for cyclone testing, comminution testing, heap leach testing and CIL testing. Of these, 12 samples from the cyclone testing were provided for geochemical analysis, along with 4 samples of leached (CIL) tailings, and 1 sample from each domain that had been detoxified using sodium metabisulphite with copper sulphate as a catalyst. Subsequent to the initial round of tests, one of the waste comminution samples from each domain was tested to provide information on the waste rock material.

[2] Peens and Associates, 2020. Kobada Gold Mine Surface Water Quantities Assessment

The interpretation of the data generated during the static test programme allows several conclusions to be drawn:

- The X-ray fluorescence (XRF) spectrometer analysis for major elements is consistent with the description of the regional geology.
- The XRF data for the 19 samples identified barium and copper at concentrations that exceeded the South African total concentration threshold (TCT) for non-hazardous waste (TCT0), but only marginally, using the South African National Norms and Standards for the Assessment of Waste for Landfill Disposal under the National Environmental Management: Waste Act, 2008 (Act No. 59 of 2008). The primary concern was arsenic, which was present at concentrations that would cause the material to be classified as hazardous (> TCT2).
- The reagent leach data shows that the barium and copper concentrations in the leachate for all the samples were below the WHO drinking water guidelines, indicating that mobility is low.
- The acid-base accounting (ABA) analysis showed that the sulphur grade was very low for all but one of the samples (C3 Cyc 1). Consequently, the maximum potential acidity for most of the samples was very low.
- The acid neutralising capacity was low (< 10 kg H2SO4/t) for all but the C3 Cyc 2 sample. This is consistent with the XRF data that showed low concentrations of alkaline earth elements in the samples.
- The calculated net acid production potential (NAPP) values were negative for all but the C3 Cyc 1 and the North domain comminution test sample, where the value was positive, but below 7 kg H2SO4/t.
- The net acid generation (NAG) tests confirmed the ABA data, with the NAG pH exceeding pH 4.5 in almost all cases, suggesting that any mineral acidity generated by the oxidation of reduced sulphur had been neutralised. Several samples had NAG pH values below pH 7, suggesting a small amount of residual acidity. This could be due to metal ions (which could undergo hydrolysis) or organic acids. The only exception was the C3 Cyc 1 sample, where the NAG pH was 3.05, suggesting a small amount of mineral acidity.
- The geochemical characterisation plot, which uses both ABA and NAG test data, indicated that 18 of the 19 samples could be classified as non-acid forming. Only the C3 Cyc 1 sample was classified as potentially acid forming.
- The reagent water leach tests yielded leachate with a pH above pH 6.96 in all cases, suggesting that acid-generating salts were not present at significant levels in any sample. The exceptions were the three samples from the comminution tests, where the pH was between pH 5.2 and 5.4. For these samples, the Electric Conductivity (EC) value was only marginally above deionised water, so the magnitude of any acid released was inconsequential.
- The base metal and metalloid analysis of the leachate yielded concentrations that were below the health-based WHO guidelines for all elements, with the exception of arsenic, which was consistently above the WHO guideline.
- The arsenic concentration in the leachate for the comminution test samples, representative of the waste rock, exceeded the WHO drinking water guideline, but was well below the Malian and IFC standards.

- Of the 12 cyclone samples tested, only one registered an arsenic concentration more than the IFC EHS level for effluent while the arsenic concentrations for the rest of the samples were consistently less than 30 % of the IFC guideline.
- The arsenic concentrations in the reagent water leachate for the detoxified tailings samples and the liquid fraction of the tailings slurry exceeded the IFC EHS guideline, most likely due to increased arsenic mobility at high pH values. The tailings should be confined to a secure, fit-for-purpose TSF; therefore, they should not impact receiving waters.

Having measured the arsenic values in the detoxified tailings an arsenic balance was conducted by following the processing flow sheet. This indicates that 0.18 % of the arsenic (1.3 ppm) reports to the filtrate post-cyanide detoxification. This warranted an arsenic precipitation process to be included in the circuit to remove arsenic below the recommended environmental release standard.

In summary, the static tests performed on the 19 samples indicate that there is very limited risk of acid generation, but the presence of elevated arsenic concentrations is a concern. While mobility during the water leach tests was low, the concentrations still exceeded the drinking water standard in most of the tests.

The analyses conducted on the three comminution test samples, representative of the waste rock, indicated very little reactivity, both from an acid generation and mineral liberation perspective. There was limited mobilisation of iron and arsenic, but this is likely to have been influenced by the small particle size. Kinetic testing, using non-milled material, will provide a more accurate assessment of any risk posed by this material.

For the sulphide study a total of 35 samples were tested during this phase of work, comprising six ore samples, seven tailings samples and 22 waste rock samples.

The interpretation of the data generated during the static test programme allows a number of conclusions to be drawn:

- The most significant result is that all samples tested contained significant amounts of arsenic, with concentrations for all 35 samples exceeding the South African TCT 0 value (5.8 mg/kg). One of the ore samples (KB19_P2_12D), five of the seven tailings samples, and two of the 22 waste rock samples (KB19_P1_11 and KB19_P2_12.5) had arsenic concentrations that exceeded the TCT1 value (500 mg/kg). In addition, two of the ore samples (KB19_P1_03 and KB19_P1_12) and one of the waste rock samples (KB19_P2_12.5) contained arsenic at concentrations in excess of the TCT 2 value (2000 mg/kg).
- In addition to arsenic, all the samples exceeded the TCT 0 value with respect to barium and copper, while the seven tailings samples had nickel concentrations in excess of the TCT 0 value. However, the mobility of these elements was low under reagent water leach conditions, with leachate concentrations remaining below Mali, IFC and WHO drinking water standards
- The sulphur grades were low (< 0.5%) for 33 of the 35 samples tested, suggesting limited potential for acid generation. The two exceptions were the ore sample KB19_P1_12 (1.08%) and the waste rock sample KB19_P2_12.5 (0.63%).

- The measured acid neutralising capacity (ANC) values were relatively high (>20 kg H2SO4/t) for most of the samples and were, in all cases, higher than the calculated maximum potential acidity (MPA) values.
- The NAG data suggested that four of the six ore samples had some acid generating potential (NAG pH < 4.5), although the magnitude of the net acid generation was low (< 10 kg H_2SO_4).
- The NAG tests confirmed the tailings samples had insignificant acid generating potential and only three of the 22 waste rock samples resulted in a NAG pH below pH 4.5. For these three samples the magnitude of the NAG was low, with the highest value (10.69 kg H2SO4/t) determined for sample KB19_p2_12.5.
- Overall, the risk of acid generation is low and any risk could be reduced by co-disposing material classified as uncertain with non-acid forming material.
- The leachate pH was neutral to alkaline in all cases, indicating the absence of readily soluble acid-generating salts. The EC values were generally low, suggesting there had been little dissolution of soluble phases during the leach test.
- The anion concentrations in the leachate were low, corresponding to the low EC and total dissolved solids (TDS) values. Sulphate and chloride were the dominant anions, but were both present at concentrations substantially below the levels required to negatively impact on human health or the environment.
- The whole rock analysis highlighted the high concentrations of arsenic present in the ore, tailings and waste rock material. Analysis of the leachate confirmed that the arsenic was relatively mobile. The arsenic concentration is the leachate from all 35 samples exceeded the Malian standard and for 32 of the 35 samples exceeded the IFC EHS guideline.
- For most of the samples, the concentrations of potentially hazardous metals and metalloids in the leachate were acceptable, below the WHO drinking water, IFC EHS and Malian standards. The only exceptions were two tailings samples that had antimony concentrations marginally above the WHO standard and the KB19_P2_12.5 waste rock sample, which has a manganese concentration marginally more than the Malian standard.

It is recommended that further kinetic test work be undertaken on selected samples to confirm the arsenic risk to the receiving environment.

Various mitigation measures have been adopted and incorporated into the design as well as the EMP of the project to address the environmental risk of the material. These are summarised in Section 20.7.

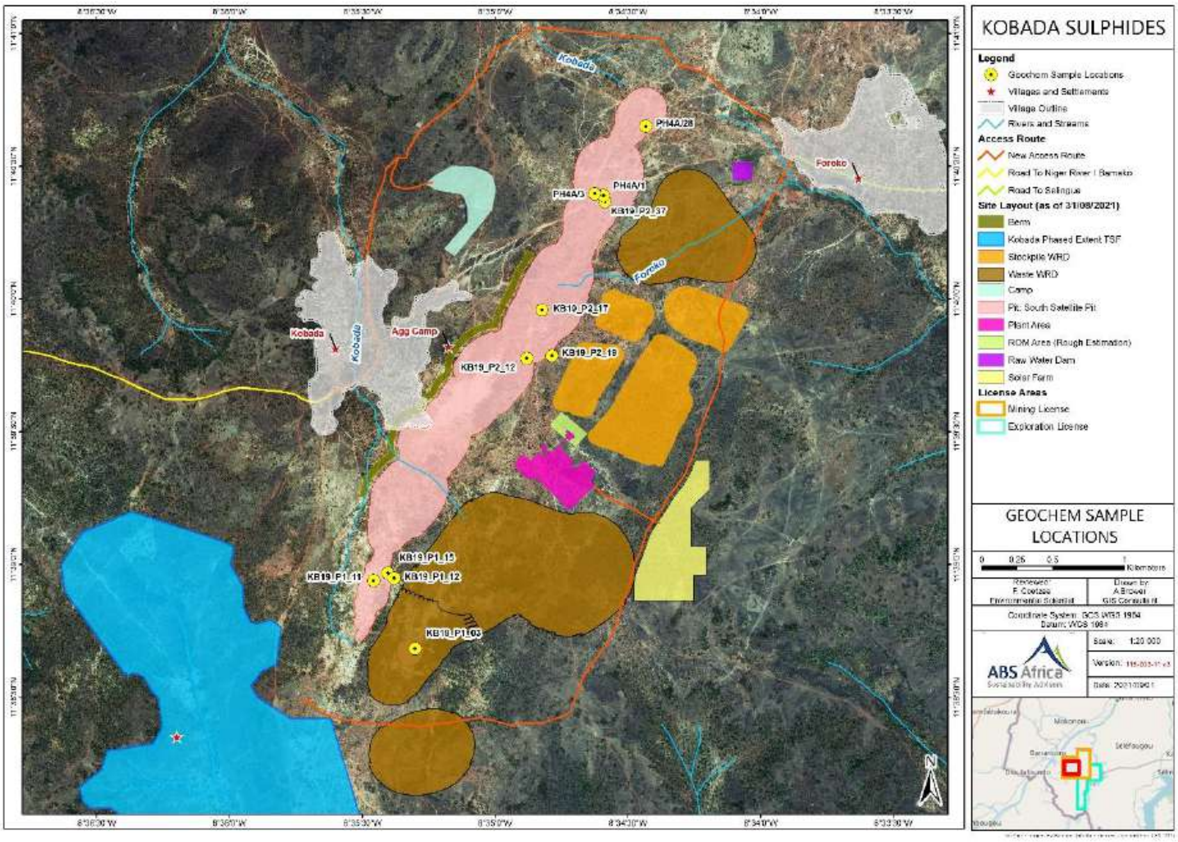


Figure 20.1: Map Indicating the Locations of the Sample Cores

20.4.1.4 Soils and Land Capability

The soils and land capability of the study area can be described as follows:

- The classification returned subtle differences in the soil texture and pedological composition across the Project Area, with generally shallow to very shallow profiles of hard pan ferricrete, which is a dominant feature of the area and considered to be sensitive due to its inability to sustain a vegetative cover. Erosion is an issue to be considered on these sensitive sites.
- A zone of highly sensitive shallow wet-based and saturated profiles is associated with the streams and well-developed waterways, and it is an area that is closely associated with the proposed pit development.
- The underlying geology and general geomorphological influences on the physical and chemical composition of the soil are again subtle, but of consequence to the sensitivity and workability (handling and storage) of the materials and the way in which the materials will react to being impacted by development (sterilisation, compaction, erosion, and possible contamination).
- There is generally a moderate to low clay content, high infiltration, and better than average drainage for most of the soils. There are also moderate to good reserves of organic carbon, a moderate to high erodibility index for the alluvial derived soils tempered by the flat to undulating terrain for all but the escarpment zone, and the

prominence of the ferricrete pavements. The outcropping and extensive flat areas of impermeable laterite returned little to no infiltration, with increased overland flows and higher than average erosion indices. These are associated with poor drainage of soil water and have wetlands status.

20.4.1.5 Surface Water

The proposed development is located in the upper reaches of the Niger River catchment. The orebody is 9 km west of the Fié River and 7 km east of the Niger River.

The Concession Area has six watercourses and four of these are within the Project Footprint. Five of these flow eastwards to the Fié River, and one flows northwards to the Niger River.

Three catchments were identified as being potentially affected by the proposed Kobada Gold Mine. Figure 20.2 indicates the three catchments draining towards the points shown as "SW-01", "SW-02" and "SW-03".

The proposed development is unlikely to impact measurably on the Fié or Niger Rivers, except for a possible new access road and bridge across the Fié River east of the proposed mine.

All catchments are predominantly flat with most of the catchment slopes measuring between 1 % and 3 %. The catchments are classified as permeable to semi-permeable. All catchments are rural with vegetation consisting mostly of thin forest and grasslands.

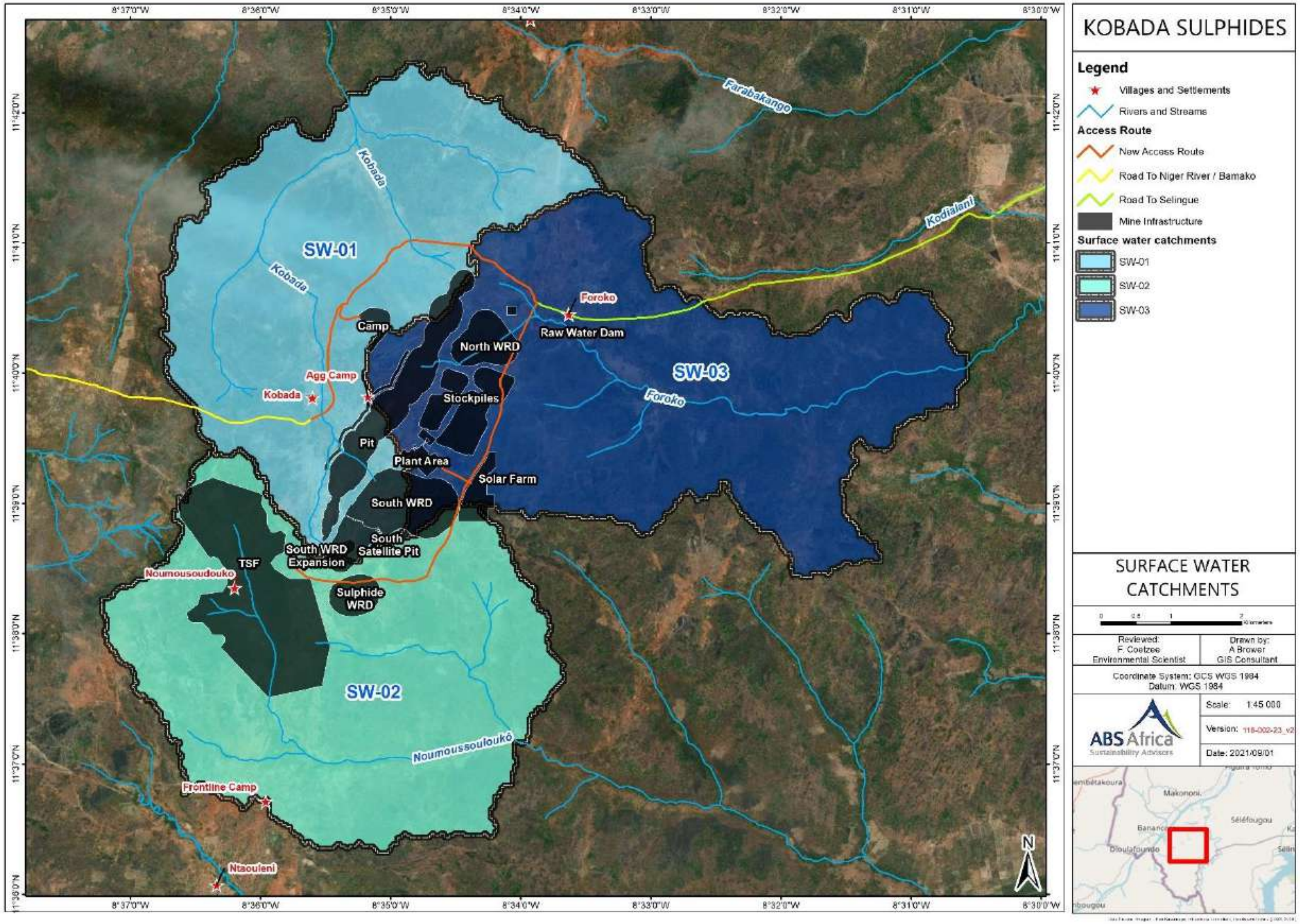


Figure 20.2: Surface Water Catchments

20.4.1.6 Groundwater

20.4.1.6.1 Aquifers

The following two aquifers occur in the area:

- Upper saprolite material aquifer:
  - The aquifer forms due to the vertical infiltration of recharging rainfall through the weathered material being retarded by the lower permeability of the underlying competent rock material. In areas, there is a transition zone consisting of saprock between the fully weathered saprolite and the underlying competent, unweathered host geology.
  - Groundwater collecting above the saprolite/saprock contact with the underlying unweathered material migrates down gradient along the contact to the lower-lying areas. In places where the contact is near surface, the groundwater can daylight on surface as springs or seepage into the various perennial and non-perennial streams and rivers that exist in the Project Area.
  - The saprolite zone can have a vertical depth of approximately 100 m.
  - The groundwater yield from this aquifer will fluctuate seasonally depending on the rainfall recharge during the wet season. It is possible that some areas of the aquifer will be laid completely dry during the dry season.
  - The groundwater qualities associated with this aquifer are expected to be relatively good due to the regular recharge from uncontaminated rainfall. However, the aquifer will also be most at risk to contamination from surface storage areas.

- Underlying competent and fractured rock aquifer:
  - Although the lower permeability of the unweathered competent rock material will retard vertical infiltration of groundwater, a percentage of the water in the upper aquifer will recharge the lower aquifer.
  - Groundwater flows in the lower aquifer are associated with secondary fracturing in the competent rock and as such will be along discrete pathways associated with the fractures. Faults and fractures in the host rock can be a significant source of groundwater depending on whether the fractures have been filled with secondary mineralisation.

The aquifers present in the area are classified as minor aquifers based on the expected relatively low yields, but they are of high importance to the surrounding communities and farmers as they are used for domestic, agricultural (stock watering) and artisanal mining purposes.

Results from aquifer tests conducted during 2014 (weathered/decomposed zone) show that the aquifer transmissivities range between 1.73×10^{-5} m^2/s and 7.64×10^{-5} m^2/s. These values equate to 1.49 m^2/d and 6.6 m^2/d. The transmissivities are relatively consistent, but it should be kept in mind that the five boreholes were drilled within less than a 50 m radius of each other, and all within the saprolite; therefore, it is expected that the values will be similar.

20.4.1.6.2 Groundwater Levels

The depth to groundwater level was measured in 10 wells. Access into a number of other groundwater points was restricted by the equipment installed, e.g. hand pumps, or the wells being closed with a lockable lid.

The depth to groundwater level in the boreholes ranged between 1.00 m and 10.30 m. The average depth to groundwater level in the saprolite aquifer was calculated to be 4.77 m based on the available data.

Plotting the depth to groundwater level compared to topographical elevation shows an 85.1 % correlation with the saprolite aquifer. The depth to groundwater level in F7Koba (see Figure 20.3) could be considered to be slightly anomalous compared to the other groundwater points and influences the correlation between topographical elevation and groundwater level elevation. The reason for the slightly lower groundwater level is not known, except that this is the second highest yielding well after F2Koba2 and is used frequently.

20.4.1.6.3 Groundwater Qualities

Groundwater samples were collected from 7 of the 23 hydrocensus points in 2020 and submitted to an ISO/IEC 17025 accredited laboratory for chemical analysis. The water qualities were compared to the WHO drinking water standards. In general, the groundwater quality is good, with few parameters exceeding the WHO drinking water quality guideline values. The elements that exceeded the guidelines are the following:

- Nickel: The nickel concentration in sample “New Camp” measured 0.187 mg/L and exceeded the WHO drinking water quality guideline value of 0.07 mg/L.
- Arsenic: The arsenic concentrations in hydrocensus points “F2Koba2” (0.019 mg/L), “New Camp” (0.114 mg/L) and “Sub1F” (0.064 mg/L) exceeded the WHO drinking water quality guideline value of 0.01 mg/L. At concentrations below 0.01 mg/L, no health effects are expected. Concentrations of between 0.01 mg/L and 0.2 mg/L are still tolerable, but there is a low risk of skin cancer in highly sensitive individuals over the long term. Concentrations between 0.2 mg/L and 0.3 mg/L increase the possibility of mild skin lesions over the long term.

In terms of cations, all the samples are sodium dominant. Chloride and bicarbonate are the dominant anions. In general, it can be said that the chloride-dominant samples are associated with the Foroko village areas, while the bicarbonate-dominant samples are from Kobada village and the TSF area. One exception is Well Sub1F, which is located within the Foroko village. When the sample was taken, chemicals were being added to the water in the well to improve the quality, which could explain the difference in groundwater character compared to the other well in the Foroko village.

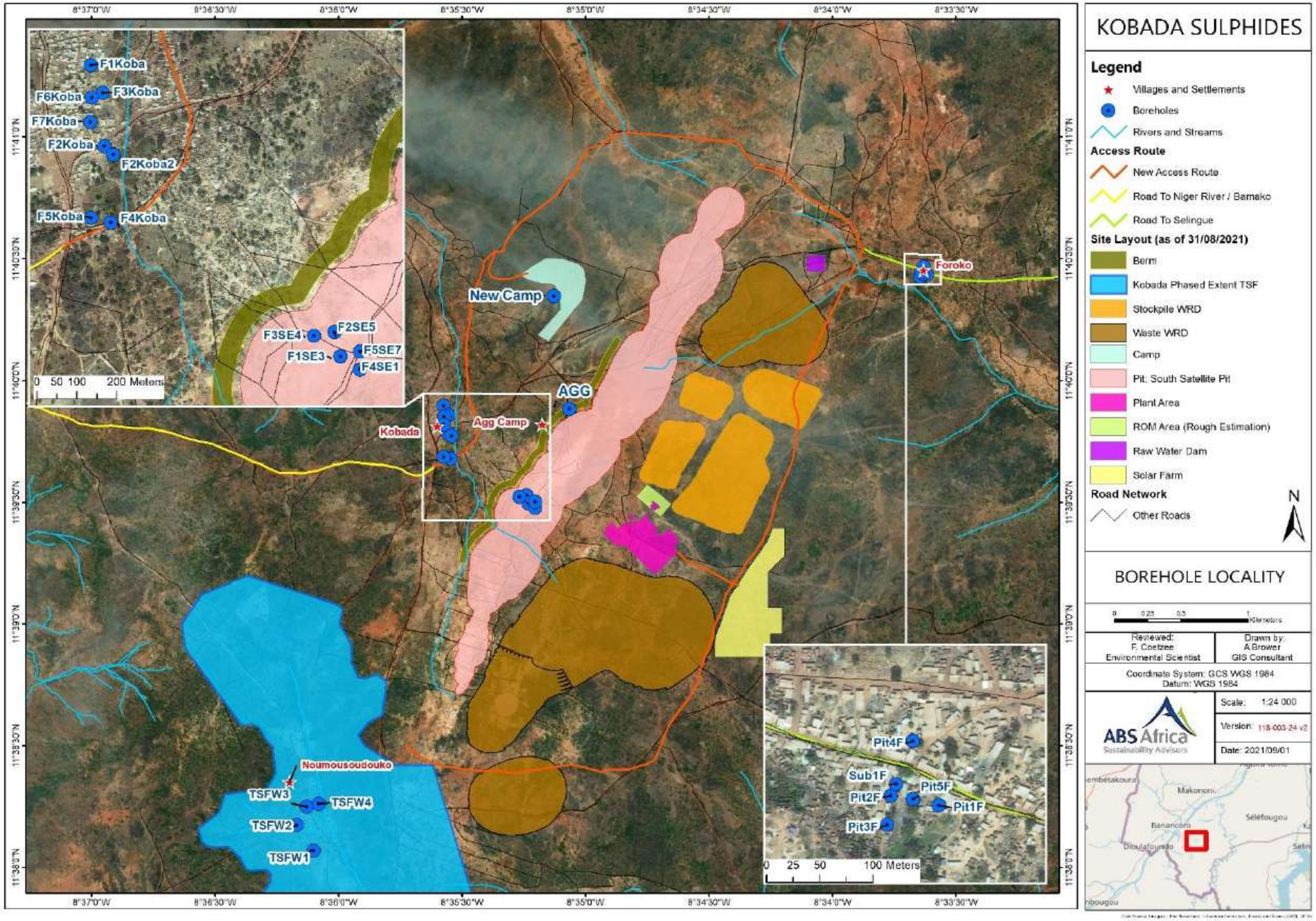


Figure 20.3: Borehole Locality

20.4.1.7 Air Quality

Sensitive receptors in the vicinity of the Kobada Project include the villages of Kobada to the west and Foroko to the north, as well as scattered residences and homesteads to the south and east of the operations.

Current sources of fugitive dust and gaseous emissions include artisanal mining, vehicle activity on unpaved roads, subsistence farming, small-scale livestock farming, wildfires, and wind erosion from exposed areas. Household fuel burning and possibly refuse burning also constitute a significant local source of low-level emissions.

The main fugitive emission sources from the Kobada Project are, in descending order of significance:

- Vehicle entrainment from unpaved haul roads
- Crushing and screening of ROM at the processing plant
- Materials handling (loading and unloading) of waste rock and ROM (in the pits and at the stockpiles and WRDs)
- Generator exhaust emissions
- Wind erosion
- Drilling and blasting
- Vehicle exhaust emissions
- Fugitive gaseous emissions from the processing plant

20.4.1.8 Environmental Noise

A baseline noise survey was undertaken in 2020, the results are summarised below.

Noise receptors (NRs) within a 3 km radius of the Kobada Project include the villages of Kobada to the west and Foroko to the north, as well as scattered residences and homesteads to the south and east of the operations.

Recorded daytime noise levels were below or slightly higher than the IFC guideline for residential receptors at all four sampling locations (see Figure 20.4). The lowest noise levels were recorded at the Camp2 sampling location while slightly higher noise levels were recorded at the other three locations, which are more heavily influenced by community activities. The daytime acoustic climate at all four sampling locations was mainly influenced by vehicle traffic (most notably motorcycles but also cars, light goods vehicles and trucks), artisanal mining activities (in particular generators), community activities (such as music and conversation), and natural noises from domestic animals (such as cattle and goats) and birds and insects.

Recorded night-time noise levels at the three locations where sampling was conducted (no sampling was conducted at the TSF sampling location due to security concerns (damages to the road due to artisanal mining activities etc.)) were below the IFC guideline for residential receptors. The night-time acoustic climate is mostly influenced by artisanal mining activities (including generators) and community activities, particularly music, with limited impact from vehicle traffic.

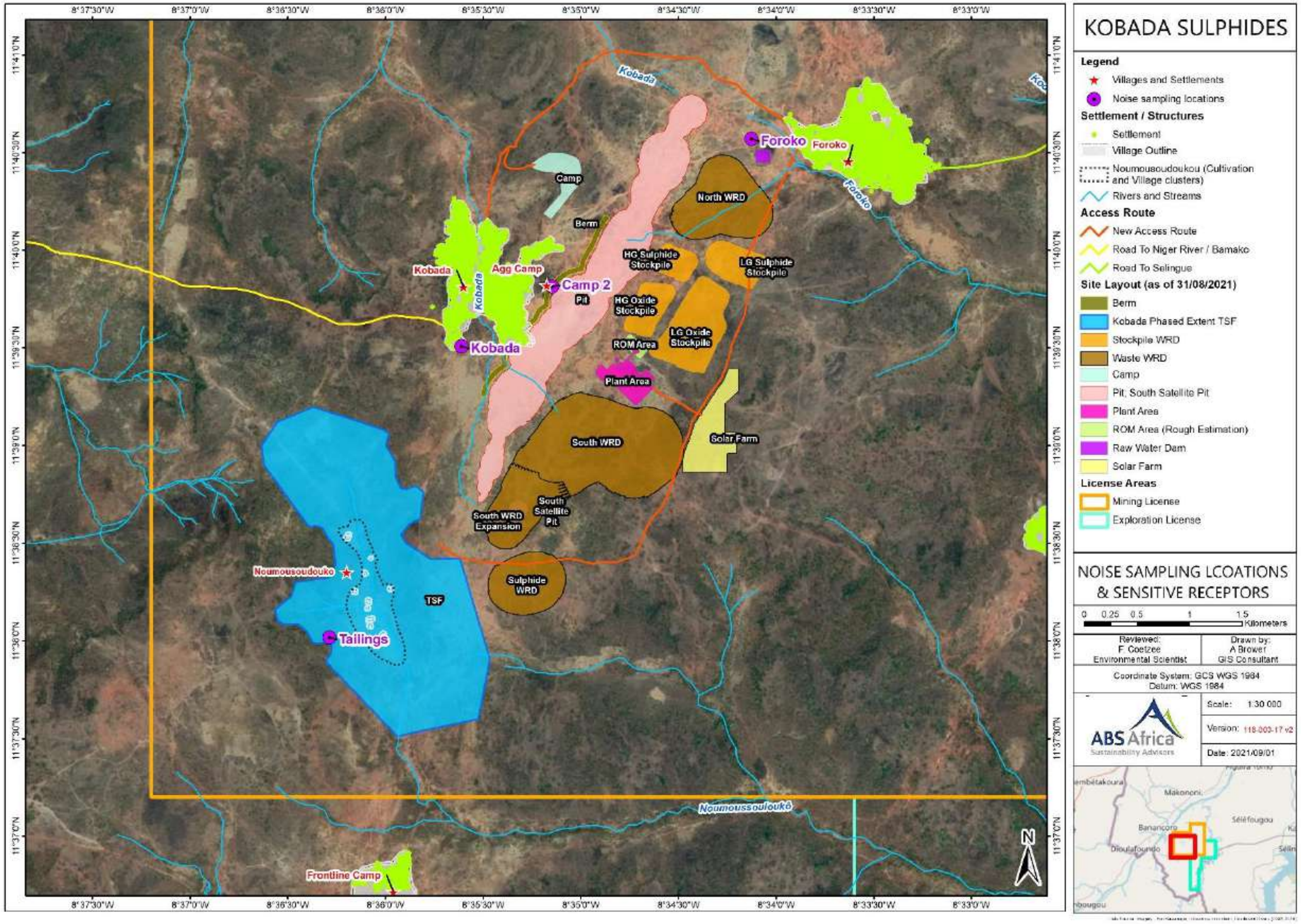


Figure 20.4: Noise Sampling Locations

Logged broadband time series results, frequency spectra and statistics are shown for each sampling location in Table 20.3.

Table 20.3: Baseline Noise Measurement Survey Results

Site	Daytime	Night-Time
	LAeq, d (dBA)	LAeq, n (dBA)
Camp2	50.7	40.8
Foroko	**57.6**	42.7
Kobada	52.2	42.1
Tailings	**55.3**	-
LAeq, d Equivalent continuous sound level, daytime LAeq, n Equivalent continuous sound level, night-time		
NOTE: Levels provided in bold exceed the IFC noise guidelines for residential areas.		

20.4.2 Biological Environment

20.4.2.1 Fauna and Flora

The Project Area falls within the Sudanian regional centre of plant endemism, near the border with the Guinea-Congolia/Sudania Regional Transition Zone. The Project Area vegetation type is characterised as “Sudanian Woodland with abundant *Isoberlinia*”. Dominant woody species in these Sudanian woodlands are *Isoberlinia* species, especially *Isoberlinia doka*, as well as *Afzelia africana, Burkea africana*, *Combretum spp*. and *Terminalia spp*.

Two threatened bird species, four threatened plant species, two near-threatened plant species and one data-deficient plant species were confirmed to occur. A mosaic of natural and modified habitat is present; the natural habitat is represented by seven floristically distinct vegetation associations while the modified habitat is represented by a degraded vegetation association (artisanal mines, villages, cultivated lands). The natural habitat with high biodiversity value in the Project Area includes grassland and wetlands on laterite hardpans, thicket/forest on laterite hardpans, riparian forest, riparian wetlands, and mature broad-leaved woodland, and these are the habitats where the Project impacts could be most significant. However, it is also evident that the Project Area is being rapidly degraded ecologically as a result of a large influx of artisanal miners and the resultant increase in pressure on natural resources. It is important to distinguish the impacts that are as a result of artisanal mining from those that are related to the Project. No critical habitats have been identified in the proposed project area.

20.4.2.2 Aquatic Environment

20.4.2.2.1 Aquatic Ecosystems

The ecological importance of aquatic ecosystems in the Project Area is Very Low, except for seasonal depressions, which have High importance because they support a moderate diversity of aquatic plant species and associated biota, including the barb *Enteromius pobeguini.*

Aquatic ecosystems in the Project Area are classified as “Modified” in terms of IFC (2012). This classification is attributed to the elevated turbidity and sedimentation caused by artisanal mining that has resulted in a low diversity and low abundance of aquatic biota, including aquatic macroinvertebrates and fish.

There are no aquatic habitats in the Project Area that warrant classification as “Critical” in terms of the IFC criteria because the aquatic habitats in the area do not support any of the following:

- Critically Endangered (CR) and/or Endangered (EN) aquatic species
- Endemic and/or restricted-range aquatic species
- Globally significant concentrations of migratory species and/or congregatory species
- Highly threatened and/or unique ecosystems
- Areas associated with key evolutionary processes

Four of the five watercourses visited during the field survey were embedded with fine sediments from artisanal mining and were critically degraded. Instream habitats in these watercourses comprised mostly shallow water (< 30 cm) with a mobile clay substrate and zero to limited abundance of emergent aquatic herbs and associated aquatic biota. These watercourses are not important in terms of aquatic biodiversity. The remaining watercourse within the Project Area, namely E-3 (see Figure 20.5), was not impacted by artisanal mining at the time of the survey. This watercourse runs alongside the access road between the proposed mine and the proposed crossing of the Fié River, and is impacted by historical mining, runoff of sediments from the existing road, and cattle grazing.

Aquatic macroinvertebrates were largely absent from the seasonal foothill streams and associated wetlands, except for a few, hardy taxa. However, moderate abundance and diversity were recorded within the seasonal depression at P1 (see Figure 20.5). Sampling of aquatic macroinvertebrates was not undertaken because of the poor quality of habitats and low abundance of macroinvertebrates. Limited sampling of adult dragonflies was undertaken, and the species recorded comprised widespread, hardy species that are typically found in areas with seasonal availability of surface water.

The risk of waterborne diseases, such as cholera, malaria and bilharzia, within the Project Area is very high because of the abundance of artificial areas of inundation associated with artisanal mining. The period with the highest risk is likely to be towards the beginning of the dry season. The snail *Bulinus cf umbilicatus* was recorded in the Project Area. It is an intermediate host of *Schistosoma haematobium,* which causes urinary bilharzia among humans. No other species of snail were recorded, but bilharzia snails, *Biomphalaria pfeifferi,* are likely to occur in the Project Area. This species is an intermediate host of *Schistosoma mansoni,* which causes intestinal bilharzia among humans.

The abundance and diversity of fish within the Project Area were very low, with only two species of fish recorded within the proposed mining area, and a further three species recorded in the Fié River near the proposed bridge. The low abundance and diversity of fish are attributed to the limited availability of suitable instream habitats, and the disturbance caused by artisanal mining. No fish species of conservational concern are expected within or near the Concession Area.

20.4.2.2.2 Water Quality

The key findings were as follows:

- Water quality was generally similar at all sites sampled.
- Conductivities in the streams flowing eastwards were low (< 7 ms/m) whereas conductivities in the stream flowing northwards were elevated (6.8 ms/m to 19.2 ms/m), and this is attributed to artisanal mining activity.
- Nutrient levels were very low except for the concentrations of nitrates that exceeded the recommended values at sites N1-1 and E2-2 (see Figure 20.5), which is equivalent to a compliance of 80 %. The elevated concentrations of nitrate are attributed to poor sanitation and poor waste management.
- Chemical oxygen demand (COD) was mostly very high and exceeded the recommended values at eight of the ten sites sampled, which is equivalent to a compliance of 20 %. Elevated COD indicates elevated organic matter, and this is attributed to poor sanitation and poor waste management.
- Turbidity was mostly very high and exceeded the recommended values at eight of the ten sites sampled, which is equivalent to a compliance of 20 %. Elevated turbidity is attributed to fines associated with artisanal mining.
- Concentrations of total iron were very high and exceeded various guidelines at nine of the ten sites sampled, which is equivalent to a compliance of 10 %.
- Alkalinity was very low, and this means that the water was poorly buffered and, therefore, vulnerable to change.
- The pH was slightly acidic to circum-neutral and ranged between 6.3 and 6.9, except at N1-1 (see Figure 20.5), where the pH was acidic (5.3).
- The concentration of arsenic was elevated at one site (N1-3).
- The concentrations of sulphate were elevated at two sites but were well within the relevant guideline values.

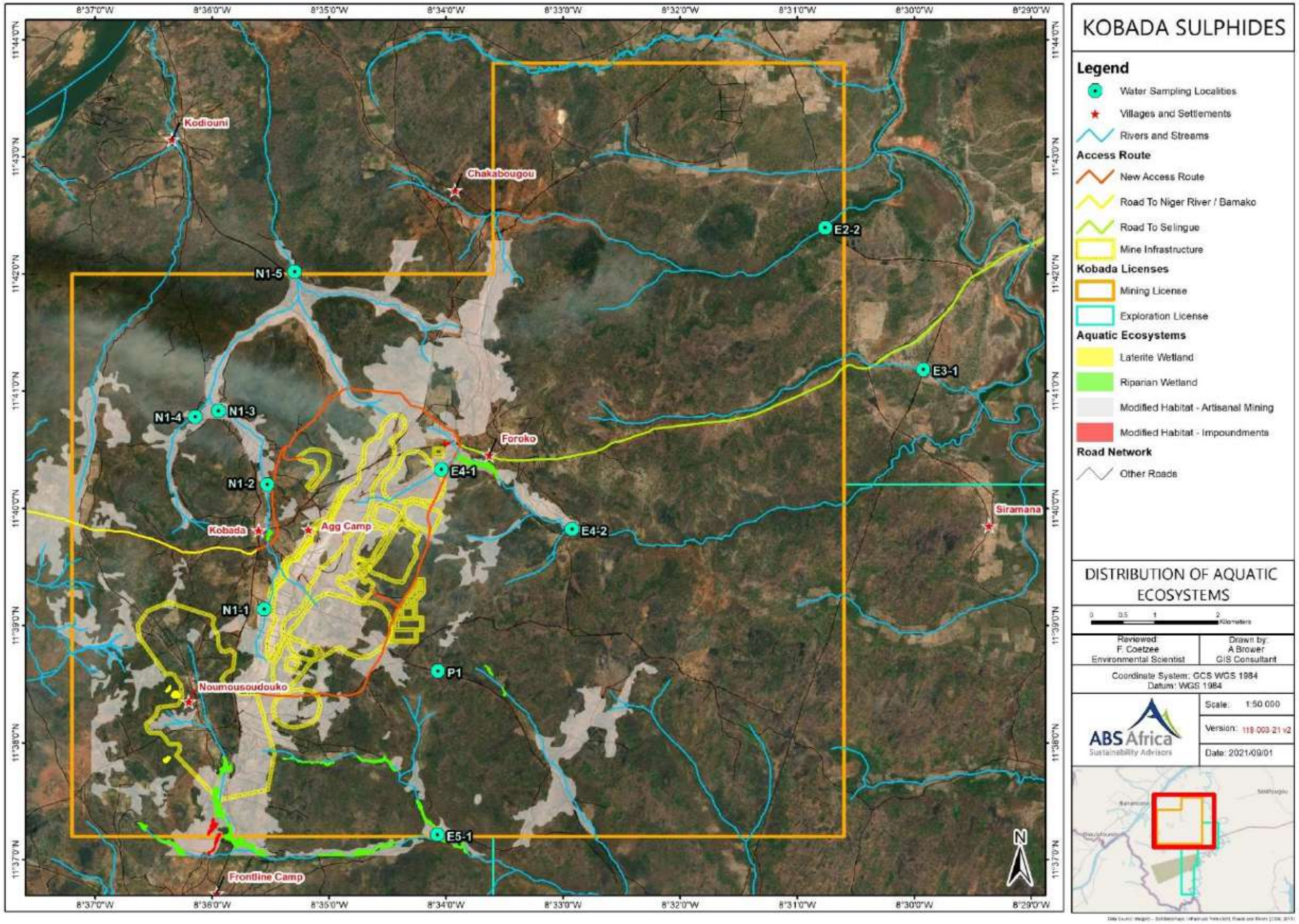


Figure 20.5: Distributions of Aquatic Ecosystems

20.5 SOCIAL BASELINE SETTING

The Kobada Mining Permit affects four communes, namely Nouga, Kaniogo, Tagandougou and Séléfougou, see below Table 20.4 . There are eleven hamlets and sub-hamlets and six “mother” villages registered in the four communes, two circles and two regions, Koulikoro and Sikasso.

Table 20.4: Communes located in the Study Area

<table>
<tr><th>Region</th><th>Circle</th><th>Commune</th><th>Village</th><th>Hamlet</th><th>Sub Hamlet</th></tr>
<tr><td rowspan="11">Koulikoro</td><td rowspan="11">Kangaba</td><td rowspan="5">Nouga</td><td>Banancoro</td><td colspan="2"></td></tr>
<tr><td rowspan="3">Samaya</td><td colspan="2">Kobada</td></tr>
<tr><td colspan="2">Noumoussoudouko</td></tr>
<tr><td colspan="2">Niouleni</td></tr>
<tr><td rowspan="3">Farabacoungo</td><td colspan="2">Foroko</td></tr>
<tr><td rowspan="3">Kaniogo</td><td colspan="2">Kodjouni</td></tr>
<tr><td>FarabaDiaban</td><td>Chakabougou</td></tr>
<tr><td rowspan="3">Séléfougou.</td><td colspan="2">Kola</td></tr>
<tr><td rowspan="3">Séléfougou.</td><td colspan="2">Famorila</td></tr>
<tr><td colspan="2">Hawala</td></tr>
<tr><td>Sanacoro</td><td colspan="2">SanacoroFirda</td></tr>
<tr><td>Sikasso</td><td>Yanfolila</td><td>Tagandougou</td><td>Baya Siekorole</td><td colspan="2"></td></tr>
</table>

The Project Area consists of a mix of ethnicities, potentially due to migration linked to the political and economic difficulties of the country. The increase in gold price and unemployment has attracted more migrants to gold-bearing areas (such as Kobada) to sustain livelihoods.

20.5.1 Main Economic Activities

The main economic activities of the population include agriculture, livestock breeding, fishing, small-scale trade, and traditional gold mining. Approximately 99 % of the population make a living out of these activities. Farmers’ organisations, cooperatives and women’s associations are very active and are mostly funded by public bodies and non-governmental organisations (NGO) such as *Banque Nationale du Mali* (BNDA) and Advocate for Community Development and Environmental Protection (ACODEP).

It should be noted that agriculture is the largest economic activity for households in the Project Area. The main crops include cotton, maize, millet, sorghum, rice, and groundnuts.

Arboriculture of cashew, banana, citrus and mango is practised by a small number of people in the community.

Artisanal mining activities are increasing in Mali and play a significant role in the improvement of livelihoods. More than 52 % of the households surveyed in the Project Area (2020 baseline assessment) practise artisanal mining. They are mostly adult males and mostly Malians, with some artisanal miners from foreign countries such as Niger, Nigeria, Côte d'Ivoire, Burkina Faso and Guinea.

Gold mining in the Project Area is usually done in the dry season and agricultural activities are more prominent in the wet season. Gold mining is traditionally under the responsibility of the Tombolomas[3].

20.5.2 Population

L'Enquête Modulaire et Permanente auprès des Ménages (EMOP) 2018 reported a total population of 19 269 836 people in Mali, of which 74 % live in rural areas.

According to the EMOP (2018), the Kobada Project is located in the two most populated regions of the country: the Sikasso region, which accounts for 18.3 % of the country's total population (3 529 399 inhabitants), and the Koulikoro region, which accounts for 16.6 % of the total population (3 189 937 inhabitants).

During the baseline assessment, 97 households (consisting of 623 people) were surveyed, and it was found that 40 % of the people surveyed were under 15 years of age, 21 % boys and 19 % girls, with a median age of 19 years. Of the remaining 60 %, adult women account for 26 % of the population compared to 34 % for adult men.

Figure 20.6 shows the population surveyed by age and gender.

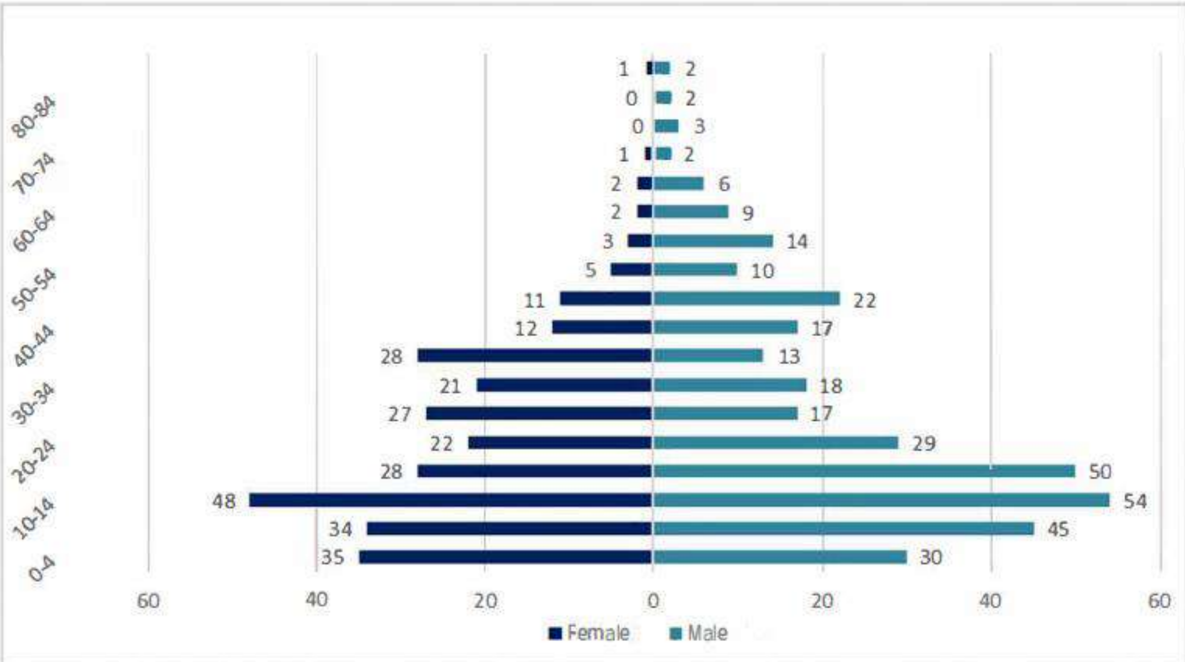


Figure 20.6: Breakdown of the Population Surveyed by Age and Gender

[3] Traditional authority that manages artisanal mining activities. This person (persons) is appointed by the village chief, without limitation of duration.

20.5.3 Household Structures

According to the household survey, 99 % of the occupants own their home with one household occupying their home with a free loan.

Three quarters of these dwellings are on family land and were acquired by inheritance. The dwellings making up the remaining quarter were acquired via a donation.

In the villages and hamlets, all the households surveyed have permanent buildings, mainly mud structures. Most of the buildings have sheet metal roofs (91 %) and cement floors (60 %), and almost a third of the buildings are constructed from cement bricks (29 %).

On average, households in the Project Area have between six and seven members with the head of the household being an average of 49 years of age.

20.5.4 Electricity

Access to electricity in the Project Area via public distribution networks such as Énergie du Mali (EDM) is almost non-existent. Electricity is mostly distributed in the localities of Banancoro and Baya Siekorole (Tagandougou). A hydropower plant is located at the Sélingué Dam.

The main sources of electricity in the Project Area are solar panels, followed by generators; only 3 % of the electricity is supplied by EDM. Nearly 12 % of the households surveyed have no access to any electricity.

Wood is the main source of energy for cooking.

20.5.5 Water Supply

According to the EMOP (2018), the lowest proportion of access to drinking water is recorded in the Koulikoro region, with only 75.7 % of households having access to drinking water. In the Sikasso region, access is higher, with 82.5 % of households having access to drinking water.

In the Project Area, almost two-thirds of the households surveyed (63 %) use “improved” sources of drinking water. These include pump wells, taps and public fountains (11 %). Some dwellings also have a spring or running water.

20.5.6 Community Health and Sanitation

The most commonly listed diseases in the Project Area are malaria and diarrhoea.

The daily use of wood for cooking exposes household members, particularly women and young children, to an increased risk of developing respiratory diseases.

Access to safe water remains a concern in the Project Area, 36 % of the population do not have access to safe drinking water sources and more than half (53 %) do not use any type of filtration method. This increases the risk of disease, such as typhoid fever.

Only 10 % of households have non-shared latrines (connected to sewer system), and 7 % have shared latrines (connected to sewer system). Most households use traditional latrines (82 %) and share them with other households (43 % of households).

The following health service infrastructures have been identified in six villages near the Project Area:

- Each village has a community health care centre.
- A dispensary is available in the villages of Samaya and Faraba Coungo.
- Maternity care is available in the villages of Banancoro, Samaya, Faraba Coungo and Sanancoro.
- A Pharmacy is available in the villages of Banancoro and Samaya.

The community health care centres are not considered to be up to standard and they have limited equipment, particularly due to a lack of funding.

The Kobada village does not currently have a health care centre.

20.5.7 Education

In recent years, education in Mali has received increased investment. Public expenditure relating to the sector increased from FCFA 173.4 billion in 2006 to almost FCFA 304.9 billion in 2015. This constitutes an average increase of 6.5 % per year.

The educational and literacy rates in the Project Area are lower than the Malian average. There is an improvement among the younger generations in Mali. However, school attendance in the Project Area remains low. This is mainly due to lack of infrastructure, the practice of artisanal mining, as well as family farming activities.

There are 19 educational facilities within the six villages, with schools lacking general equipment.

20.5.8 Cultural Heritage Sites

Eleven cultural heritage sites have been identified in the villages of Banancoro, Foroko, Faraba Coungo and Séléfougou (see Table 20.5).

Table 20.5: Cultural Heritage Sites

Locality	Description of the Sites
Banancoro	Jeu et Koni Jetu (sacred woods)
	Miagatin Sira (sacred hill)
	Mènè (sacred pond)
	Dougoutin (sacred forest)
	Dankun (genius residence)
Foroko	Sinsèrè (genius residence)
	Wourou (genius residence)
Faraba Coungo	Sinrèlé (genius residence)
	Dankun (genius residence)
Séléfougou	Big wells or sacred hut
	Sacred forest

20.6 KEY ENVIRONMENTAL AND SOCIAL IMPACTS

20.6.1 Topography

Infrastructures that are likely to affect the topography in the area include the following:

- Overburden and WRDs
- Ore, low-grade and high-grade stockpile for oxides and sulphides respectively
- TSF
- Opencast pits

During the operational phase, opencast operations will commence, resulting in the disposal of waste material from the pit. The above-mentioned facilities will be established near the pit. The pit will develop over the LOM.

The material from the pit that is not backfilled into the pit will remain as waste rock material and will be disposed of in the WRD facilities provided for.

The following features and facilities will remain post closure:

- WRDs
- The satellite pit will be backfilled and only the main pit will be flooded and remain after mining
- TSF

20.6.2 Geology and Landscape Stability

Placement of the WRDs and TSF could result in changes to the stability of the existing landscape and could lead to the creation of geological hazards (i.e. landslides) at the operational and closure stages of both facilities.

Pit wall failure during operations or post closure could pose a safety hazard.

Failure of the TSF is likely to affect structures downstream of the facility. The study allowed for the resettlement of people directly or indirectly affected by the TSF footprint as well as those within the failure Zone of Influence.

20.6.3 Soils, Land Capability and Land Use

The following impacts are considered important as part of the impact-significance study:

- The loss of the soil resource due to the change in land use and the removal of the resource from the existing system. The construction of the facilities will change the land utilisation potential (land capability), resulting in the complete loss of the soil resource for the life of the activity.
- The loss of the soil resource due to the erosion (wind and water) of unprotected soils and the possible contamination of the downstream receptors by sediment and/or dirty materials (reagents, hydrocarbons etc.).
- The loss of the utilisation potential of the soil and land capability due to compaction of areas adjacent to the constructed facilities.

- The loss of the soil resource due to the removal of materials for use in other activities (borrow pits).
- The contamination of the soil resource (both in situ and stored) due to the spillage of hydrocarbons, reagents and raw materials while in transit.
- The contamination of stored or in-situ materials due to dust or emissions fallout from the processing and transporting of the raw materials off site (road trains), and the use of dirty water as an irrigation source and for dust suppression.
- The loss of the soil utilisation potential due to the disturbance of the soils and the potential loss of nutrient and organic carbon stores through infiltration and de-nitrification of the materials by rainfall.
- The loss of the utilisation of the soil resource will impact the land use practice of subsistence agriculture and grazing and the natural wilderness status of the areas that have not already been affected by artisanal mining while the limited traditional hunting and the gathering of medicinal plants will also stop within the mine lease area. These activities are perceived to be of social and to some extent economic benefit to the local people and receiving environments.

20.6.4 Groundwater

The cumulative zone of influence of the groundwater level drawdown cones could extend up to 2.2 km from the pit boundaries. All the wells within the Kobada village are expected to be impacted to some extent by the groundwater level drawdown cone.

Several the surrounding streams fall partially within the zone of influence and will, therefore, be impacted by a reduction in baseflow contribution.

It is assumed that, with proper maintenance of mining vehicles and other operations-related best practices, there will be a limited impact on the groundwater quality from general surface activities. The saprolite will, however, be vulnerable to contamination from the low-and high-grade stockpiles, the ROM pad, as well as the oxides and sulphides WRDs. In addition to this, the aquifer is directly exposed to the mine workings where contamination can impact the groundwater qualities.

All 35 samples evaluated contained high to extremely high arsenic concentrations and mobilisation of the arsenic was significant in most cases, even used deionised water as a leach solution. There is a significant risk that leachate arising from the percolation of water through tailings, waste rock impoundments and ore stockpiles could accumulate arsenic to concentrations more than the Mali standards. Kinetic tests would provide additional data on the rate and extent of arsenic mobilisation from material with a larger particle size. Static test work is known to be conservative and therefore overstate the contamination risk of the material. Specific mitigation measures have been adopted based on the outcome of the geochemical assessment and are discussed in Section 20.7.

Decant from the opencast pit areas, which could start once the water levels reach the lowest elevation of the pit boundaries was evaluated. Decant from the Main Pit is unlikely while decant from the south satellite pit is likely, if not backfilled. The current design allows for the backfill of the satellite pit with oxide material. Decant volumes at the South Satellite pit will be in the order of 20 m^3/day, while decant from the larger Main Pit (in the unlikely event should it occur) can be a maximum of 6 300 m^3/d. It should be noted that decant at the Main Pit is unlikely to be sustainable for a long period, and will only occur directly after a high rainfall event.

The assessment will be updated as part of this ESIA process and an assessment of the TSF decant water quality should be undertaken to establish whether it complies with the IFC PS and Malian requirements. It is recommended that a detailed geochemical pit lake quality study be conducted to calculate the long-term water qualities in the pit.

20.6.5 Surface Water

Hydrocarbons are likely to originate from lubrication bays, wash bays, workshops, and areas where equipment is likely to be serviced and maintained.

A spill of hazardous chemicals and materials due to inappropriate handling and storage is likely to result in significant water contamination in the immediate vicinity of the site if significant quantities are released into the environment. This can result in further secondary impacts on the aquatic environment, which will be difficult to rehabilitate.

Surface water may be contaminated due to the inadequate management and disposal of general waste, including liquid wastes. During the mine operation, an integrated waste management system will be implemented to manage the various waste streams. The laboratory discharges from the washing and rinsing of various utilities will be contaminated with various acids and other chemicals used for testing. The problems associated with waste management may be associated with lack of waste sorting at source, poor disposal practices, lack of containment facilities, poor land filling practices, poor containment of liquid waste, spillages, and other poor management practices. Impacts associated with poor waste management could endanger the health of mine workers who operate in an unhygienic and polluted environment in addition to the contamination of water sources, deterioration of air quality and nuisances caused by malodour and loss of aesthetics. The use of portable latrines can cause malodour and hence become a nuisance to workers if not properly managed.

Even where efficient disposal methods are used, monitoring is essential to minimise the chances of contamination of groundwater sources through seepage of buried waste. With appropriate mitigation measures, the impact is expected to be low medium for the duration of the Project.

The potential for decant from the opencast pit areas were evaluated and could start once the water levels reach the lowest elevation of the pit boundaries. Decant from the Main Pit is unlikely while decant from the south satellite pit is likely. Decant volumes at the South Satellite pit will be in the order of 20 m^3/d, while decant from the larger Main Pit (in the unlikely event should it occur) can be a maximum of 6 300 m^3/d. It should be noted that decant at the Main Pit is unlikely to be sustainable for a long period and will only occur directly after a high rainfall event.

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Preliminary results obtained for the sulphides ESIA shows the sulphide ore and sulphide waste rock material would have to be separated from the oxides and managed separately. This would mean that the oxides and sulphides have separate proposed WRD, stockpiles and PCD's.

20.6.6 Air Quality

Sensitive receptors in the vicinity of the Kobada Project include the villages of Kobada to the west and Foroko to the north, as well as scattered residences and homesteads to the south and east of the operations. All identified sensitive receptor locations are shown in Figure 5 6. Because management measures will need to be implemented to minimise the impact of the Kobada Project on ambient air quality at these nearby receptors, the impact at more distant receptors is expected to be insignificant.

Due to the proximity of the mining operations to sensitive receptor locations, unmitigated particulate emissions would have a catastrophic impact on the receiving environment, with particulate concentrations at sensitive receptors with orders of magnitude higher than the WHO Air Quality Guideline (AQG).

The interim targets (IT) are intended as incremental steps in a progressive reduction of air pollution in more polluted areas; they are intended to promote a shift from concentrations involving acute, serious health consequences to concentrations that, if achieved, would result in significant reductions in the risk of acute and chronic effects (WHO, 2005). Such progress towards guideline values should be the objective of air quality management and health risk reduction in all areas and are summarised in Table 20.6.

Table 20.6: WHO Air Quality Guideline Values

Mean Level	PM_{10} (µg/m³)	$PM_{2.5}$ (µg/m³)	Basis for the selected level
Annual Mean Level			
WHO interim target – 1 (IT-1)	70	35	These levels were estimated to be associated with about 15 % higher long-term mortality than at AQG
WHO interim target – 2 (IT-2)	50	25	In addition to other health benefits, these levels lower risk of premature mortality by approximately 6 % (2 % to 11 %) compared to WHO-IT1
WHO interim target – 3 (IT -3)	30	15	In addition to other health benefits, these levels reduce mortality risks by another approximately 6 % (2 % to 11 %) compared to WHO-IT2 levels.
WHO AQG	20	10	These are the lowest levels at which total, cardiopulmonary and lung cancer mortality have been shown to increase with more than 95 % confidence in response to $PM_{2.5}$ in the American Cancer Society (ACS) study (Pope et al., 2002 as cited in WHO 2006). The use of the $PM_{2.5}$ guideline is preferred.
24-Hour Mean Level*			
WHO interim target – 1 (IT-1)	150	75	Based on published risk coefficients from multi-centre studies and meta-analyses (about 5 % increase of short-term mortality over AQG)

Mean Level	PM_{10} (µg/m³)	$PM_{2.5}$ (µg/m³)	Basis for the selected level
WHO interim target – 2 (IT-2)	100	50	Based on published risk coefficients from multi-centre studies and meta-analyses (about 2.5 % increase of short-term mortality over AQG)
WHO interim target – 3 (IT -3)**	75	37.5	Based on published risk coefficients from multi-centre studies and meta-analyses (about 1.2 % increase of short-term mortality over AQG)
WHO AQG	50	25	Based on relation between 24-hour and annual levels
* 99th percentile (3 days/year) ** for management purposes, based on annual average guideline values; precise number to be determined on basis of local frequency distribution of daily means			
Source: WHO (2005)			

Simulated annual average particulate matter PM_{10} concentrations exceed the WHO AQG (20 µg/m³) up to 300 m from the mining operations, and the IT-3 (see Table 20.7) level (30 µg/m³) up to 200 m from the mining operations. The IT-2 and IT-1 levels are only exceeded in the immediate vicinity of the mining operations.

Simulated annual average $PM_{2.5}$ concentrations are below the WHO AQG at all the sensitive receptor locations.

Simulated annual average NO_2 concentrations are below the WHO AQG at all the sensitive receptor locations. Simulated CO concentrations are well below (< 10 %) all international standards and guidelines.

Simulated SO_2 concentrations are in compliance with the WHO IT-1 level, as well as all other international standards and guidelines, at all the sensitive receptor locations for all averaging periods.

Simulated dust fallout is localised to the vicinity of the mining, hauling and processing operations, with dust fallouts of higher than 100 mg/m²/d occurring in the eastern part of Kobada (up to 300 m from the mining operations).

It is recommended that a 350 m buffer zone be implemented between the mining operations and any sensitive receptor locations to protect the health of the residents of nearby villages. For the purposes of this study a 500 m buffer zone was adopted.

The ambient air quality guidelines for the various international organisations as accepted by the world bank are provided in Table 20.7.

Table 20.7: Ambient Air Quality Guidelines for Various International Organisations as Accepted by the World Bank

Pollutant	Averaging Period	WHO Guideline Value (µg/m³)	EC Directive Limits (µg/m³)	United States NAAQS (µg/m³)	South African NAAQS (µg/m³)
Sulphur Dioxide (SO_2)	1 year	–	20[b]		50
	24 h	125 (IT-1)	125[c]		125[m]
	–	50 (IT-2)[a]	–	–	–
	–	20 (guideline)	–	–	–
	1 h	–	350[d]	196[h]	350[n]
	10 min	500 (guideline)	–		500[o]
Nitrogen Dioxide (NO_2)	1 year	40 (guideline)	40[e]	100[p]	40
	1 h	200 (guideline)	200[f]	188[j]	200[n]
Carbon Monoxide	8 h	10 000	10 000	9 000	10 000
Particulate Matter (PM_{10})	1 year	70 (IT-1)	40[g]	150[k]	40[m, i]
	–	50 (IT-2)	–	–	–
	–	30 (IT-3)	–	–	–
	–	20 (guideline)	–	–	–
	24 h	150 (IT-1)	50[h]		75
	–	100 (IT-2)	–	–	–
	–	75 (IT-3)	–	–	–
	–	50 (guideline)	–	–	–
Particulate Matter ($PM_{2.5}$)	1 year	37.5 (IT-1)	–	12[l,q]	40[m]
	–	25 (IT-2)	–	–	–
	–	15 (IT-3)	–	–	–
	–	10 (guideline)	–	–	–
	24 h	75 (IT-1)	25[i]	35[j,q]	75
	–	50 (IT-2)	–	–	–
	–	37.5 (IT-3)	–	–	–
	–	25 (guideline)	–	–	–

NOTES:

[a] Intermediate goal based on controlling motor vehicle emissions, industrial emissions and/or emissions from power production. This would be a reasonable and feasible goal to be achieved within a few years for some developing countries and lead to significant health improvement.

[b] EC First Daughter Directive, 1999/30/EC (http://europa.eu.int/comm/environment/air/ambient.htm). Limit value to protect ecosystems. Applicable two years from entry into force of the Air Quality Framework Directive 96/62/EC.

[c] EC First Daughter Directive, 1999/30/EC (http://europa.eu.int/comm/environment/air/ambient.htm). Limit value to protect health, to be complied with by 1 January 2005 (not to be exceeded more than 3 times per calendar year).

[d] EC First Daughter Directive, 1999/30/EC https://ec.europa.eu/environment/air/quality/existing_leg. Limit value to protect health, to be complied with by 1 January 2005 (not to be exceeded more than 24 times per calendar year).

[e] EC First Daughter Directive, 1999/30/EC (http://europa.eu.int/comm/environment/air/ambient.htm). Annual limit value for the protection of human health.

[f] EC First Daughter Directive, 1999/30/EC (http://europa.eu.int/comm/environment/air/ambient.htm). Not to be exceeded more than 18 times per year.

Pollutant	Averaging Period	WHO Guideline Value (µg/m³)	EC Directive Limits (µg/m³)	United States NAAQS (µg/m³)	South African NAAQS (µg/m³)
[g] US NAAQS (www.epa.gov/air/criteria.html). Not to be exceeded more than once per year on average over three years. [h] US NAAQS (www.epa.gov/air/criteria.html). 99th percentile of 1 h daily maximum concentrations, averaged over three years. [i] EC Directive, 2008/50/EC (http://ec.europa.eu/environment/air/quality/legislation/directive.htm). Not to be exceeded more than 35 times per calendar year. [j] US NAAQS (www.epa.gov/air/criteria.html). 98th percentile, averaged over three years. [k] US NAAQS (www.epa.gov/air/criteria.html). Not to be exceeded more than once per year on average over three years. [l] US NAAQS (www.epa.gov/air/criteria.html). Annual mean, averaged over three years. [m] A total of 4 permissible frequencies of exceedance per year. [n] A total of 88 permissible frequencies of exceedance per year. [o] A total of 526 permissible frequencies of exceedance per year. [p] US National Ambient Air Quality Standards (NAAQS) (www.epa.gov/air/criteria.html). 99th percentile of 1 h daily maximum concentrations, averaged over three years. [q] US NAAQS (www.epa.gov/air/criteria.html). 98th percentile, averaged over three years.					
Source: Airshed Planning Professionals (2020)					

Kobada Sulphides Air Quality Assessment

Based on information received regarding project layout, major equipment, support equipment and service equipment for the sulphides project at the Kobada Gold Mine, it is not anticipated that air quality impacts due to the sulphides project will differ significantly from those assessed for the oxides project, given that mining rates remain the same.

20.6.7 Greenhouse Gas Emissions (GHG) Inventory for The Kobada Project

A greenhouse gas emissions inventory was compiled for the Kobada Project using provided estimated fuel consumption rates and Intergovernmental Panel on Climate Change (IPCC) emissions factors for diesel combustion by non-road diesel combustion sources.

Emission rates are calculated in tonnes of CO2 equivalent per annum, taking into account the 100-year global warming potential of each of the released pollutants. Estimated annual GHG emission rates are shown for the year with the highest expected fuel usage rates (see Table 20.8 – Year 8) as well as for average fuel usage over the 16-year life of mine (see Table 20.9).

Table 20.8: Estimated GHG Emission Rates – Maximum Annual Emissions – Year 8

Aspect	Fuel Usage	CO2	CH4	N2O	Total
	L/a	tonne CO2eq/a	tonne CO2eq/a	tonne CO2eq/a	tonne CO2eq/a
Mining	18,694,328.0	53,470.6	45.5	894.8	54,410.9
Power Plant - Diesel	2,066,725.0	5,911.4	5.0	14.8	5,931.2
Power Plant - HFO	7,706,218.0	24,932.1	20.3	59.9	25,012.3
Process Plant	400,000.0	1,144.1	1.0	2.9	1,147.9
Miscellaneous	300,000.0	858.1	0.7	14.4	873.2
Total		86,316.3	72.5	986.8	87,375.5

Table 20.9: Estimated GHG Emission Rates – Average Annual Emissions – Year 1 to 16

Aspect	Fuel Usage	CO2	CH4	N2O	Total
	L/a	tonne CO2eq/a	tonne CO2eq/a	tonne CO2eq/a	tonne CO2eq/a
Mining	9,963,815.9	28,499.1	24.2	476.9	29,000.2
Power Plant – Diesel	1,621,719.4	4,638.5	3.9	11.6	4,654.1
Power Plant – HFO	6,597,961.6	21,346.5	17.4	51.3	21,415.2
Process Plant	400,000.0	1,144.1	1.0	2.9	1,147.9
Miscellaneous	300,000.0	858.1	0.7	14.4	873.2
Total		56,486.3	47.3	557.1	57,090.7

20.6.8 Blasting and Vibration

The proposed sulphide mining has the highest negative impact on the south-eastern side of Kobada with a much lower impact on Foroko. Although the document focusses on Kobada Town, there is a possibility that unidentified dwellings may be scattered throughout the area that could be impacted. The recommendations in this report apply also to those dwellings.

Vibration is not likely to have any serious impact on the community for properties that are further than 100 m for 5 m bench blasting and 150 m for 10 m bench blasting. A 500 m buffer will be implemented between the pit and structures/people within the blasting area. With further testing and monitoring it is anticipated that the buffer can be reduced to 350 m without affecting the safety of the community.

20.6.9 Noise

A baseline noise survey was undertaken in 2020 for the Kobada oxides ESIA, the results are summarised below.

Noise receptors (NRs) within a 3 km radius of the Kobada Project include the villages of Kobada to the west and Foroko to the north, as well as scattered residences and homesteads to the south and east of the operations.

The simulated equivalent continuous daytime rating level (LReq,d) of 55 dBA extends ~500 m from the mining operations. Levels in excess of 70 dBA are only simulated in the immediate vicinity of the operations (i.e. inside the pits and plant, and on top of the stockpiles). Because baseline noise levels in Kobada and Foroko are already a little higher than background noise levels in other areas, the mining, hauling and processing operations are expected to have only a slight impact during the day, with a simulated increase of 3 dBA only at NRs closest to the mining operations. Based on simulated daytime noise levels and the SANS 10103 guidelines, few or sporadic complaints can be expected due to daytime activities.

Conversely, due to the low baseline night-time environmental noise levels at NRs (as sampled during the February 2020 sampling campaign), the incremental increase in night-time noise levels is expected to be significant, with a simulated increase more than 15 dBA at the closest NRs, and an increase of > 10 dBA at NRs up to ~300 m and an increase of 5 dBA up to 800 m from the opencast mining operations. An increase of 3 dBA in sound pressure levels is simulated up to 1 km from the operations, including for most of Kobada and the western part of Foroko. For a person with average hearing acuity, an increase of less than 3 dBA in the general ambient noise level is not detectable.

Based on the measured low baseline night-time environmental noise levels, as well as the simulated increase in night-time noise levels, sporadic complaints and medium community action can be expected from most of Kobada and Foroko, but more widespread complaints and stronger community action can be expected from the NRs in the southeast of Kobada.

While a 350 m buffer zone is considered adequate, a 500 m buffer will be implemented from mining operations to the closest sensitive receptors, and the mitigation measures given in the EMP should be considered to further reduce the noise impact on sensitive receptors.

Kobada Sulphides Noise Assessment

Based on information received regarding project layout, major equipment, support equipment and service equipment for the sulphides project at the Kobada Gold Mine, it is not anticipated that environmental noise impacts due to the sulphides project will differ significantly from those assessed for the oxides project, given that mining rates remain the same.

20.6.10 Terrestrial Ecology

The following key impacts on flora have been identified:

- Loss of natural habitat of high biodiversity value;
- Loss of conservation-important plant species;
- Loss of medicinal plant species;
- Increased utilisation of plant resources as a result of an influx of people into the Project Area;
- Introduction/proliferation of alien invasive species; and
- Ecosystem degradation and loss of ecosystem services.

The following key impacts on fauna have been identified:

- Loss of faunal habitat;
- Disturbance/loss of fauna species due to construction and operational activities;
- Introduction/invasion of alien fauna and spread of diseases;
- Increased utilisation of animal resources as a result of an influx of people into the Project Area; and
- Habitat fragmentation and associated reduced/disrupted ecosystem functioning.

Mitigation measures that must be implemented to reduce the above-mentioned impacts include the following:

- Infrastructure should be located away from the Natural Habitat wherever possible, and progressive rehabilitation and reinstatement of disturbed areas should take place;
- Continuous environmental awareness raising, and training should aim at minimising the impacts on fauna; and

- Strict implementation of speed control measures, as well as warnings at animal crossing and along all access roads, could reduce the risk of incidents.

Controlling the establishment of alien fauna species is challenging but the key to managing these indirect impacts and ensuring that they do not become a significant issue lies in continuous good housekeeping and waste management.

An assessment of potential direct and indirect impacts of the project on terrestrial ecology has identified three impacts of High significance (Loss of Natural Habitat of High Biodiversity Value, Loss of Conservation-Important Plant Species, Introduction/proliferation of alien invasive species). Mitigations have been provided that reduce these impacts to Medium or Medium-Low significance. Some of these mitigation measures have already been applied prior to report submission, most notably the relocation of the Solar Farm from Grassland on laterite to degraded woodland.

20.6.11 Aquatic Ecosystem

Generally, the impact on the aquatic ecosystems is low, assuming that mitigation measures are implemented.

Stream crossings would require the construction of causeways and/or culverts, and these could have long-term, localised negative impacts on riparian and wetland habitat quality. Stream crossings that are poorly designed can disrupt the natural movement of instream and riparian biota along watercourses, and in doing so, fragment aquatic ecosystems.

The potential impacts of the proposed development on aquatic ecosystems after mitigation were classified as follows:

- Low significance:
  - Elevated sediments;
  - Habitat disturbance;
  - Stream crossings; and
  - Altered flows, particularly dewatering.
- Low potential for occurrence if properly mitigated:
  - Altered water quality (domestic, industrial, dewatering, TSF seepage); and
  - TSF failure.

The following measures are recommended to avoid or reduce potential impacts on aquatic ecosystems:

- Control sediments;
- Control erosion at storm water discharge points;
- Utilise buffer zones to protect aquatic ecosystems, where feasible;
- Design and maintain stream crossings;
- Control decanting (post operation);
- Manage surface water quality; and
- Monitor TSF.

20.6.12 Socio-Economic

The following potential social and economic impacts have been identified as part of the assessment:

- Migration and urbanisation;
- Increase in social tension/conflict;
- Economic displacement;
- Resettlement;
- Loss of gold washing areas in the Kobada area: The impact is considered significant, especially in the areas directly affected by the mine infrastructure and associated facilities. While the artisanal miners do not have a legal right to mine in the Kobada area, the development of the pit and associated WRDs will prevent future access to the current workings. It is anticipated that the majority of the artisanal miners will relocate to nearby areas not currently earmarked for development;
- Loss of agricultural resources: Along the haul route, some cultivated fields and plantations may be affected during the construction phase. The impact is considered limited, provided that mitigation measures are implemented;
- Degradation of conditions of access to natural resources;
- Inequalities in socio-economic structures;
- New economic opportunities;
- Diseases and the risk of transmission of human immunodeficiency virus, acquired immunodeficiency syndrome (HIV/AIDS);
- Opportunities available via the Community Development Plan (CDP); and
- Erosion of the traditional culture and loss of identity.

20.6.12.1 Employment and Economy

The primary socio-economic impacts associated with the Project will be positive in that residents of the Project Area and the region will be offered employment opportunities during the construction and operational phase. This will translate into an improved standard of living for those hired and their families.

National, regional and local businesses and contractors will benefit both directly and indirectly from Project-related construction and operational activities due to the purchase of goods and services. Increased incomes and profits of local businesses and major suppliers of goods and services manufactured and supplied in Mali will be realised. This increased revenue to local businesses will in turn contribute positively to the local economy through expanded local employment, increased disposable incomes, and growth in the local consumer base.

20.6.12.2 Training and Education

Project development has the potential to provide increased availability and opportunity for a wide range of skills development and job training, particularly for women and local youth. Job opportunities made available to women could increase the health and well-being of families in general. The CDP may improve local education through direct support/improvements made to school facilities and may also positively impact school enrolment, attendance and completion due to increased household incomes (for school fees) and student anticipation of future employment.

20.6.12.3 Population Influx, Inflation, and Increase in Crime

The social and economic pressures of population growth will continue as additional people move to the area to find employment and require accommodation and community services (particularly during construction). The influx of people in search of jobs associated with Project development is expected to be limited if mitigation measures are implemented.

20.6.12.4 Disadvantaged and Vulnerable Groups

Within the Project Area, the most marginalised or vulnerable groups will be those whose access to the land has been impacted, as well as those who have to purchase a large percentage of their foodstuffs and rent accommodation (e.g., civil servants, pensioners, and the poor). Some women, for whom access is dependent upon a husband or male family member owning land, could also be classified as vulnerable. In addition, where men will no longer be involved in the family's agricultural activities because of mine employment, food security could be threatened, increasing the vulnerability of this group. Youth who will not gain compensation for the loss of land but will not have future access to the land that is their inheritance will also be considered a vulnerable group.

20.6.13 Resettlement Planning and Implementation

The Project requires the partial resettlement of the Kobada village due to the placement of the Project infrastructure and to ensure a safe buffer zone between the mining activities and the community. Based on the mitigation measures adopted for the Project, a minimum buffer zone of 350 m is recommended. For the purposes of the study a 500 m buffer zone was adopted. Approximately 2409 structures and temporary structures are located within the zone of influence.

The development will also result in the economic displacement and compensation of various farmers in the area and on farms carrying subsistence and cash crops due to the establishment of the mining infrastructure and associated activities.

Several migrants moved into the Project Area after the current exploration and mining permits were issued and established settlements, likely without permission from the local and traditional leaders or permission from the mining permit holder. However, this would need to be confirmed during the social impact assessment and public consultation process.

Resettlement and compensation will be undertaken in terms of the relevant Malian legislation, which includes the following key points:

- Determining eligibility for compensation and resettlement assistance, including livelihood initiatives;
- Considering approaches to land access and management;
- Establishing rates of compensation; and
- Establishing mechanisms to resolve grievances among affected people related to compensation and eligibility.

International best practice guidelines, which include the IFC's Performance Standards and the World Bank Group's Operational Policies, have also been taken into consideration and will be continued during the detailed planning and implementation phases.

The resettlement process will be aligned with the legislative requirements of Mali as well as international best practice guidelines as described earlier. In summary, the process can be outlined as follows:

- Stakeholder engagement;
- Declaration of the moratorium;
- Rapid Asset Survey (RAS);
- RAP;
- Negotiations and development of agreement with affected parties; and
- Physical construction.

The partial resettlement of Kobada individuals/households is limited to the houses and associated infrastructure located within the prescribed buffer from the edge of the final open pit.

There will be a zone of influence around the non-mining-related infrastructure, such as the plant, WRDs, TSF and associated infrastructure, and it is assumed that no activities will be allowed within 100 m of these facilities.

20.6.14 Government Revenue and Royalties

During all the phases of the Project, payment of dividends, tax on taxable income, royalties and surface rent will contribute to the fiscus. These contributions are expected to improve the financial capacity of the government to improve community infrastructure and service delivery in the district. The rates that apply include the royalty rate of 3 % of the net revenue and a corporate tax rate of 30 %.

AGG will also contribute to community development as per the CDP that will last beyond the life of the Project.

20.6.15 Community Development

Several consultations with communities in the project area have been undertaken to date.

Consultations with the Nouga and Kaniogo Municipalities were undertaken in 2015 by AGG to develop a CDP. The development focus areas that were identified during these consultations are summarised in Table 20.10, and AGG should build on these needs within the affected communities. The community requirements will be further updated once the public participation process for the Kobada project has been completed.

The company will target development projects that are self-sustainable where possible, so that these projects can continue long after the mine has closed. A revised CDP will be generated after consultation with the local village heads and a priority list will be drawn up and actioned.

Table 20.10: Community Development Focus Areas

No.	Areas	Specific Goals	Activities To Be Carried Out
1	Education	Increase/upgrade school and educational infrastructure	Construct classrooms and latrine blocks.
			Strengthen the Capacity Development for Education (CapEFA).
			Provide schools with school equipment and supplies.
			Construct and maintain educational centres.
			Strengthen the technical capabilities of facilitators.
			Provide financial support for school management committees to contract teachers.
			Develop school libraries.
2	Health, Hygiene/ Sanitation	Improve the working conditions of health workers	Construct or improve a health care centre.
			Recruit qualified health workers.
			Ensure adequate equipment and training for health care centre.
		Increase access to safe drinking water	Establish manual pumps.
			Train maintenance workers to repair pumps.
		Improve conditions of hygiene and sanitation	Organise sessions and debates on the importance of good hygiene practices.
3	Agriculture	Ensure food security	Train management committees for grain banks.
			Provide material and financial support to the various grain banks in the Project Area.
			Conduct periodic monitoring of the management of grain banks.
		Diversify people's sources of income	Construct market gardening perimeters in the villages.
			Provide equipment for market gardening and production.
			Train the market gardeners on cultivation techniques.
			Provide financial and material support to women's organisations for the feeding and breeding of sheep/goats.
4	Forest	Reduce the destruction of forests for wood	Create community nurseries.
			Provide focus and training for reforestation.
			Regulate the excessive use of wood.
			Provide training on how to reduce the potential of soil erosion.
5	Employment and local businesses	Develop and promote local businesses and employment	Promote local employment and local supply of goods and services.
6	Public Buildings/ Communication	Unlock and facilitate communication and equip communities in the Project Area with basic development infrastructure	Construct a paved road linking the Project Area to the city of Sélingué.
			Construct shops in the Project Area.
			Facilitate effective telephone network coverage in the area (Orange, Malitel, etc.)
			Provide material and financial support to local FM radio stations in the Project Area.

No.	Areas	Specific Goals	Activities To Be Carried Out
			Support sports and competitions in the communities in the Project Area
			Provide support for the development of football pitches and provide sports equipment to the youth of the municipalities in the Project Area.
7	Conflict Prevention/ Management and Transition	Conflict prevention and management	Develop an agreement for the prevention and management of conflicts in the municipalities in the Project Area.
			Strengthen the framework on pasture management for consultation between farmers, communities, conservationists and agricultural workers.
			Support the allocation of passageways for animals in the Project Area.

Community concerns also documented during the 2020 and 2021 public consultations. The main points identified during this round of consultations are summarised as follows:

- Community infrastructure and access to basic services, especially water;
- Employment opportunities for youth and woman;
- Protection and conservation of cultural heritage; and
- Support for agriculture and compensation for those who lose their land.

20.7 ENVIRONMENTAL AND SOCIAL MANAGEMENT PLAN

In order to achieve the appropriate environmental management standards and ensure that the findings of the environmental studies are implemented through practical measures, the recommendations from the ESIA have been used to compile an Environmental and Social Management Plan (ESMP). The role of the ESMP is to assist AGG in reducing potential impacts and risks and achieving its environmental objectives as well as fulfilling its commitment to the environment. The ESMP will be used to ensure compliance with environmental specifications, monitoring and management measures. The ESMP will be implemented from site preparation through to decommissioning and closure.

With the appropriate mitigation measures implemented, impacts associated with the Project can be mitigated to an acceptable level. It is noted that social impacts relating to the procurement of local goods and services, and access and mobility are considered to be positive impacts.

Some of the key mitigation and management measures incorporated into the design and EMP of the project include:

- Inclusion of an arsenic removal process in the plant.
- Ongoing kinetic test work to confirm the risk of the sulphide material.
- Removal of CN to ensure compliance with the International Cyanide Management Code.
- Lining of the TSF and aligning the design with the current best practice guidelines.
- Separating of the oxide and sulphide ore and waste streams.

- Allowing for an impermeable clay liner for the sulphide waste as well as the sulphide ore stockpiles.
- Establishing lined pollution control dams for the sulphide ore and plant area.
- Prioritising the reuse of TSF return water as well as the PCD and pit seeps at the plant.
- Separating clean and dirty water and establishing storm water control infrastructure.
- Installation of a noise and safety berm between the pit and the Kobada village.
- Partial Resettlement of the Kobada Village.
- Implementation of a Community Development Plan.

20.8 STORM WATER MANAGEMENT

A stormwater management plan was developed as part of the oxides DFS and will be updated to address the sulphide project layout. There are expected to be no material changes to the designs.

20.9 MONITORING

Environmental monitoring is intended to provide constant feedback on the effectiveness of mitigation measures identified in and implemented through the various ESMPs. The monitoring plans define the actions required for the continuous tracking of these ESMPs and will allow for the identification of any problems encountered whilst providing the opportunity to adjust and mitigate ESMPs and monitoring plans accordingly. The various monitoring plans have been based directly on the ESMPs.

AGG will develop detailed procedures for each of these monitoring plans and will ensure that monitoring reports are prepared and reported internally, on a monthly basis, and externally, on an annual basis.

20.10 ONGOING AND FUTURE STUDIES

The following work and studies are ongoing or planned prior to the development of the project:

- Update of the ESIA and EMP and the subsequent submission of an amendment to the DNACPN for approval to ensure that the environmental permit is aligned with the Sulphide DFS project description.
- Kinetic geochemical testing on selected samples to confirm the environmental risk that the oxide and sulphide material is likely to pose to the environment.
- Develop a geochemical model to assess the contamination risk of the waste and ore streams as well as the post closure pit lake water quality.
- Update the storm water management plan and updating the design of the storm water management infrastructure.
- Development of a Resettlement Action Plan
- Establish environmental boreholes and undertaking test pumping to confirm the aquifer characteristics used in the geohydrology assessment.
- Updating the geohydrology assessment to assess the effectiveness of the out of pit dewatering holes.
- Development of various IFC Action Plans, including
    - Air Quality Management Plan

  - Noise and Vibration Management Plan
  - Emergency Preparedness and Response Plan
  - Cyanide Management Plan
  - Transportation Management Plan
  - Soil and Land Management Plan
  - Integrated Water and Waste Management Plan
  - Aesthetics and Visual Impacts Management Plan
  - Soil Management Plan

Additional plans and actions to be developed prior to the implementation of the project include:

- Develop an Environmental and Social Management System
- GHG Tracking and Monitoring Plan
- Climate Change Risk Assessment
- Biodiversity Management Plan and Strategy
- Ecological Rehabilitation Plan
- Ecosystem Services Assessment
- Biodiversity Action Plan
- Biodiversity Monitoring and Evaluation Plan
- Invasive Alien Species Management Plan
- Updating and amending the Community Development Plan
- Air Quality Monitoring
- Artisanal Miner Study
- Ongoing Water Quality Monitoring of surface and ground water resources.

21 CAPITAL AND OPERATING COSTS

21.1 PROJECT REQUIREMENTS

21.1.1 Introduction

The purpose of this capital cost (CAPEX) and operating cost (OPEX) estimate is to provide costs to an accuracy of +15 % −10 % that can be utilised for the economic analysis of the Kobada 3 Mt/a gold project capable of producing 100,000 oz/a of gold.

21.1.2 Responsibilities

The project's capital and operating cost estimate breakdown with associated responsibilities consists of the following:

- Mining costs estimated by DRA Americas
- Process plant and on-site infrastructure costs estimated by SENET
- TSF development and closure costs estimated by Epoch
- Other supporting infrastructure costs estimated by SENET
- Owner's pre-production costs estimated by SENET
- Environmental management: rehabilitation and closure costs estimated by ABS Africa

21.1.3 Escalation

The 2020 DFS CAPEX has been escalated where relevant for this 2021 DFS update. There has been no further escalation allowed for in this 2021 DFS CAPEX update. The EPCM Contractor's rates reflect the rates expected in the third quarter of 2021.

21.1.4 Exclusions

The following were not included in this CAPEX estimate:

- Financing costs
- Taxes and duties
- Permits
- Sunk costs
- Currency fluctuations
- Ongoing Exploration Costs
- AGG Mali Corporate Costs

21.2 CAPITAL COSTS

21.2.1 Basis of Estimate

SENET was approached in 2020 to conduct a DFS for the Kobada gold project which was concluded in June of 2020. In 2021, AGG contracted SENET to perform an update of the CAPEX estimate, based on the DFS completed in July 2020. The approach for the CAPEX update, was to use the 2020 design and equipment as a basis, updating project rate of exchange, get formal enquiries from market and to escalate the 2020 adjudicated prices according to STATSA indices, see Figure 21.1.

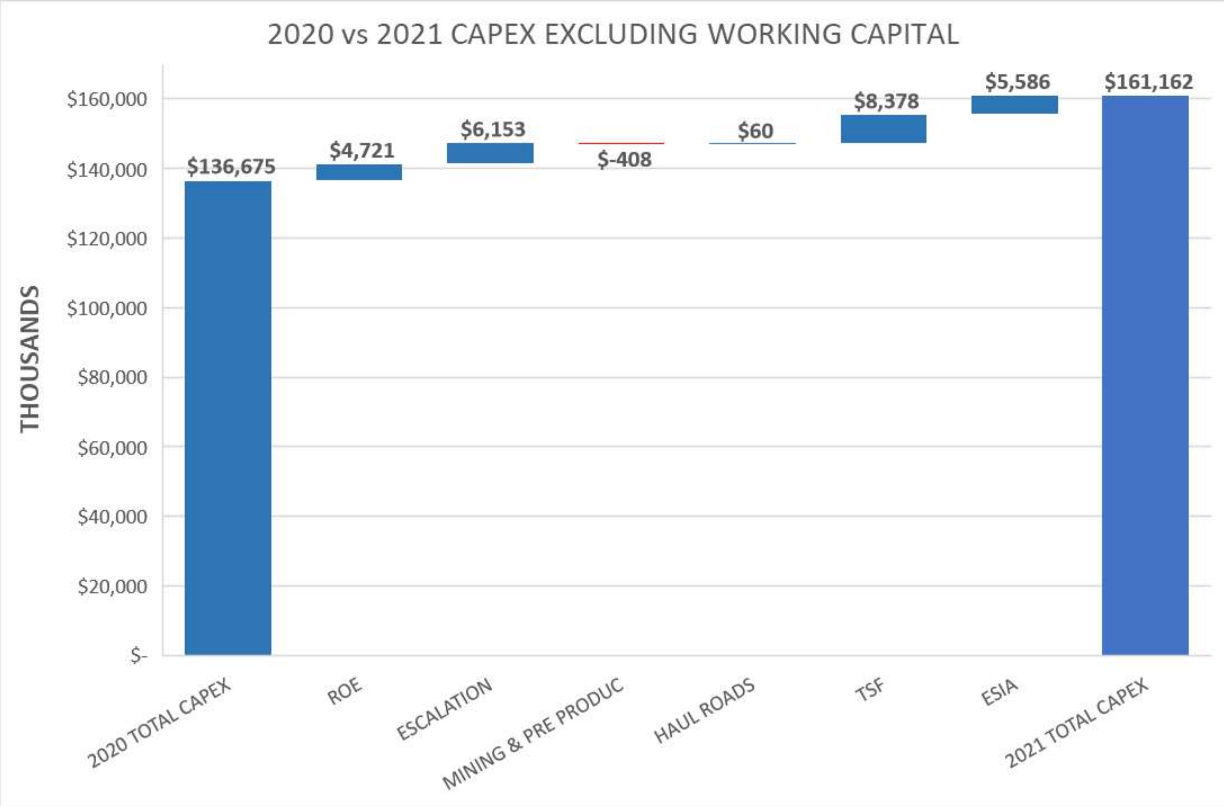


Figure 21.1: 2020 vs 2021 CAPEX Excluding Working Capital

Table 21.1 indicates the cost difference per discipline in more detail.

Table 21.1: Cost Difference per Discipline

Description	2020 Total CAPEX Incl. Contingency (US$)	2021 Total CAPEX Incl. Contingency (US$)	Total Difference (US$)
Earthworks	7,291,202	8,293,405	1,002,203
Civil Works	7,828,790	9,317,916	1,489,126
Infrastructure	1,706,077	1,987,890	281,812
Structural Steel	2,516,222	3,672,598	1,156,377
Plate Work	1,060,929	1,681,559	620,630
Tankage	1,882,023	2,455,013	572,990
Machinery and Equipment	11,594,331	13,002,591	1,408,260
Piping	3,505,116	4,671,617	1,166,502
Valves	286,090	345,478	59,388
Electricals	3,465,636	4,415,411	949,774
Instrumentation	1,218,609	1,529,953	311,344
Transport	7,360,293	7,947,391	587,098
E&I Installation	3,249,864	3,242,535	-7,329
SMPP Installation	8,319,341	8,349,537	30,196
Tools and Mobile Equipment	1,235,126	1,402,909	167,784

Description	2020 Total CAPEX Incl. Contingency (US$)	2021 Total CAPEX Incl. Contingency (US$)	Total Difference (US$)
TOTAL DIRECT FIELD COSTS	**62,519,648**	**72,315,803**	**9,796,155**
Commissioning Spares	174,799	221,119	46,320
2-Year Operational Spares	1,016,002	892,563	-123,439
Insurance and Critical Spares	975,532	1,579,459	603,927
Vendor Services	793,555	857,311	63,756
First Fills	702,427	724,138	21,711
TOTAL INDIRECT FIELD COSTS	**3,662,315**	**4,274,591**	**612,276**
TOTAL FIELD COST	**66,181,963**	**76,590,393**	**10,408,431**
Project Management (EPCM)	9,410,700	9,496,671	85,971
Insurances and Guarantees	435,600	427,680	-7,920
Project Taxes	1,769,420	2,156,689	387,269
TOTAL HOME OFFICE COSTS	**11,615,720**	**12,081,040**	**465,320**
TOTAL PROJECT COST	**77,797,682**	**88,671,434**	**10,873,751**
TSF	21,047,828	29,425,331	8,377,503
Mining Pre-Production	23,983,233	23,574,882	-408,351
Mining Establishment	3,872,000	3,872,000	-
Haul Roads	744,594	804,162	59,568
IFC PS Readiness Assessment		384,366	384,366
Owners Pre-Production Cost	8,263,660	8,263,660	-
Resettlement Action Plan	965,503	6,166,640	5,201,137
TOTAL OTHER COST	**58,876,818**	**72,491,040**	**13,614,223**
TOTAL COST	**136,674,500**	**161,162,474**	**24,487,974**
WORKING CAPITAL	**10,883,747**	**4,782,847**	**-6,100,899**

From Table 21.1, major changes in capital expenditure are increases in steel prices, rate of exchange and a redesign of the tailings storage facility and associated resettlement plan as a result of a significantly increased TSF footprint.

21.2.2 Factors that Increased CAPEX

21.2.2.1 Rate of Exchange

In the 2020 DFS the USD vs ZAR exchange rate applied was R17,00 to $1 based on July 2020 according to xe.com and now in the 2021 DFS update the exchange rate applied was R14,50 to $1, the Rand has strengthened significantly against other currencies since the

CAPEX was originally done in the 2020 study. Rate of exchange impact alone results in an increase of US$4,720,573.

The 2020 vs 2021 Rate of Exchange is given in Table 21.2.

Table 21.2: 2020 vs 2021 Rate of Exchange

	2020	2021
USD:ZAR	17.00	14.50
USD:AUD	1.57	1.57
USD:EUR	0.93	0.93
USD:GBP	0.81	0.81

21.2.2.2 Escalation/Updated Rates

The approach applied, was to escalate the adjudicated prices from the 2020 DFS, however on certain items for example the installation contracts and a few other mechanical equipment packages, SENET obtained updated costs from the respective vendors and contractors, which results in an increase of US$6,153,178.

21.2.2.3 Haul Roads

The increase of US$59,568 is related to new rates received back from the Earthworks contractor that will be responsible for the haul road construction.

21.2.2.4 TSF

As a result of the increased mining production, the TSF size increased to 45 Mt and as a result an overall increase in CAPEX of US$8,377,503. For more details, refer to Section 21.2.6.1.

21.2.2.5 ESIA

An increase is seen in the ESIA capital cost allowances, mainly due to an increase in associated resettlement costs because of the increased mining and TSF footprint. The associated capital cost has increased by US$5,585,503. Refer to Table 21.3 with cost estimated by the support environmental consultant ABS.

Table 21.3: Resettlement and Livelihood Restoration Costs

KOBADA GOLD PROJECT - 2021 SULPHIDES DFS		
Resettlement and Livelihood Restoration - 500m Buffer Zone		
Description		**Provision**
Structures		
Structures and Shelters		$ 2,799,809
Technical Studies and Land Acquisition	15%	$ 419,971
RAP Development and Implementation		$ 1,200,000
Administration Costs	2.00%	$ 55,996
Vulnerable Group Assistance	2.00%	$ 55,996
Monotoring - External	1.00%	$ 27,998
Transfer of assets to new site	1.50%	$ 41,997
Agricultural Assets		
Cultivated crops, trees and other agricultural assets		$ 974,268
Protected Trees		
Protected Trees - DNACPN Charge		$ 30,000
Sub-Total		**$ 5,606,037**
Contingency - Housing and Agriculture	10%	$ 560,604
Total - Structures & Agriculture		**$ 6,166,640**

There are opportunities for optimising the above cost once the project moves into the construction phase.

An additional allowance of $384,366 is included in the 2021 DFS CAPEX update for IFC PS Readiness Assessment.

21.2.3 Direct Field Cost

Figure 21.2 represents the increase in direct field cost from the 2020 DFS to 2021 DFS which is mostly affected by rate of exchange and escalation.

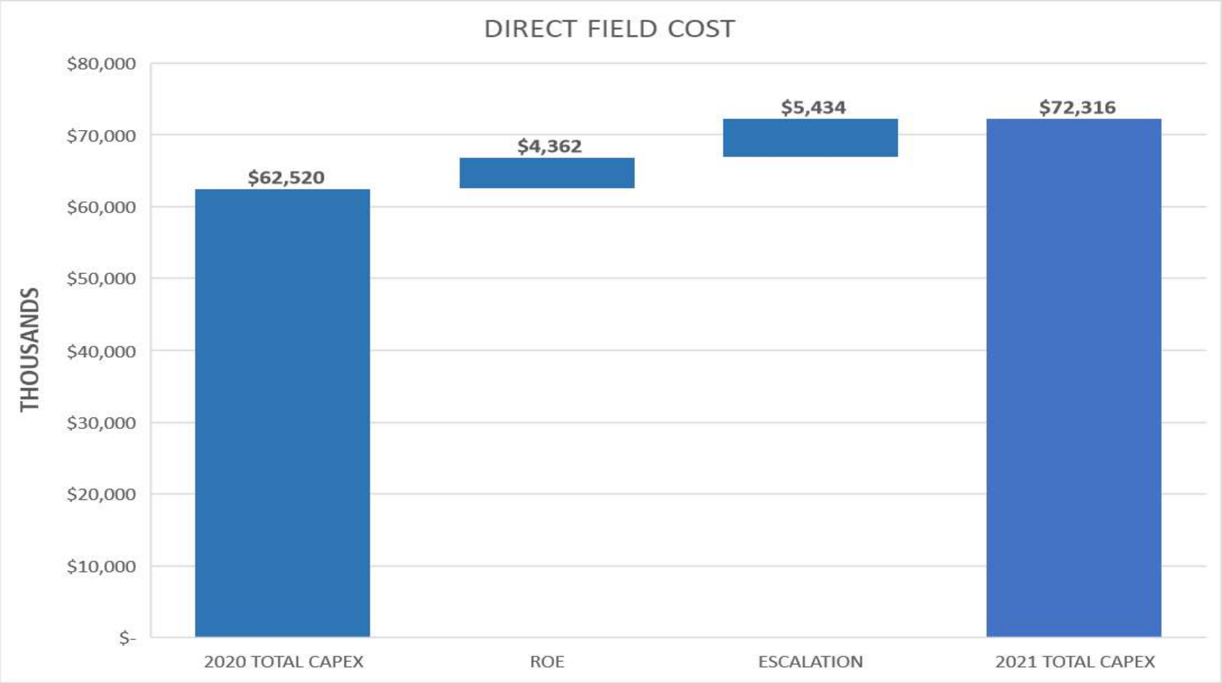


Figure 21.2: Direst Field Costs

21.2.3.1 Earthworks

For the updated cost, SENET issued the previous 2020 Bill of Quantities (BOQ) back to the market for repricing, no changes were made to the quantities except for diversion channels that was not part of the 2020 study, the new rates were adjudicated and applied to the BOQ, rates received back from the market have increased since 2020 so the total earthwork price increased by 17 % with the inclusion of the diversion channel cost. Refer to Table 21.4 for a comparison of the 2020 vs 2021 Earthworks DFS CAPEX.

Table 21.4: 2020 vs 2021 DFS Earthworks CAPEX Comparison

Description	CAPEX 2020 (US$)	CAPEX 2021 (US$)	Difference (US$)
Earthworks	7,291,202	8,293,405	1,002,203

All Bulk Earthworks engineering design in accordance with applicable codes and standards, including calculation, design, and construction of all foundations for the Works. Any re-shaping or imported fill, compacted to the engineer's specification to form suitable structural engineered materials or any other necessary earthworks or civil works to achieve the acceptable conditions for the plant foundations.

The process plant and peripheral infrastructure earthworks main contract scope was based on a detailed BOQ derived from earthwork terrace drawings and associated cut and fill long sections and quantities. The BOQ for the process plant earthworks was prepared in accordance with SANS 1200 A and SANS 1200 D.

The earthworks enquiry (FIDIC based form of contract) was issued to suitable Contractors currently based in Mali and surrounding countries and included preliminary and general (P&G) cost lists and contractual conditions against which Tenderers quoted. Fully inclusive "wet rates" and plant/labour histograms were received from the respective tenderers, and these were compared and adjudicated accordingly. The contractor will provide accommodation for their own workforce and will be completely self-sufficient. Included in the earthworks capital cost is the following:

- Earthworks- Off Site Infrastructure.
- Earthworks- Process Plant.
- Earthworks- On Site Infrastructure.
- Earthworks- Fie River Bridge.
- A geotechnical investigation was carried out by Inroads Consulting for the oxides phase and was used as the basis for the engineered fill requirements for the plant areas. Due to the relocation of the process plant position, the investigation may have to be expanded to include the final plant position.
- A surfmate model was done based on the required terrace levels and the quantities of cut and fill were derived. Due to the nature of the in-situ ground conditions, the philosophy was to keep all foundations above the laterite cap, and hence the platforms were deemed to be predominantly made up of imported filled terraced.
- Total earthworks quantities (oxides scope only) comprise of approximately 330,000 m³ of excavation and compacted fill.
- A detailed BOQ incorporating the construction of plant terraces, HDPE lined raw water, process water and event ponds, the ROM wall construction, inclusive of the engineered fill behind the wall (Maccaferri gabion baskets with tie backs), drainage and in-plant gravel roads.
- Provision has been made for a free-haul distance of 2-kilometers for all earthworks.
- The Earthworks enquiry was issued to market and a comprehensive adjudication was subsequently done in favour of De Simone, a competent and experienced earthworks contractor.
- The time for completion of the Sulphides earthworks has been scheduled for two months.
- The HDPE lining will be supplied by ROWAD and installed by Engineered Linings.

21.2.3.2 Civil Works

SENET issued the original BOQ of the 2020 study to the market for repricing, no changes were made to the quantities. The new rates were adjudicated and applied to the BOQ; rates have increased since 2020 so the total civil work price increased by 19 %.

All Civil Works engineering design in accordance with applicable codes and standards, including calculation, design and construction of all foundations for the Works.

Based on engineering concept design, construction quantities were derived from layouts and modelling to form the basis of tender bills of quantities. Same where issued to market as procurement packages for pricing by vendors. The tendered scope of work for the civil works discipline includes amongst others, the construction to final line and level for the following project areas:

The process plant and peripheral infrastructure civil works main contract scope was based on a detailed BOQ derived from civil outline drawings, mechanical general arrangement drawings and the site block plan. The BOQ for the process plant civil works was prepared in accordance with SANS 1200 A and SANS 1200 G.

The scope of work for the process plant civil works includes the following:

- Reinforced concrete foundations for the support of mechanical equipment, structural steelwork, and plate work.
- Reinforced concrete surface beds and bund walls with trenches and sumps to contain spillages within the process plant.
- Site storm water drainage including a network of v-drains.
- Reinforced concrete foundations and surface beds for:
    - Process plant prefabricated buildings.
    - Process plant pre-engineered steel buildings (reagents stores, plant workshop and warehouse).
- Building works and architectural finishes for the brickwork offices and stores within the plant workshop and warehouse.
- Reinforced concrete foundations for the plant containerised MCCs, control room and server room.
- Brickwork transformer bay buildings.
- Sewerage reticulation for the process plant, and trenching for electrical, potable water and fire water reticulation.
- Security fencing for the process plant.

All civil materials of construction are included in the Civil Contractor's scope of supply (cement, reinforcement, mesh, HD bolts, bricks, etc.).

The civil works enquiry (FIDIC based form of contract) was issued to suitable contractors currently based in Mali and surrounding countries and included P&G cost lists and contractual conditions against which tenderers quoted. Fully inclusive "wet rates" and plant/labour histograms were received from the respective tenderers, and these were compared and adjudicated accordingly. The contractor will provide accommodation for their own workforce and will be completely self-sufficient.

For the purposes of the Sulphides update to the DFS, it is advisable to include the raft foundation civil works associated with the proposed secondary ball mill with the Oxides scope of work for practical reasons such as ease of construction. A single combined raft for both mills will then be designed and constructed as a unit as opposed to two separate structures.

The tendered scope of work for the civil works discipline includes amongst others, the construction to final line and level for the following project components:

- Plant Front-End and Comminution circuit:
    - Secondary crushing, screening and transfer station structures.
    - Tertiary crushing, screening and transfer station structures.

- Secondary and Tertiary crushing conveyors.
- Secondary ball mill.

- For the Oxides study phase, the total structural concrete quantity is approximately 7600m3 with and associated blinding quantity of 310m³ based on estimated concrete quantities from previous similar projects.
- The Sulphides study phase comprise of approximately 1540m3 of structural concrete and 310m³ of blinding, based on estimated concrete quantities from previous similar projects
- Construction of reinforced concrete foundations for the support of mechanical equipment, structural steelwork and plate work.
- Construction of reinforced concrete surface beds and bund walls with trenches and sumps to contain spillages within the processing plant.
- Installation of acid-proofing membranes to the acid wash, elution and reagents make-up areas.
- Construction of site storm water drainage including a network of v-drains
- Construction of reinforced concrete foundations and surface beds for:
  - The process plant prefabricated buildings.
  - The process plant pre-engineered steel buildings (reagents stores, plant workshop and warehouse).
- Building works and architectural finishes for the brickwork offices and stores within the plant workshop and warehouse.
- Construction of reinforced concrete foundations for the plant containerised motor control centres (MCCs), control room and server room.
- Construction of brickwork transformer bay buildings.
- Installation of sewerage reticulation for the processing plant, and trenching for electrical, potable water and fire water reticulation.
- Installation of security fencing for the processing plant (free issued to contractor).
- All reinforcing steel and other construction materials will be supplied by the contractor.
- The time for completion of the Sulphides civil works has been scheduled for four (4) months.
- The following contractors will execute the various scopes:
  - De Simone has been selected as the preferred contractor for the Civil works scope for both the Oxides and Sulphides phases of the project.
  - Acid proofing will be performed by Afrores and Trinidad (A&T), and the contract will be based on a BOQ, against which quoted rates will be applied. Refer to Table 21.5 for a comparison of the 2020 vs 2021 Civil DFS CAPEX.

Table 21.5: 2020 vs 2021 DFS Civil CAPEX Comparison

Description	CAPEX 2020 (US$)	CAPEX 2021 (US$)	Difference (US$)
Civil Works	7,828,790	9,317,916	1,489,126

21.2.3.3 Infrastructure

Escalation was applied to the infrastructure quotes received for the 2020 DFS study this, together with the strengthening of the ZAR from 2020 to the 2021 DFS update, results in an increase of US$281,812. Refer Table 21.6 for a comparison of the 2020 vs 2021 Infrastructure DFS CAPEX.

Table 21.6: 2020 vs 2021 DFS Infrastructure CAPEX Comparison

Description	CAPEX 2020 (US$)	CAPEX 2021 (US$)	Difference (US$)
Infrastructure	1,706,077	1,987,890	281,812

21.2.3.3.1 Prefabricated Buildings

The CAPEX of the infrastructure and process plant buildings includes the following:

- Prefabricated plant and infrastructure buildings.
- Pre-engineered steel buildings.
- Process plant furniture.
- Change house building.
- Plant office building.
- Gatehouse building.
- Weighbridge.
- Control room.

The prefabricated building package includes the supply and site installation of the building, including all internal electrical reticulation and fittings, all internal water reticulation, plumbing, sanitary fittings, and air conditioning. The concrete foundations and floor slab will be provided by the Civil Contractor.

The Prefabricated Building Contractor has allowed for the relevant P&G costs for the installation of the buildings, including the supply of their own tented construction camp.

A supply and installation enquiry package were issued to the market for prefabricated buildings. The vendor was selected based on both technical and commercial adjudications.

21.2.3.3.2 Pre-Engineered Steel Buildings

The following pre-engineered steel buildings will be supplied:

- Plant Workshop.
- Plant Warehouse.
- Reagent Stores.

The pre-engineered steel building package includes the design, fabrication, and supply of the steel buildings. Site installation of the buildings and all internal electrical reticulation and fittings are included in this package. The concrete foundations and floor slab will be provided by the Civil Contractor.

A design, fabrication and supply enquiry package were issued to the market for pre-engineered steel buildings. The vendor was selected based on both technical and commercial adjudications.

21.2.3.3.3 Mine/Plant Assay Laboratory

The assay laboratory package includes the supply, installation and ultimately the operation of the assay laboratory. The supply includes all the laboratory equipment, furniture, and fixtures, including the supply of the pre-populated modularised/containerised building arrangements. These buildings include all internal electrical reticulation and fittings, all internal water reticulation, plumbing, sanitary fittings, and air conditioning. The concrete foundations and floor slab will be provided by the Civil Contractor.

The Assay Laboratory Contractor has allowed for the relevant preliminary and general costs for supervision during erection of the contractor-supplied pre-populated containerised buildings by the SMPP Contractor.

A supply, install and operate enquiry package was issued to the market for the assay laboratory. The vendor was selected based on both technical and commercial adjudications.

An interconnecting steel roof structure between the containerised assay laboratory buildings has been included in the steel supply contract and shall be erected by the SMPP Contractor, who shall also install the raw and potable water connections to the facility.

Electrical supply to the laboratory is included in the electrical, control and instrumentation (E, C&I) contract.

21.2.3.3.4 Prefabricated Off-Site Infrastructure and Camp Buildings

The following off-site buildings will be supplied as prefabricated buildings:

- Main administration building adjacent to process plant.
- Single and double staff quarters located at existing senior management new camp.
- Laundry building located at existing senior management new camp.
- Recreation hall located at existing senior management new camp.

The prefabricated building package includes the supply and site installation of the buildings, including all internal electrical reticulation and fittings, all internal water reticulation, plumbing, sanitary fittings, and air conditioning.

The concrete foundations, floor slab and connection to external sewerage reticulation and potable water will be provided by the Civil Contractor.

The Prefabricated Building Contractor has allowed for the relevant preliminary and general costs for the installation of the buildings, including the supply of his own tented construction camp.

A supply and installation enquiry package were issued to the market for prefabricated buildings. The vendor was selected based on both technical and commercial adjudications.

Electrical supply to the respective prefabricated buildings is included in the E, C&I contract. Furniture supply for the respective buildings is included in the furniture contract.

21.2.3.3.5 Senior Management New Camp Infrastructure and Upgrade

The senior management New Camp is an existing camp built in 2019 that comprises the following:

- Single and double quarters (sized for 40 people – sharing).
- Dining hall and kitchen (sized for 100 people).
- Borehole, raw water storage and septic tank arrangements (sized for 40 people).
- Services reticulation (sized for 40 people).
- Fencing, gate, and gatehouse for security control.
- 0 kVA and 30 kVA generators, complete with reticulation to existing building units.

The senior management new camp shall be upgraded as follows:

- Additional prefabricated single and double quarters (sized for 60 people; this will result in a total capacity of 100 people).
- Prefabricated laundry (sized for 100 people).
- Raw water take-off from the process plant raw water supply from the Niger River to the camp, complete with localised potable water treatment plant and additional potable water storage tank.
- Containerised sewage treatment plant (sized for 100 people).
- Additional services reticulation (sized for 60 people; this will result in a total capacity of 100 people).
- Earthworks for terraces and civil works for new building slabs.
- Additional fencing.
- Additional equipment and furniture, i.e., catering, laundry and accommodation.
- Electrical supply from the new HFO power plant, complete with reticulation to the new prefabricated buildings (The 10 kVA and 30 kVA generators shall serve as backup.).

The upgrades shall enable operational senior management and technical staff to be accommodated at the new camp, with catering and housekeeping being provided by an outsourced catering contractor.

The CAPEX for the earthworks and civil works for the upgrades to the new camp was compiled based on BOQs that accompanied the main process plant earthworks and civil works enquiries, respectively.

The CAPEX for the fencing of the new camp was based on the Fencing Supply Contractor's unit rates with the Civil Contractor's rates used for erection.

The potable water treatment plant, sewage treatment plant, kitchen and laundry equipment, furniture and bedding as required for the New Camp were provided, taking cognisance of the existing inventory.

A small fire utility truck will be acquired to extinguish fires at the new camp, and all the new camp buildings will be equipped with 6 kg dry powder fire extinguishers.

21.2.3.4 Structural Steel and Plate Work

The structural steel Bill of Quantities was issued to the market for revalidation of the 2021 DFS costs. The results in a CAPEX increase of 52% associated with structural steel and plate work supply and fabrication. See Table 21.7 for a comparison of the 2020 vs 2021 Structural Steel and Plate Work DFS CAPEX.

Table 21.7: 2020 vs 2021 DFS Structural Steel and Plate Work CAPEX Comparison

Description	CAPEX 2020 (US$)	CAPEX 2021 (US$)	Difference (US$)
Structural Steel	2,516,222	3,672,598	1,156,377
Plate Work	1,060,929	1,681,559	620,630

Quantities were established based on material take-offs (MTOs) derived from plant general arrangement drawings produced during the 2020 DFS study. Unit rates for supply and fabrication were obtained from fabricators and were applied to the MTOs.

The respective rates were applied to the bill of materials for the following equipment:

- Structural steelwork.
- Plate work.
- Liners.
- Grating and flooring.
- Handrailing.
- Sheeting.

There is approximately 940t of structural steelwork and 261t of plate work that has been estimated for the plant.

The BOQ, together with the specifications formed the basis for procurement enquiries.

The procurement enquiries were issued to the respective fabricators, companies that are capable of detailing, supplying, fabricating, and painting such scope of works.

Their tender proposals were reviewed and adjudicated against commercial, technical and schedule aspects.

21.2.3.5 Tankage

The finally adjudicated tank prices of the 2020 DFS were escalated utilising STATSA as a reference; this resulted in a 30 % increase from the 2020 DFS study costs. See Table 21.8 for a comparison of the 2020 vs 2021 Tankage DFS CAPEX.

Table 21.8: 2020 vs 2021 DFS Tankage CAPEX Comparison

Description	CAPEX 2020 (US$)	CAPEX 2021 (US$)	Difference (US$)
Tankage	1,882,023	2,455,013	572,990

The unit rates for supply and fabrication were obtained from fabricators and were applied to the tank schedule to estimate the plate work cost.

- Based on the process design requirements for the SP0800 Oxides circuit, a tank schedule was developed.
- The smaller tanks, 2 m diameter and less will be shop fabricated where the tanks larger than 2 m in diameter will be supplied in kit form and erected on site. The consumables for completing these tanks would need to be supplied by the selected SMPP contractor.
- The tank schedule with corresponding information was issued to selected fabricators to quote.

21.2.3.6 Mechanical Equipment

The mechanical cost was updated using various methods, including costs escalation applied to certain equipment, that results in an increase to costs of 3 %. New prices were obtained from vendors through formal enquiries such as for the Mills, crushers, pumps, and all materials handling equipment for the front-end section of the Sulphides plant circuit. All of this resulted in an increase of 12 %. See Table 21.9 for a comparison of the 2020 vs 2021 Mechanical Equipment DFS CAPEX.

Table 21.9: 2020 vs 2021 DFS Mechanical Equipment CAPEX Comparison

Description	CAPEX 2020 (US$)	CAPEX 2021 (US$)	Difference (US$)
Machinery and Equipment	11,594,331	13,002,591	1,408,260

The mechanical equipment quantities were derived from the equipment lists and process flowsheets. The mechanical scope of work for the project is to supply the equipment as detailed in the mechanical equipment list, mechanical data sheets and mechanical drawings.

Enquiries were prepared, inclusive of equipment data sheets, and were then sent to equipment vendors/suppliers. Quotations were received and were commercially and technically adjudicated.

The Mechanical Equipment List (MEL), was developed from the process design requirements and P&IDs.

The MEL generated from the original SP0800 Kobada Gold Project, was used as the basis and expanded to include the additional equipment needed for the Sulphides scope of works.

21.2.3.7 Piping and Valves

For the updated CAPEX the same bill of quantities was used as for the 2020 DFS study, an average escalation of 8% was applied, because of the various piping material specifications. Majority of the equipment was priced in ZAR, so the stronger rand also increased the cost. See

Table 21.10 for a comparison of the 2020 vs 2021 Pipes and Valves DFS CAPEX.

Table 21.10: 2020 vs 2021 DFS Pipes and Valves CAPEX Comparison

Description	CAPEX 2020 (US$)	CAPEX 2021 (US$)	Difference (US$)
Piping	3,505,116	4,671,617	1,166,502
Valves	286,090	345,478	59,388

Quantities were based on MTOs derived from piping and instrumentation diagrams (P&IDs) and general arrangement drawings and were benchmarked against similar projects recently executed by SENET. MTOs for all carbon steel, stainless steel and HDPE piping were forwarded to vendors for pricing.

A valve list compiled from the P&IDs, detailing each manual valve type, was submitted to vendors for pricing. The actuated valve costs were included in the instrumentation cost estimate. The quotations received were technically and commercially adjudicated, and preferred vendors were selected. The scope of work relating to piping and valves is indicated in the P&IDs.

21.2.3.7.1 Piping and Fittings

The piping and fittings were derived from the relevant set of P&IDs and can be categorised under the following headings:

- Carbon Steel Piping and Fittings.
- HDPE or Non-Metallic Piping and Fittings.
- Stainless Steel Piping and Fittings

These bills were issued to the market to obtain unit rates for the respective items. Following receipt of the respective quotes, a technical and commercial review of the offer was completed to ensure that the offer aligns with the specifications and requirements stipulated in the official enquiry.

21.2.3.7.2 Manual Valves

The type, size and quantity of manual valves was calculated from a valve list that was extracted from the P&IDs.

The following valve types are included:

- Moisture drains.
- Air release valves.
- Ball valves.
- Diaphragm valves.
- Butterfly valves.
- Non-return valves.
- Knife gate valves.

- Pinch valves.
- Pressure relief valves.
- Pressure reducing valves.
- Strainers.

The valve bills were issued to the market to obtain unit rates of each type and size. Following receipt of the respective quotes, a technical and commercial review of the offer was completed to ensure that the offer aligns with the specifications and requirements stipulated in the official enquiry.

21.2.3.8 Electricals

The electrical cost was revalidated with updated market rates received through formal enquiries. This resulted in an increase of 27 %. See Table 21.11 for a comparison of the 2020 vs 2021 Electricals DFS CAPEX.

Table 21.11: 2020 vs 2021 DFS Electrical CAPEX Comparison

Description	CAPEX 2020 (US$)	CAPEX 2021 (US$)	Difference (US$)
Electricals	3,465,636	4,415,411	949,774

The process plant and peripheral infrastructure electrical equipment detailed BOQ was derived from layout drawings, process plant general arrangement drawings, electrical single line diagrams, mechanical equipment lists and motor lists.

The electrical equipment includes the following:

- Medium-voltage switchgear.
- Step-down transformers.
- MCCs.
- Low-voltage and medium-voltage cables.
- Cable racking, luminaires, and earthing.
- Power factor correction units.
- Backup power system.

Electrical equipment supply enquiries were prepared, inclusive of equipment data sheets, and sent to approved equipment vendors/suppliers. Quotations were received and were commercially and technically adjudicated.

- A load list was created based on the mechanical equipment list with the inclusion of the sulphide's equipment.
- Based on the load list and plant layout, SLDs were drawn up and subsequent MTOs, upon which vendors were engaged for equipment supply, i.e. MCCs, transformers, cables etc.
- Only the electrical equipment that have been directly impacted by the sulphide's inclusions costs were derived from a tender process, whereby an enquiry was sent to several suppliers.

- The original adjudicated equipment pricing was adjusted according to the latest relative market information's and recently complete projects.

21.2.3.9 Instrumentation

The instrumentation cost was revised and updated with updated market rates received, all of this resulted in an increase of 26 %. See Table 21.12 for a comparison of the 2020 vs 2021 Instrumentation DFS CAPEX.

Table 21.12: 2020 vs 2021 DFS Instrumentation CAPEX Comparison

Description	CAPEX 2020 (US$)	CAPEX 2021 (US$)	Difference (US$)
Instrumentation	1,218,609	1,529,953	311,344

The process plant requires the implementation of a process automation system (PAS). The PAS comprises a SCADA, PLC, and instrumentation. SCADA and PLC equipment will be in the plant control room and the equipment room, located adjacent to the plant control room. A security system, including CCTV cameras and access control to site and the gold room, is also required.

The communications system (consisting of an office LAN, satellite link for Internet and email, telephone, and radio systems) will be provided by the Client. The PLC and SCADA costs were based on a typical plant configuration with full plant control from a central control room. Provision was also made for a sequel server for constant data logging and trending.

Instrumentation costs were based on instrument and valve lists. The instrumentation BOQ was developed from data derived from the P&IDs, as well as the instrument list and the instrumentation drawings.

Dedicated remote I/O panels, located in the specific plant areas, are utilised to connect the field instruments to the PLC. Digital instruments are wired to the remote I/O panel via multipair cables. Analogue instruments are connected to the remote I/O panels via a Profibus PA network.

21.2.3.9.1 General

The Kobada Sulphides project requires the implementation of a PAS. This includes instrumentation, cabling, distributed control system (DCS) and an automation local area network (LAN).

21.2.3.9.2 C&I Equipment Quantities

As part of the PAS, instrumentation has been allowed for to monitor and control the various functions of the process plant, as described in the P&IDs. The C&I capital cost estimate was based on quantities as per the P&IDs and control valve list:

21.2.3.9.3 Control Valves

Control Valves include the following types.

- In-line Solenoid valves.
- Actuated valves (on/off).
- Monitored hand valves.
- Control valves (Modulating).

21.2.3.9.4 Instruments

The Following types of instruments were identified on the P&IDs:

- Conveyor instrumentation, including the following:
  - Belt pull-keys.
  - Belt rip/tear detectors.
  - Belt alignment switches.
  - Belt under-speed switches.
  - Blocked chute detectors.
  - Limit switches on take-ups.
- Densitometers.
- Flowmeters and transmitters.
- Flow switches.
- Flow indicators.
- Level transmitters.
- Level switches.
- • Pressure transmitters.
- Pressure indicators.
- Flame detectors.
- pH transmitters.
- Conductivity transmitters.

21.2.3.9.5 Control System

The number of instruments, control valves and mechanical packages to be interfaced to the control system were used to plan the size and architecture of the control system.

The planned control system consists of a DCS. The automation backbone consists of a LAN, utilising Ethernet and other proprietary control protocols.

The estimate is based on a Siemens PCS7 typical architecture. The operator interface consists of two dual screen operator stations located in the Central Control Room. The system also includes a dual redundant set of Operator Station Servers, one dual screen Engineering Station, a Webserver, and a Historian.

The control system will back-up all the data for a pre-defined period and store it in a SQL data format for extraction / interface to other systems (MIS, MES or ERP, outside of SENET scope).

The purpose of the Webserver is to give controlled access to the real time screens and information of the control system, in a secure manner. This means that privileged information can be seen / accessed on the internet, in a secure and controlled manner, only by the persons with the correct viewing rights.

The system architecture also includes remote I/O stations (9 × RIO panels) situated in the plant and connected to the controllers via a Profibus DP network running on multimode fibre. The 2 × DCS controllers are utilised for controlling the plant.

The DCS controller hardware will be housed in the equipment room, which will normally be locked, or have limited access, and will also be climate controlled. Extension cables for the operator stations (dual screens, mouse, keyboard) will be run from the equipment room to the control room, as the computer CPUs will be kept in an environment that is secure and tamper free as well as climate controlled and dust free.

The DCS software will be based on the P&IDs and the functional specifications provided by the process engineering department. Standards will be used as much as possible to make factory acceptance testing (FAT) and site acceptance testing (SAT) as seamless as possible, and to ensure that future additions and deletions to software is done in a well-orchestrated manner.

Software licences for the DCS were sized adequately to enable future expansions if needed.

The hardware was designed to allow for 20 % populated spare capacity, with additional 20 % unequipped spare capacity.

The automation network is based on an Ethernet topology, where the main backbone is a fibre-optic network.

Copper / CAT6 UTP cable is utilised to connect the control equipment located in the control room and equipment room (operator stations, servers, controllers).

21.2.3.9.6 Pneumatic Equipment

The quantities of pneumatic equipment were estimated based on the number of valves as per P&ID set for the project. Pneumatic equipment includes the following items:

- Stainless Steel Instrument Air Manifolds. The quantity is based on the number of valves estimated for the project. The manifolds are indicated on the P&IDs.
- 12 mm Decabon tubing. Quantity based on an average length of 10m tubing per valve.
- Quick connect fittings as required.
- 20 % spare capacity included.

21.2.3.9.7 Instrument Cable

Quantities are based on instrument counts and standard lengths (30 m per instrument). Provisional amounts are included to connect to MCC’s and vendor packages. The following Cable types are included:

- Cable – 1.5mm² 1Pr Xlpe/Oam/Pvc/Apl/Pe Fr Uv.
- Cable – 1.5mm² 2Pr Xlpe/Ioam/Pvc/Apl/Pe Fr Uv.
- Cable – 1.5mm² 4Pr Xlpe/Ioam/Pvc/Apl/Pe Fr Uv.
- Cable – 1.5mm² 8Pr Xlpe/Ioam/Pvc/Apl/Pe Fr Uv.
- Cable – 1.5mm² 12Pr Xlpe/Ioam/Pvc/Apl/Pe Fr Uv.
- Cable – 1.5mm² 24Pr Xlpe/Ioam/Pvc/Apl/Pe Fr Uv.
- Cable – 1.5mm² 3c Pvc/Pvc/Swa/Pvc.
- Cable – Fibre Optic - 24 Fibre - CST - Multimode cable.
- Profibus DP Cable.
- Cable – Ethernet/Profinet, Cu-cable.

21.2.3.9.8 Cable Racking

Quantities of the HDG cable racking are based on:

- Estimated number of RIO panels.
- Instrument count from P&IDs.
- MCC count.
- Estimated requirements for mechanical vendors.
- Conveyor requirements.

20 % spare capacity is included.

21.2.3.10 Transport

For revalidated transport cost, SENET obtained new rates for transporting goods to site, these rates were applied to the transport list and resulted in an increase of 8%. See Table 21.13 for a comparison of the 2020 vs 2021 Transport DFS CAPEX.

Table 21.13: 2020 vs 2021 DFS Transport CAPEX Comparison

Description	CAPEX 2020 (US$)	CAPEX 2021 (US$)	Difference (US$)
Transport	7,360,293	7,947,391	587,098

The freight costs for the project are based on the actual mass of the mechanical equipment, the mass of the structural steelwork as generated in the respective MTOs, and the calculated mass of piping and valves as contained in the respective BOQs. Actual quotes were obtained from local freight forwarders.

Most of the rates utilised in the freighting cost have been obtained from third-party service providers and statutory agencies involved in the management of shipments and documentation. Rates offered by these bodies are subject to change without notice and, therefore, cannot be held as fixed and firm. The contractor will have to negotiate these increases on an ad-hoc basis with the Client as and when they arise, fully supporting the application with documentary evidence of such increases.

21.2.3.11 Installation Contracts

Both the E&I and SMPP packages was revalidated, and CAPEX was updated accordingly with minor increase in SMPP. See Table 21.14 for a comparison of the 2020 vs 2021 Installation Contracts DFS CAPEX.

Table 21.14: 2020 vs 2021 DFS Installation Contracts CAPEX Comparison

Description	CAPEX 2020 (US$)	CAPEX 2021 (US$)	Difference (US$)
E&I Installation	3,249,864	3,242,535	-7,329
SMPP Installation	8,319,341	8,349,537	30,196

21.2.3.11.1 E&I Works Contract

The process plant and peripheral infrastructure E&I works main contract scope was based on a detailed BOQ derived from process plant and infrastructure layout drawings, process plant general arrangement drawings, electrical single-line diagrams, mechanical equipment lists and motor lists.

The E&I works enquiry was issued to suitable contractors currently based in Mali and surrounding countries, including contractors from South Africa. The enquiry included P&G cost lists and contractual conditions against which tenderers quoted. Fully inclusive “wet rates” and plant/labour histograms were received from the respective tenderers, and these were compared and adjudicated accordingly. The contractor will provide accommodation for their own workforce and will be completely self-sufficient.

21.2.3.11.2 SMPP Works Contract

The process plant and peripheral infrastructure SMPP works main contract scope was based on a detailed BOQ derived from general arrangement drawings, piping and instrumentation drawings and details of the free-issue mechanical equipment.

The SMPP works enquiry was issued to suitable contractors currently based in Mali and surrounding countries, including contractors from South Africa. The enquiry included P&G cost lists and contractual conditions against which tenderers quoted. Fully inclusive “wet rates” and plant/labour histograms were received from the respective tenderers, and these were compared and adjudicated accordingly. The contractor will provide accommodation for their own workforce and will be completely self-sufficient.

The bill of quantities for the Oxides and Sulphides circuit were quantified based on plant sections and structures from this previously completed Gold Project.

The BOQ, together with the specifications, equipment schedules, process flow diagrams and layout drawings formed the basis for procurement enquiries.

The procurement enquiries were issued to the respective contractors, companies that are capable of mobilising, constructing, installing, and demobilising required resources in Mali.

Their tender proposals were reviewed and adjudicated against commercial, technical and schedule aspects.

Finally, selected Contractor for the Oxides plant: ATC.

Finally, selected Contractor for the Sulphides plant: ATC.

21.2.3.12 Tools and Mobile Equipment

Escalation was applied to the enquiries received in 2020 DFS; this results in an increase of costs of 14 %. See Table 21.15 for a comparison of the 2020 vs 2021 Tools and mobile Equipment DFS CAPEX.

Table 21.15: 2020 vs 2021 DFS Tools and Mobile Equipment CAPEX Comparison

Description	CAPEX 2020 (US$)	CAPEX 2021 (US$)	Difference (US$)
Tools and Mobile Equipment	1,235,126	1,402,909	167,784

21.2.4 Indirect Field Costs

Figure 21.3 represents the increase in the indirect field cost from the 2020 DFS to 2021 DFS.

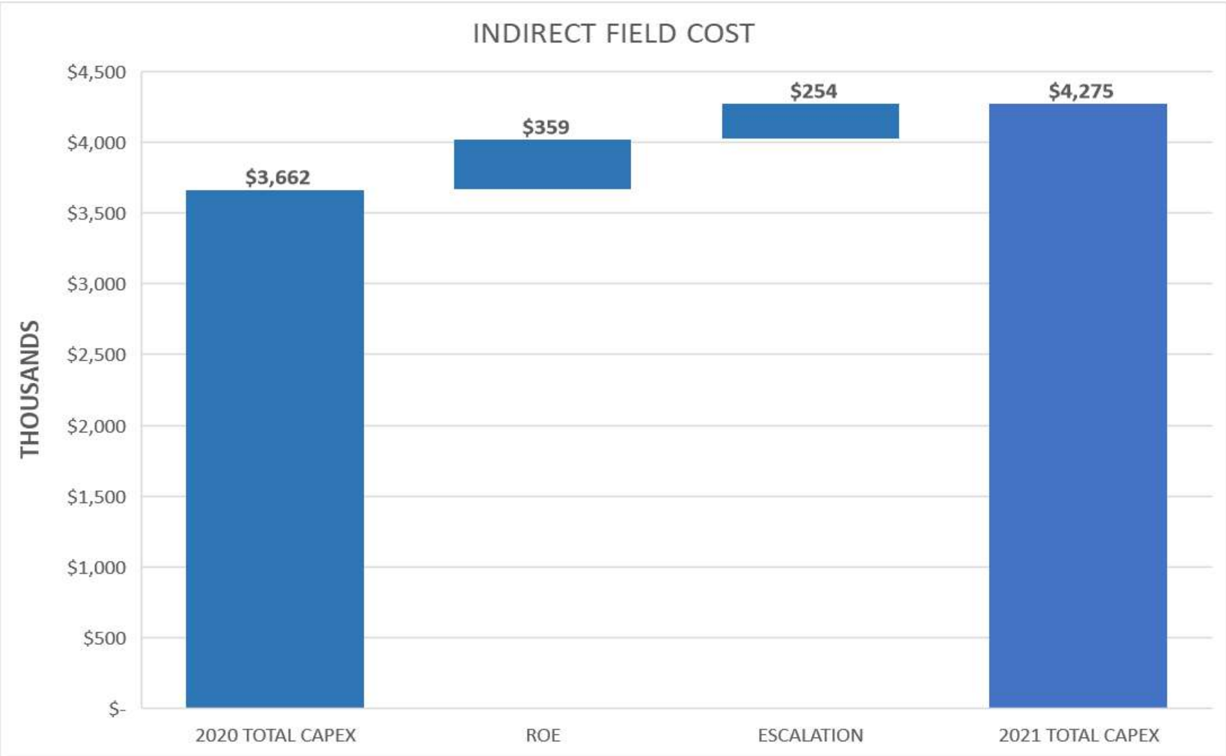


Figure 21.3: Indirect Field Costs

All the indirect field costs increased by 17% because of the Mechanical equipment packages increases. See Table 21.16 for a comparison of the 2020 vs 2021 Indirect Field Costs DFS CAPEX.

Table 21.16: 2020 vs 2021 DFS Indirect Field Costs CAPEX Comparison

Description	CAPEX 2020 (US$)	CAPEX 2021 (US$)	Difference (US$)
Commissioning Spares	174,799	221,119	46,320
2-Year Operational Spares	1,016,002	892,563	-123,439
Insurance and Critical Spares	975,532	1,579,459	603,927
Vendor Services	793,555	857,311	63,756
First Fills	702,427	724,138	21,711

21.2.4.1 Spares

Three categories of spares were considered and included in the CAPEX namely:

- Commissioning spares.
- Two-year operating spares.
- Strategic or insurance spares.

21.2.4.2 Vendor Services

The cost for vendor services includes all the items where the presence of the vendor is required during the construction phase for guarantees to be honoured. It also includes items where construction supervision is required, particularly for the installation of the large and/or critical equipment items. The costs are based on actual quotes obtained from the respective vendors.

21.2.4.3 First Fills

The first-fill costs were developed from first principles. These were defined as those costs incurred prior to commissioning in preparing the circuit to accept ore. These costs included the addition of steel balls (various sizes) to the ball mill to design charge levels, which will constitute a graded charge, and the addition of carbon at 12 g/L to the CIL circuit, as well as filling the elution heating system with thermal oil.

21.2.5 Home Office Costs

Figure 21.4 represents the home office cost increase from the 2020 DFS to 2021 DFS.

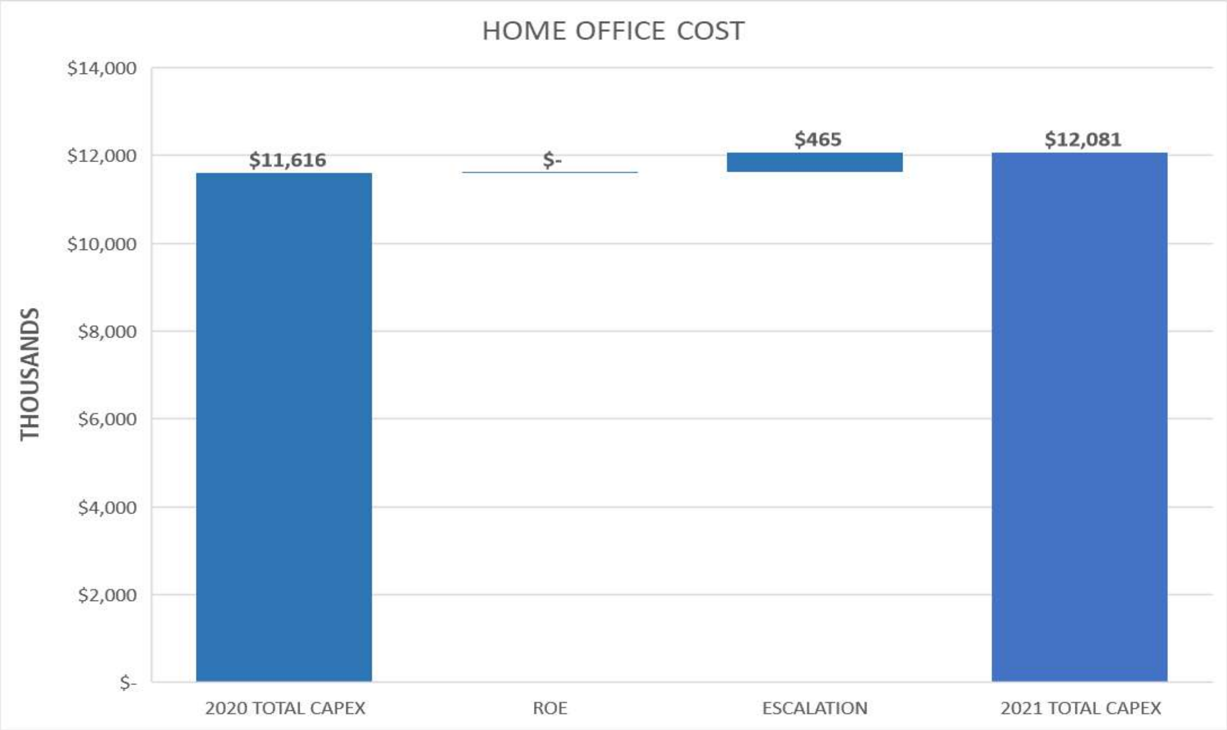


Figure 21.4: Home Office Costs

Table 21.17: 2020 vs 2021 DFS Home Office CAPEX Comparison

Description	CAPEX 2020 (US$)	CAPEX 2021 (US$)	Difference (US$)
Project Management (EPCM)	9,410,700	9,496,671	85,971
Insurances and Guarantees	435,600	427,680	-7,920
Project Taxes	1,769,420	2,156,689	387,269

The difference in Table 21.17 is mainly due to the rate of exchange impact.

21.2.5.1 Engineering, Procurement and Construction Management (EPCM)

Engineering, project management and drawing office man-hours are based on the estimated number of man-hours required to complete the detailed design of the project. Unit rates for man-hours represent actual rates currently being charged on similar projects.

Site construction management is based on a highly skilled team of engineers and site staff who will supervise the construction crew's activities. This part of the estimate assumes that construction will be subcontracted to earthworks, civil works, SMPP and E&I construction companies. This, however, requires a higher level of supervision on the part of the EPCM Contractor and Owner's representative.

21.2.5.2 Insurances, Guarantees and Project Taxes

The EPCM Contractor will be responsible for the necessary insurance related to workmen's compensation for their supervisory personnel on site, where required by law. Allowance has been made for the professional indemnity insurance premium, based on the full execution value of the plant upgrade.

Allowance has been made for all taxes, including offshore services withholding tax payable in Mali.

21.2.6 Other Cost

21.2.6.1 Tailings Storage Facility (TSF)

21.2.6.1.1 Summary of Cost Estimate

The CAPEX associated with the TSF has been determined to a Class 2/3 AACE accuracy of (+25 %, -15 %) based on quantities measured by Epoch and rates sourced from earthworks and liner tender enquiries undertaken and provided by SENET.

The CAPEX estimate has been apportioned across the LOM for each phase of the TSF, with the first Phase 1A as the initial CAPEX, and the sustaining CAPEX comprising the remaining phases. Phasing of the facility and the associated CAPEX over the LOM allows for the deferral of costs into sustained CAPEX, thus improving the initial, LOM cash flow requirements, internal rate of return and net present value of the project.

Table 21.18 presents the initial and sustaining TSF CAPEX that occurs over the LOM. The P&Gs are set to 26.7 % of the measured works based on the original 2020 BFS quantities. No contingencies have been allowed for and this is catered for under an overall project contingency elsewhere.

Provision has been made to construct compacted walls and fills with selected and suitable material from approved borrow pits, excavations, stockpiles and compact to required specifications, assuming all suitable materials to be available within a 2-kilometer free-haul distance.

Table 21.18: Initial and Sustained Capital Expenditure for the Kobada Graphite Mine TSF

Description	Amount (in Million US$)					
	Phase 1A Initial & Sustained CAPEX					
Schedule	Phase 1A	Phase 1B	Phase 2	Phase 3	Phase 4	Phase 5
Site Clearance	1,583,798	0	403,531	389,119	298,723	228,988
Earthworks and Excavations	5,681,079	3,085,256	6,234,915	7,381,229	6,368,268	6,670,770
Drainage	13,026,446	784,598	3,190,656	3,053,082	2,370,143	1,843,287
Concrete Structures	429,010	0	329,215	324,839	324,839	324,839
Pipe Work	1,042,464	213,223	2,885,801	672,539	531,914	672,539
Gabions	143,454	0	143,454	143,454	143,454	143,454
Catwalk	12,609	0	0	0	0	0
Warning Signage and Safety	346,667	0	0	0	0	0

Description	Amount (in Million US$)					
Phase 1A Initial & Sustained CAPEX						
Schedule	**Phase 1A**	**Phase 1B**	**Phase 2**	**Phase 3**	**Phase 4**	**Phase 5**
Miscellaneous	263,404	0	101,572	105,305	105,305	105,305
Total Measured Works	**22,528,930**	**4,083,078**	**13,289,143**	**12,069,567**	**10,142,648**	**9,989,182**
Preliminary and Generals	**6,015,224**	**1,090,182**	**3,548,201**	**3,222,574**	**2,708,087**	**2,667,112**
Total Expenditure per Phase	**28,544,154**	**5,173,259**	**16,837,344**	**15,292,141**	**12,850,735**	**12,656,294**
Total Expenditure						**91,353,927**

Figure 21.5 provides the percentage split of the initial CAPEX associated with TSF Phase 1A, with the earthworks (25%) and drainage (including HDPE lining and geotextile) (58%) making up the predominant costs, with the remaining items collectively amounting to 17% of the total initial TSF CAPEX.

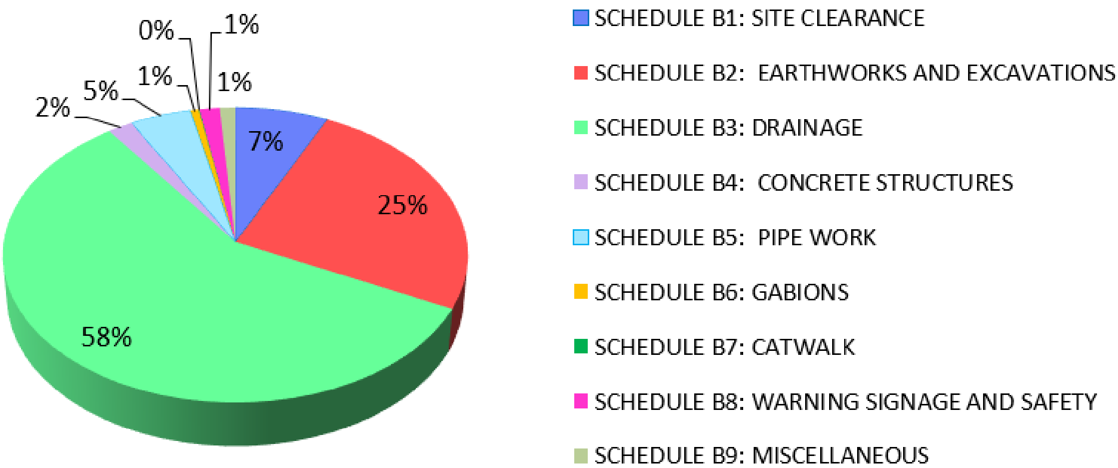


Figure 21.5: Phase 1A Capital Expenditure % Split

21.2.6.2 Haul Roads

A BOQ for the site access roads and mine haul roads earthworks has been prepared in accordance with SANS 1200 A and SANS 1200 D, using the topographical drawings, plant location, TSF location and mine planning designs as a basis for measurement.

The road works comprise the following:

- Construction of the main haul road, pit haul roads (Nos 1 to 11 inclusive) and haul road culverts as per mine plan
- Construction of main access roads to process plant and TSF
- Provisional sum allowed for maintenance of site access roads during construction period
- Construction of a low-level bridge crossing at the Fié River (Sélingué route to Bamako)

Original bidders from the 2020 DFS was approached for updated quotations and commercially and technically adjudicated.

21.2.6.3 Owners Pre-Production Cost

The Owner's pre-production costs are based on costs that will be incurred from the start of the project implementation phase up to the commissioning and handover to plant operations.

The Owner's pre-production costs comprise the following:

- General and administration salaries, including the Owner's project team; the health, safety and environmental (HSE) department; finance; procurement; and human resources
- Mining department labour costs prior to commencement of pre-stripping
- Plant and laboratory labour costs prior to commencement of plant commissioning
- Costs associated with the administration of an off-site office
- Camp catering costs
- Training package implementation and contractor engagement
- Vehicle running and maintenance costs
- Other administrative support costs

21.2.6.3.1 Pre-Production Labour

The pre-production labour cost for the 18-month construction period includes the following:

- Pre-production labour salaries
- Pre-production labour flights
- Dedicated vehicle costs (diesel and maintenance)
- Recruitment costs

21.2.6.3.2 Other Pre-Production Costs

The other pre-production costs for the 18-month construction period include the following, which were based on the general and administration costs:

- Camp food costs
- Facilities maintenance
- Off-site offices and travel costs
- Supplies and spare parts
- Security
- Other administration costs
- Environmental and social costs

21.2.6.4 Working Capital

Due to change in gold price the working capital could reduce as the price of gold had increased from the 2020 DFS date till 2021 DFS.

Working capital was defined as those fixed and variable costs incurred by the mine from commissioning to the point where the mine is cash flow positive and the revenue from gold sales can pay for the mine's operational costs.

The working capital cost has been calculated from first principles, estimating the ramp-up period (period for plant to reach design production capacity) of two months. In this calculation, the following costs were considered:

- Operating costs for the whole operation, i.e. mining, process plant, and TSF
- General and administration costs
- Mining and process plant assay costs
- Refining costs
- Stockholding costs
- Gold lockup costs

Table 21.19 provides a comparison of 'other costs' from the 2020 DFS to 2021 DFS.

Table 21.19: 2020 vs 2021 DFS Other Costs CAPEX Comparison

Description	CAPEX 2020 (US$)	CAPEX 2021 (US$)	Difference (US$)
TSF	21,047,828	29,425,331	8,377,503
Mining Pre-Production	23,983,233	23,574,882	-408,351
Mining Establishment	3,872,000	3,872,000	-
Haul Roads	744,594	804,162	59,568
IFC PS Readiness Assessment		384,366	384,366
Owners Pre-Production Cost	8,263,660	8,263,660	-
Resettlement Action Plan	965,503	6,166,640	5,201,137
TOTAL OTHER COST	**58,876,818**	**72,491,040**	**13,614,223**

21.3 OPERATING COSTS

21.3.1 Summary of Operating Costs

The purpose of this operating cost (OPEX) estimate is to provide operating costs, and the associated general and administrative (G&A) costs, to an accuracy of +15 % −10 % that can be utilised for the economic analysis of the Kobada Gold Project.

The project's annual OPEX estimate for the LOM consists of the following:

- Mining operating costs estimated by DRA Mining
- Process plant and raw water supply operating costs estimated by SENET
- TSF operating costs estimated by Epoch
- Site G&A costs estimated by SENET
- Bullion transport, insurance and refining costs estimated by SENET

The overall LOM OPEX for the Kobada Gold Project is summarised in Table 21.20 with the cost distribution shown in Figure 21.6.

Table 21.20: Overall LOM Plant OPEX Summary

Description	LOM		
	US$/t processed	US$/oz	Distribution
Mining	11.25	421.24	47.81 %
Processing	8.55	320.39	36.36 %
G&A	2.23	83.34	9.46 %
Refining and Transport	0.10	3.70	0.42 %
Royalties	1.40	52.46	5.95 %
Total	**23.53**	**881.13**	100 %

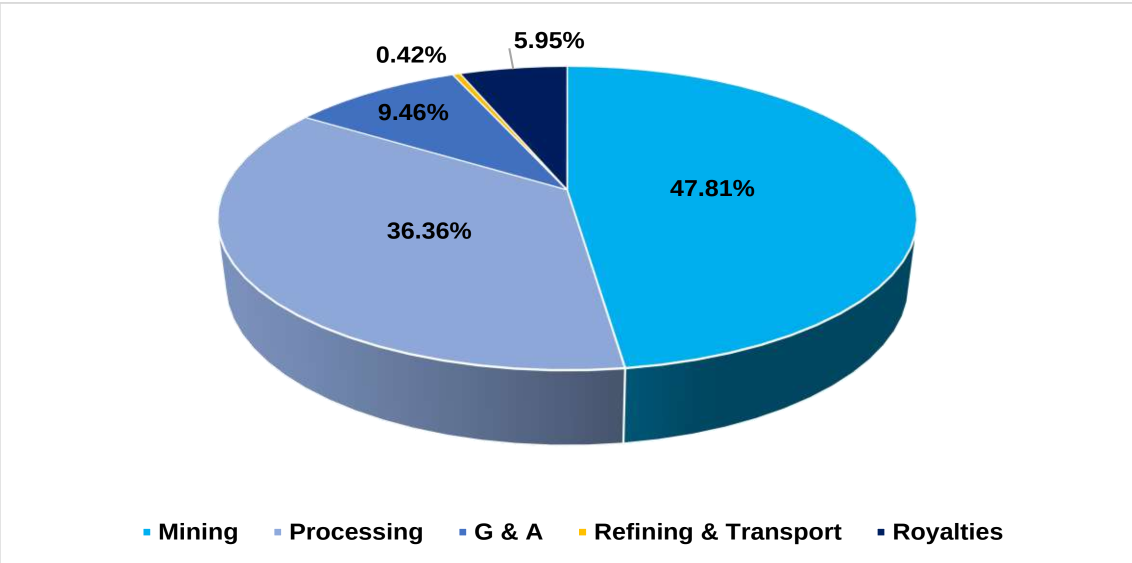


Figure 21.6: OPEX Distribution

The scheduled start of production is July 2023. The monthly plant operating cost for the first year of production ramp-up and the annual process plant operating costs for the LOM for the Project are summarised in Table 21.21 and Table 21.22, respectively.

Royalties have not been included in these tables and are addressed in Section 22.

Details of the operating costs are shown in the sections that follow.

Table 21.21: Year 1 (2023) Monthly OPEX Summary

Description	Unit	Year 1 Totals	Jul-23	Aug-23	Sep-23	Oct-23	Nov-23	Dec-23
			Month 1	Month 2	Month 3	Month 4	Month 5	Month 6
Mining	US$	23,386,714	4,162,592	3,767,348	3,799,819	3,819,034	3,841,266	3,996,655
Process plant, TSF and assay	US$	11,080,011	1,207,607	1,947,813	1,996,203	1,973,834	1,976,963	1,977,590
G&A	US$	3,313,495	552,249	552,249	552,249	552,249	552,249	552,249
Bullion transport, insurance and refining	US$	186,291	30,055	31,404	31,093	31,324	31,169	31,245
Government Royalties	US$	2,641,185	440,197	440,197	440,197	440,197	440,197	440,197
TOTAL	US$	38,846,906	6,392,702	6,739,011	6,379,364	6,376,442	6,401,647	6,557,739
Plant Throughput	Mt	1,377,348	125,871	245,969	253,820	250,190	250,698	250,800
Total Cost	**US$/t feed**	**28.20**	**50.79**	**27.40**	**25.13**	**25.49**	**25.54**	**26.15**

Table 21.22: LOM OPEX Summary

Description	Unit	LOM	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
			Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
Ore Processed																		
Throughput - Oxide (Saprolite)	t	28,290,947	1,041,199	2,704,187	2,842,214	2,783,174	2,659,879	2,742,496	2,338,542	2,054,526	515,617	3,885	1,868,248	2,873,879	2,727,521	1,135,275	304	0
Throughput - Oxide (Laterite)	t	1,624,004	336,145	292,341	145,900	31,360	58,988	27,647	1,482	2,405	101,029	0	32,780	109,602	188,316	280,777	307	14,923
Throughput - Transition	t	3,751,735	4	403	18,849	91,656	210,736	193,595	620,950	972,921	41,951	97,828	27,270	0	2,233	902,773	499,584	70,982
Throughput - Sulphide	t	11,362,145	0	0	0	0	0	0	0	0	2,337,024	2,957,126	1,028,781	14,032	14	712,062	2,540,700	1,772,407
Total Plant Throughput	t	45,028,831	1,377,348	2,996,931	3,006,964	2,906,190	2,929,603	2,963,738	2,960,975	3,029,853	2,995,620	3,058,838	2,957,078	2,997,514	2,918,084	3,030,887	3,040,896	1,858,313
Au Feed Grade																		
Au Grade - Saprolite	g/t	0.88	1.14	1.10	1.09	1.09	1.09	1.12	1.10	0.85	0.50	0.74	0.48	0.46	0.45	0.43	0.35	0.00
Au Grade - Laterite	g/t	0.85	1.29	1.14	1.12	0.99	0.95	1.08	0.74	0.66	0.46	0.00	0.48	0.46	0.45	0.45	0.48	0.45
Au Grade - Transition	g/t	0.87	,	0.45	0.89	1.26	1.17	1.16	1.20	1.07	1.12	1.16	1.04	0.56	0.55	0.47	0.44	0.41
Au Grade - Sulphide	g/t	0.84	,	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.33	1.04	0.97	0.80	0.80	0.49	0.46	0.44
Au Grade - Average	g/t	0.85	1.18	1.10	1.09	1.10	1.09	1.12	1.12	0.92	1.16	1.05	0.66	0.46	0.45	0.46	0.46	0.44
OPEX																		
Mining	US$	506,451,220	23,386,714	52,350,160	56,462,505	58,310,600	56,530,874	61,965,452	65,533,889	65,952,048	29,324,836	14,196,783	16,239,315	1,470,094	1,325,127	1,377,299	1,229,327	796,198
Process plant, TSF and assay	US$	385,195,669	11,080,011	23,048,746	23,119,153	22,776,405	22,976,062	23,178,485	23,028,120	23,616,250	30,360,154	32,605,509	26,189,933	23,397,730	22,888,467	26,127,596	31,459,855	19,343,193
G&A	US$	100,196,366	3,313,495	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	4,105,007
Bullion transport, insurance and refining	US$	4,448,421	186,289.66	379,085.73	394,816.47	366,378.39	366,938.42	381,066.50	382,158.12	319,007.78	391,594.59	360,207.10	217,852.70	156,626.43	148,640.30	154,427.15	153,633.78	89,697.81
Government Royalties	US$	63,068,991	2,641,184.56	5,374,616.00	5,597,643.86	5,194,453.47	5,202,393.43	5,402,699.06	5,418,175.91	4,522,840.60	5,551,964.61	5,106,957.86	3,088,680.27	2,220,624.11	2,107,398.09	2,189,443.05	2,178,194.85	1,271,720.98
TOTAL	**US$**	**1,059,360,667**	**40,607,695**	**87,779,599**	**92,201,108**	**93,274,827**	**91,703,257**	**97,554,692**	**100,989,333**	**101,037,137**	**72,255,540**	**58,896,447**	**52,362,772**	**33,872,065**	**33,096,623**	**36,475,755**	**41,648,001**	**25,605,817**
Plant Throughput	t	46,046,270	1,377,348	2,996,931	3,006,964	2,906,190	2,929,603	2,963,738	2,960,975	3,029,853	2,906,190	2,929,603	2,963,738	2,960,975	3,029,853	3,029,853	2,995,620	3,058,838
Total Cost	**US$/t feed**	**23.01**	**29.48**	**29.29**	**30.66**	**32.10**	**31.30**	**32.92**	**34.11**	**33.35**	**24.86**	**20.10**	**17.67**	**11.44**	**10.92**	**12.04**	**13.90**	**8.37**
Total Cost	**US$/oz**	**881**	**34**	**73**	**77**	**78**	**76**	**81**	**84**	**84**	**60**	**49**	**44**	**28**	**28**	**30**	**35**	**21**

21.3.2 Exchange Rates

The costs of the Project are reported in the United States dollar. The exchange rates used are shown in Section 22.

21.3.3 Escalation

No escalation has been allowed for in the OPEX estimate.

21.3.4 Exclusions

The following items were excluded from the OPEX estimate:

- Schedule delays, such as those caused by
  - Scope changes
  - Labour disputes
- Receipt of information beyond SENET's control
- Currency fluctuations
- Force majeure
- Contingencies

21.3.5 Mining Operating Costs

The mining OPEX has been updated based on the planned movements and the selected Mining Contractor's budgetary offer.

The mining OPEX estimate includes the following items:

- Mining Contractor's costs
- Mining Contractor's overhead costs and charges
- Fuel costs
- Grade control drilling costs
- Mine Owner's team manpower costs

The LOM overall OPEX summary is shown in Table 21.23 and the yearly breakdown for the mining OPEX is shown in Table 21.24 and Table 21.25.

Table 21.23: Mining Overall LOM OPEX Summary

Description	Unit	Total OPEX
Total Mine OPEX over the LOM	US$ Million	506
OPEX per tonne mined	US$/t	2.62
OPEX per tonne ore delivered to the mill	US$/t	11.25
OPEX per ounces of gold	US$/oz	422

Table 21.24: Mining OPEX Breakdown Summary (2023 to 2030)

Description	Unit	LOM Total	2023	2024	2025	2026	2027	2028	2029	2030
			Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Contractor's Waste Mining Cost	US$	**208,213,843**	7,930,567	21,193,876	28,194,672	25,871,756	30,939,523	28,481,100	26,716,111	26,394,431
Contractor's Ore Mining Cost	US$	**58,283,762**	5,347,995	9,606,027	5,816,651	8,060,560	3,134,178	6,483,113	8,150,566	6,876,881
ORE Rehandling Costs	US$	**11,871,252**	353,766	556,019	574,360	509,730	866,509	432,227	128,469	106,602
Contractor's Drill and Blast Cost	US$	**36,166,956**	298,640	693,189	1,156,050	2,408,770	1,353,343	4,208,824	6,970,632	9,145,014
Contractor's Overhead Costs and Charges	US$	**47,250,000**	2,250,000	4,500,000	4,500,000	4,500,000	4,500,000	4,500,000	4,500,000	4,500,000
Contractor's Diesel Cost	US$	**110,727,473**	4,925,943	11,541,971	12,986,878	13,164,143	13,408,686	14,291,902	14,936,077	15,202,164
Clearing and Grubbing Costs	US$	**243,109**	26,234	68,249	148,626	0	0	0	0	0
Grade Control Drilling Costs	US$	**18,719,309**	1,570,830	2,825,354	1,719,792	2,411,200	944,195	2,183,845	2,747,595	2,342,515
Manpower (Mine Supervision and Technical Services)	US$	**14,975,516**	682,738	1,365,475	1,365,475	1,384,440	1,384,440	1,384,440	1,384,440	1,384,440
TOTAL	**US$**	**506,451,220**	23,386,714	52,350,160	56,462,505	58,310,600	56,530,874	61,965,452	65,533,889	65,952,048

Table 21.25: Mining OPEX Breakdown Summary (2031 to 2038)

Description	Unit	LOM Total	2031	2032	2033	2034	2035	2036	2037	2038
			Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
Contractor's Waste Mining Cost	US$	**208,213,843**	9,046,436	1,540,585	1,882,458	21,320	21	984	0	0
Contractor's Ore Mining Cost	US$	**58,283,762**	1,492,133	1,401,411	1,872,544	38,830	39	2,835	0	0
ORE Rehandling Costs	US$	**11,871,252**	1,034,877	960,867	831,934	1,193,393	1,167,228	1,211,729	1,216,358	727,182
Contractor's Drill and Blast Cost	US$	**36,166,956**	5,360,558	1,953,112	2,521,936	42,614	43	2,791	0	51,441
Contractor's Overhead Costs and Charges	US$	**47,250,000**	4,500,000	4,500,000	4,500,000	0	0	0	0	0
Contractor's Diesel Cost	US$	**110,727,473**	5,919,063	1,879,210	2,471,436	0	0	0	0	0
Clearing and Grubbing Costs	US$	**243,109**	0	0	0	0	0	0	0	0
Grade Control Drilling Costs	US$	**18,719,309**	587,329	577,158	774,566	16,157	16	1,180	0	17,576
Manpower (Mine Supervision and Technical Services)	US$	**14,975,516**	1,384,440	1,384,440	1,384,440	157,780	157,780	157,780	12,968	0
TOTAL	**US$**	**506,451,220**	29,324,836	14,196,783	16,239,315	1,470,094	1,325,127	1,377,299	1,229,327	796,198

21.3.6 Process Plant Operating Costs

21.3.6.1 Basis of Estimate

The process plant operating costs were compiled from a variety of sources, notably:

- First principles, where applicable
- Supplier quotations on reagents and consumables
- SENET's in-house experience and database where applicable
- Client input

The following are the main cost elements of the process plant:

- Reagents and consumables
- Fuel Farm
- Power
- Process plant operating and maintenance labour
- Maintenance parts and supplies
- Assay

21.3.6.2 Process Plant OPEX Summary

The overall process plant LOM OPEX is summarised in Table 21.26, and the distribution of the costs is shown in Figure 21.7. A yearly breakdown of the process plant OPEX is provided in Table 21.27.

Table 21.26: Process Plant LOM OPEX Summary

Description	LOM Cost	LOM Cost
	US$	US$/t feed
Labour	30,022,350	0.67
Fuel Farm	5,672,629	0.13
Power	16,110,313	0.36
Maintenance	7,747,712	0.17
Assay	23,521,885	0.52
Consumables	119,116,421	2.65
Reagents	170,548,723	3.79
TSF	12,455,638	0.28
TOTAL	385,195,669	8.55

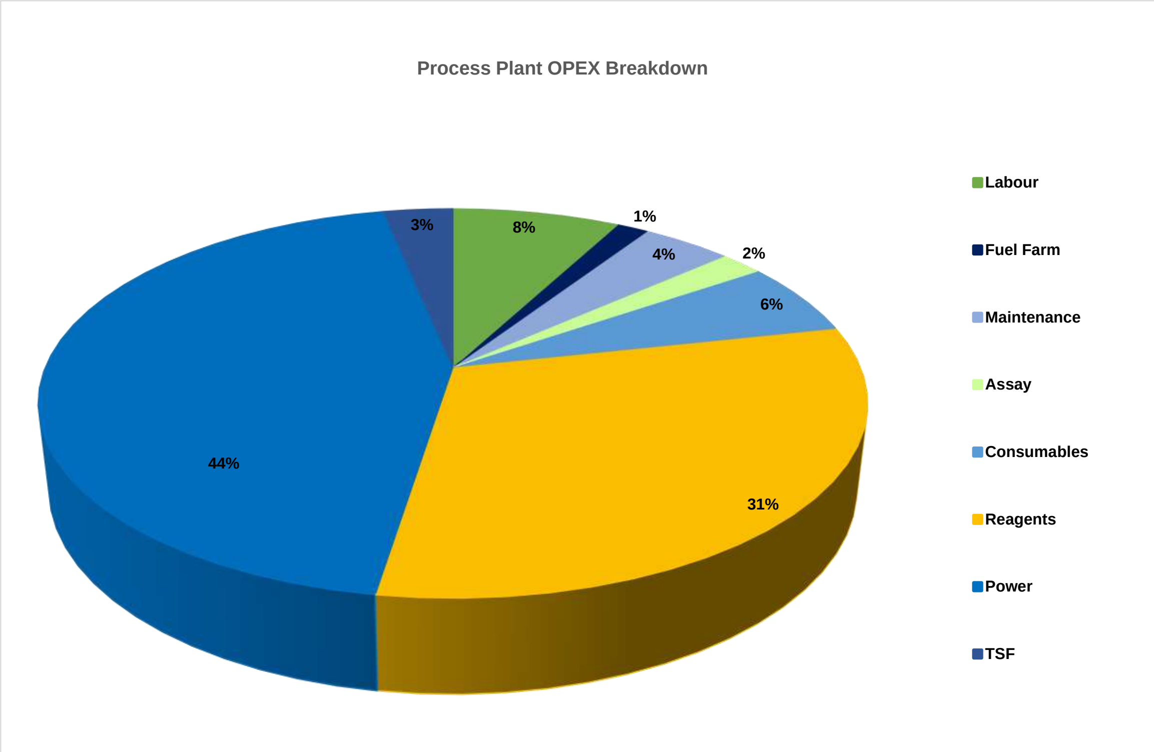


Figure 21.7: Plant OPEX Distribution

Table 21.27: Process Plant LOM OPEX

Description	Unit	LOM	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
			Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
Labour	US$ million	30.02	0.99	1.99	1.99	1.99	1.99	1.99	1.99	1.99	1.99	1.99	1.99	1.99	1.99	1.99	1.99	1.23
Fuel Farm	US$ million	5.67	0.19	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.38	0.23
Maintenance	US$ million	16.11	0.43	0.86	0.86	0.86	0.86	0.86	0.86	0.86	1.27	1.27	1.27	1.27	1.27	1.27	1.27	0.79
Assay	US$ million	7.94	0.26	0.51	0.51	0.51	0.51	0.51	0.51	0.51	0.51	0.51	0.51	0.51	0.51	0.51	0.51	0.51
Total Fixed Cost	**US$ million**	**59.75**	**1.87**	**3.73**	**3.73**	**3.73**	**3.73**	**3.73**	**3.73**	**3.73**	**4.14**	**4.14**	**4.14**	**4.14**	**4.14**	**4.14**	**4.14**	**2.76**
Consumables	US$ million	24	0.62	1.36	1.37	1.33	1.36	1.38	1.44	1.53	1.88	2.05	1.57	1.36	1.32	1.68	2.02	1.24
Reagents	US$ million	119	3.74	8.14	8.17	7.91	8.00	8.09	8.16	8.41	7.34	7.31	7.68	8.14	7.93	8.16	7.48	4.45
Power	US$ million	171	4.12	8.97	9.00	8.71	8.79	8.89	8.94	9.18	16.24	18.36	12.05	9.02	8.74	11.39	17.06	11.08
TSF	US$ million	12.46	0.73	0.85	0.85	1.09	1.09	1.09	0.76	0.76	0.76	0.74	0.74	0.74	0.76	0.76	0.76	0.00
Total Variable Cost	**US$ million**	**325.64**	**9.21**	**19.32**	**19.39**	**19.05**	**19.24**	**19.45**	**19.30**	**19.89**	**26.22**	**28.46**	**22.05**	**19.25**	**18.75**	**21.98**	**27.32**	**16.78**
Total Variable + Fixed OPEX	**US$ million**	**385**	**11.08**	**23.05**	**23.12**	**22.78**	**22.98**	**23.18**	**23.03**	**23.62**	**30.36**	**32.61**	**26.19**	**23.40**	**22.89**	**26.13**	**31.46**	**19.54**
Total Variable + Fixed OPEX	**US$/t feed**	**8.56**	**8.04**	**7.69**	**7.69**	**7.84**	**7.84**	**7.82**	**7.78**	**7.79**	**10.13**	**10.66**	**8.86**	**7.81**	**7.84**	**8.62**	**10.35**	**10.51**

21.3.6.3 Reagents and Consumables

The reagents and consumables costs were calculated by using vendor supply costs together with the consumptions of the respective reagents or consumables based upon test work results. The reagents and consumables supplied costs are shown in Table 21.28.

Table 21.28: Reagents and Consumables Supplied Costs

Description	Unit	Cost
Grinding Media – Primary Mill – 100 mm	US$/t	1,195
Grinding Media – Secondary Mill – 100 mm	US$/t	1,195
Crusher Liners – Stationary Jaw	US$/Set	11,482
Crusher Liners – Swing Jaw	US$/Set	28,125
Secondary Crusher	US$/Set	12,530
Tertiary Crusher	US$/Set	12,530
Mill Liners – Primary Mill	US$/Set	129,498
Mill Liners – Secondary Mill	US$/Set	194,247
Activated Carbon	US$/t	2,400
Copper Sulphate (equivalent to 10 ppm)	US$/t	2,330
Ferrous Sulphate	US$/t	275
Flocculant	US$/t	1,150
Hydrochloric Acid	US$/t	280
Hydrogen Peroxide	US$/t	540
Lime – Quicklime	US$/t	195
Silica	US$/t	350
Sodium Borate (Borax)	US$/t	280
Sodium Carbonate	US$/t	350
Sodium Cyanide	US$/t	2,200
Sodium Hydroxide (Caustic)	US$/t	600
Sodium Metabisulphite	US$/t	350
Diesel	US$/L	0.80

The reagents and consumables LOM consumptions and operating costs are provided in Table 21.29 and Table 21.30, respectively.

Table 21.29: Reagents and Consumables Consumptions

Description	LOM	2023 Year 1	2024 Year 2	2025 Year 3	2026 Year 4	2027 Year 5	2028 Year 6	2029 Year 7	2030 Year 8	2031 Year 9	2032 Year 10	2033 Year 11	2034 Year 12	2035 Year 13	2036 Year 14	2037 Year 15	2038 Year 16
	t	t/a	t/a	t/a	t/a	t/a	t/a	t/a	t/a	t/a	t/a	t/a	t/a	t/a	t/a	t/a	t/a
Grinding Media – 100 mm	11,998	331	719	723	704	718	725	754	794	913	983	796	721	700	848	973	596
Grinding Media – 50 mm	6,499	0	0	0	0	0	0	0	0	1,337	1,691	588	8	0	407	1,453	1,014
Primary Crusher Liners	79	2	5	5	5	5	5	5	5	6	7	5	5	5	5	7	4
Secondary Crusher Liners	43	0	0	0	0	0	0	0	0	9	11	4	0	0	3	10	7
Tertiary Crusher Liners	42	0	0	0	0	0	0	0	0	9	11	4	0	0	3	9	7
Mill Liners – Primary Mill	999	18	40	40	40	42	43	50	57	110	130	70	40	39	77	125	79
Mill Liners – Secondary Mill	864	0	0	0	0	0	0	0	0	178	225	78	1	0	54	193	135
Activated Carbon	1,126	34	75	75	73	73	74	74	76	75	76	74	75	73	76	76	46
Copper Sulphate	3,189	103	223	224	216	218	221	220	226	189	185	205	223	217	215	190	113
Ferrous Sulphate	8,286	256	557	560	542	549	555	562	582	525	529	536	557	543	571	539	322
Flocculant	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Hydrochloric Acid	3,316	101	221	221	214	216	218	218	223	221	225	218	221	215	223	224	137
Hydrogen Peroxide	521	16	35	35	34	34	34	34	35	35	35	34	35	34	35	35	21
Lime – Quicklime	38,288	1,253	2,727	2,738	2,653	2,685	2,715	2,751	2,846	2,107	2,004	2,419	2,724	2,656	2,650	2,135	1,225
Silica	10	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Sodium Borate (Borax)	20	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Sodium Carbonate	10	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Sodium Cyanide	18,030	551	1,199	1,204	1,167	1,181	1,194	1,210	1,252	1,172	1,191	1,172	1,199	1,168	1,240	1,205	725
Sodium Hydroxide (Caustic)	2,116	65	141	141	137	138	139	139	142	141	144	139	141	137	142	143	87
Sodium Metabisulphite	23,544	759	1,651	1,656	1,601	1,614	1,633	1,631	1,669	1,391	1,357	1,515	1,650	1,607	1,591	1,393	827
Diesel for Plant Use	4,665	143	310	311	301	303	307	307	314	310	317	306	311	302	314	315	193

Table 21.30: Summary of Reagents and Consumables LOM OPEX

Description	LOM	2023 Year 1	2024 Year 2	2025 Year 3	2026 Year 4	2027 Year 5	2028 Year 6	2029 Year 7	2030 Year 8	2031 Year 9	2032 Year 10	2033 Year 11	2034 Year 12	2035 Year 13	2036 Year 14	2037 Year 15	2038 Year 16
	US$ '000	US$ '000	US$ '000	US$ '000	US$ '000	US$ '000	US$ '000	US$ '000	US$ '000	US$ '000	US$ '000	US$ '000	US$ '000	US$ '000	US$ '000	US$ '000	US$ '000
Grinding Media – 100 mm	18,937	522	1,135	1,141	1,111	1,133	1,144	1,189	1,254	1,442	1,552	1,256	1,137	1,106	1,339	1,535	941
Grinding Media – 50 mm	16,033	0	0	0	0	0	0	0	0	3,298	4,173	1,452	20	0	1,005	3,585	2,501
Primary Crusher Liners	1,599	49	106	106	103	103	105	105	108	108	110	105	106	103	108	110	67
Secondary Crusher Liners	165	0	0	0	0	0	0	0	0	34	43	15	0	0	10	37	26
Tertiary Crusher Liners	209	0	0	0	0	0	0	0	0	43	54	19	0	0	13	47	33
Mill Liners – Primary Mill	2,985	54	118	120	119	127	127	150	171	328	389	210	119	115	231	372	235
Mill Liners – Secondary Mill	16,033	0	0	0	0	0	0	0	0	3,298	4,173	1,452	20	0	1,005	3,585	2,501
Activated Carbon	3,133	96	209	209	202	204	206	206	211	208	213	206	209	203	211	212	129
Copper Sulphate	8,662	279	606	608	588	593	600	599	613	514	503	558	606	590	585	515	306
Ferrous Sulphate	8,401	260	565	567	550	556	563	570	590	533	537	544	565	550	578	547	327
Flocculant	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Hydrochloric Acid	2,210	68	147	148	143	144	145	145	149	147	150	145	147	143	149	149	91
Hydrogen Peroxide	475	15	32	32	31	31	31	31	32	32	32	31	32	31	32	32	20
Lime – Quicklime	25,228	826	1,797	1,804	1,748	1,769	1,789	1,813	1,875	1,388	1,320	1,594	1,795	1,750	1,746	1,407	807
Silica	3	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Sodium Borate (Borax)	6	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Sodium Carbonate	3	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Sodium Cyanide	47,760	1,461	3,178	3,191	3,092	3,129	3,164	3,206	3,316	3,101	3,153	3,104	3,178	3,095	3,285	3,191	1,917
Sodium Hydroxide (Caustic)	1,881	58	125	126	121	122	124	124	127	125	128	124	125	122	127	127	78
Sodium Metabisulphite	16,978	547	1,190	1,194	1,154	1,164	1,177	1,176	1,203	1,003	979	1,092	1,190	1,159	1,147	1,005	596
Diesel for Plant Use	4,374	134	291	292	282	285	288	288	294	291	297	287	291	283	294	295	181

21.3.6.3.1 Crusher Liners

The primary crusher liner costs were obtained from vendor information, by estimating the number of liner changes per annum using the abrasion indices obtained from metallurgical tests and the expected liner life for a given throughput. Quotations for the crusher liners, including the weights of the liners, were obtained from a vendor. The estimated delivered costs (using the customs clearance, port handling and transport costs to site obtained from Bolloré Logistics) were calculated and used together with the liner supply cost.

21.3.6.3.2 Mill Liners

The ball mill liner costs were based on estimating the liner consumption by using the abrasion index results obtained from test work. OMC used the test work data to simulate the expected wear rates. This was further cross-referenced by wear rates from other operating mines using the same type of liners and grinding media size based on SENET's experience. The current pricing for a set of rubber liners for the ball mill was obtained from a liner supply vendor and used in the cost estimate based on the number of liner changes per annum for a given throughput. The delivered costs were estimated using the customs clearance, port handling and transport costs to site obtained from Bolloré Logistics.

21.3.6.3.3 Mill Grinding Media

The grinding media costs were obtained by estimating the consumption in the ball mill based on Bond's estimating method and using the standard method abrasion index results that were obtained from laboratory tests. OMC used the test work data to estimate the expected mill grinding media consumptions. In addition, the mill throughputs and quotations for 100 mm balls were obtained from suppliers. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the grinding media costs.

21.3.6.3.4 Quicklime

Quicklime consumption in CIL leaching was estimated from test work. The quicklime consumption for cyanide destruction was estimated from test work results and industry practice. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the quicklime costs.

21.3.6.3.5 Sodium Cyanide (Cyanide)

The cyanide consumptions for the CIL and ILR circuits were estimated from test work results. The cyanide consumption associated with elution was calculated from first principles, taking into account the designed consumption usage in these three circuits over a given period, and was then used, together with quotations obtained from reagent suppliers, to estimate the delivered cyanide costs.

21.3.6.3.6 Activated Carbon

Carbon consumption was based on industry practice for CIL plants. Budget quotations obtained from suppliers were then used to estimate the carbon costs per annum for a given throughput.

21.3.6.3.7 Copper Sulphate

The amount of copper sulphate used for the purposes of cyanide destruction (in plant tails slurry and tails return water) was estimated from test work results and first principles. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the copper sulphate costs.

21.3.6.3.8 Sodium Metabisulphite

The amount of sodium metabisulphite used for the purposes of cyanide destruction (in plant tails slurry) was estimated from test work results and first principles. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the sodium metabisulphite costs.

21.3.6.3.9 Ferrous Sulphate

The amount of ferrous sulphate used for the purposes of arsenic precipitation (in plant tails slurry) was estimated from test work results and first principles. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the ferrous sulphate costs.

21.3.6.3.10 Hydrogen Peroxide

The design is based on hydrogen peroxide at a strength of 60 % being delivered in intermediate bulk containers, from which it can be pumped into the storage tank using a drum pump. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the hydrogen peroxide costs.

21.3.6.3.11 Hydrochloric Acid

The hydrochloric acid (33 %) consumption associated with acid washing was calculated from first principles, taking into account the designed number of acid washes and expected acid strength in the solution. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the hydrochloric acid costs.

21.3.6.3.12 Sodium Hydroxide (Caustic)

Caustic consumption associated with elution/electrowinning and intensive cyanidation was calculated from first principles taking into account the following parameters:

- Designed number of elutions
- Volume of gravity concentrates to be treated
- Expected caustic strength in these solutions

Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the caustic costs.

21.3.6.3.13 Smelting Reagents

Silica, sodium borate (borax) and sodium carbonate are reagents used in the smelting process.

The reagents consumptions were determined from first principles using the number of smelts per month, a flux to sludge ratio of 1:1, and the following flux ratios:

- Borax: 50 %
- Silica: 25 %
- Sodium carbonate: 25 %

Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate smelting reagents costs per annum for a given throughput.

21.3.6.3.14 Flocculant for ILR

The flocculant consumption in the ILR was calculated based on the consumption of 125 g of flocculant per batch of ILR, which was an assumption based on SENET's projects database.

21.3.6.3.15 Plant Diesel

The diesel consumption associated with elution (for the thermic oil heaters) and regeneration was calculated from first principles, taking into account the designed number of elutions and regenerations, and the diesel consumptions specified by the vendors for the respective pieces of equipment. The number of elutions was estimated based on the feed gold grades provided by DRA Mining. The diesel cost excludes the fuel associated with in-plant vehicle use. The diesel supply cost of US$0.56/L, together with the consumption, was then used to estimate the plant diesel costs.

21.3.6.4 Fuel Farm

The annual fuel farm storage cost for plant diesel and power plant storage was estimated based on vendor quotations for managing and maintaining the fuel farm.

21.3.6.5 Power

The power consumption can be categorised into two forms: fixed and variable power. The average continuous fixed power consumption was determined by taking into account the installed power rating of each of the equipment in the plant and infrastructure, excluding standbys and the projected running times. The fixed power draw includes the absorbed operating loads associated with the process plant equipment as detailed in the mechanical equipment list and on-site infrastructure, including the following buildings:

- Sewage Treatment Plant
- Fuel Farm
- Change House
- First-Aid Building
- Plant Offices
- Assay Laboratory
- Administration Building
- Weighbridge
- Control Room
- Gatehouse
- Warehouse

- Reagents Stores
- Ball Storage and Bunker
- Workshop

The variable power was estimated using the gross specific energy reported by OMC as detailed in the process design criteria (see Section 6) This mill-specific energy was used with the mill throughput to calculate the annual power usage (kilowatt hours per annum).

The power draw and power costs for the Kobada plant and infrastructure were determined using the power consumption basis detailed above and the unit energy cost of US$0.157/kWh.

Table 21.31 and Table 21.32 show the power draw summaries for fixed and variable power, respectively.

Table 21.31: Power Draw Summary – Fixed Power

Process Area	Process Area Description	Oxides			Sulphides		
		Installed Power	Operating Power	Power Consumption	Installed Power	Operating Power	Power Consumption
		kW	kW	kWh/a	kW	kW	kWh/a
1000	Infrastructure	409	186	1,154,101	409	186	1,154,101
2200	Primary Crushing, Stockpile and Reclaim (Excludes Crusher)	484	377	2,198,711	484	377	2,198,711
2300	Secondary and Tertiary Crushing (Excludes Crusher)				641	529	2,915,790
3100	Milling (Excludes Mill)	1,276	893	3,023,182	1,276	893	3,023,182
3100	Secondary Milling (Excludes Mill)				312	179	1,408,395
3300	Gravity and Intensive Cyanidation (Excludes Crusher)	95	67	459,827	95	67	459,827
3500	Trash Handling	22	13	104,069	22	13	104,069
4100	CIL	932	652	3,929,047	932	652	3,929,047
5100	Carbon Safety, Detoxification and Arsenic Precipitation	1,962	1,314	6,342,660	1,962	1,314	6,342,660
5100	Tailings Disposal				280	210	1,655,640
5500	Tailings Dam and Return Water	500	350	1,517,670	500	350	1,517,670
5600	Secondary Detoxification	9	6	1,967.4	9	6	1,967.4
6100	Acid Wash	36	25	6,232	36	25	6,232
6200	Elution	65	47	167,001	65	47	167,001
6300	Electrowinning	92	64	251,947	92	64	251,947
6400	Carbon Reactivation	50	35	48,118	50	35	48,118
6500	Gold Room	102	71	203,018	102	71	203,018
7100	Cyanide and Caustic	38	27	90,125	38	27	90,125
7200	Lime	7	5	16,590.6	7	5	16,590.6
7400	Detoxification Reagents	36	26	37,317	36	26	37,317
8100	Compressed Air and Diesel Services	574	402	2,026,779	574	402	2,026,779
8300	Raw Water Supply	396	277	740,623	396	277	740,623
8400	Process Water	332	232	866,656	332	232	866,656
8600	Raw Water Distribution	640	440	1,520,940	640	440	1,520,940
	Total	**6,781**	**5,509**	**6,342,660**	**8,013**	**6,427**	**12,322,485**

Table 21.32: Power Draw Summary – Variable Power

Process Area	Process Area Description	Oxides				Sulphides			
		Installed Power	Specific Energy	Operating Power	Power Consumption	Installed Power	Specific Energy	Operating Power	Power Consumption
		kW	kWh/t	kW	kWh/a	kW	kWh/t	kW	kWh/a
2200	Primary Crusher	160	0.13	70	390,000	160	0.20	108	600,000
2300	Secondary Crushing		0.00	0	0	355	0.36	194	1,080,000
2300	Tertiary Crushing		0.00	0	0	710	0.70	378	2,100,000
3100	Primary Ball Mill	2,900	5.80	2,207	17,400,000	2,900	6.30	2,397	18,900,000
3100	Secondary Ball Mill		0.00	0	0	5,000	11.50	4,376	34,500,000
Total		**3,060**	**5.93**	**2,277**	**17,790,000**	**9,125**	**19.06**	**7,453**	**57,180,000**

21.3.6.6 Plant Operating and Maintenance Labour

The estimated annual plant operating and maintenance labour cost was estimated at **US$1,985,679**. The cost was derived from first principles where the actual labour complement for each plant area and maintenance function was identified, and the required number of personnel and their levels were established. The complement derived was then benchmarked against other operations of similar size and complexity.

The operating and maintenance labour cost was broken down as follows:

- Expatriate Personnel:
  - Labour remuneration
  - Medical
  - Income tax as a percentage of net salary
  - In-country taxation
  - Social security
- Mali National Personnel:
  - Labour remuneration
  - Medical
  - Thirteenth cheque
  - Social security
  - Vacation/sick leave
  - Housing allowance

The following costs were excluded as they have been captured in the G&A operating costs:

- Camp food and catering facility
- Expatriate travel
- Safety supplies
- Training
- Consultants' fees

The labour schedule was developed assuming a seven-weeks-on and three-weeks-off roster for expatriate personnel and three eight-hour, three-shift cycles for Mali national personnel.

The salaries for expatriate personnel and Mali national personnel were based on remuneration rates in line with market rates internationally and in Mali, considering the scenario of both qualified and unqualified labour availability in the mine locale. Expatriate personnel will be employed in some managerial and supervisory positions. The remaining positions will be occupied by Mali nationals local to the mine site.

The categories selected for the various positions are described in Table 21.33.

Table 21.33: Process Plant Labour Categories

Employee Category	Description
E1	Expatriate (RSA, Australia, Canada, etc.)
E2	Expatriate (Regional – West Africa)
L	Mali National, local to mine site

Table 21.34 shows the labour cost summary. The detailed breakdown is given in Table 21.35.

Table 21.34: Process Plant Labour Cost Summary

Position Description	Total Number of Employees	Total Cost	Total Cost
		US$/month	US$/a
Plant Management	9	57,966	636,096
TSF and Return Water Ponds – Management	1	15,000	180,000
Subtotal – Management	**10**	**72,966**	**816,096**
Operational Labour	78	93,472	902,916
Plant Maintenance Labour	11	14,853	178,242
TSF and Return Water Ponds Operators	8	7,369	88,425
Subtotal – Plant, Maintenance and TSF Labour	**97**	**115,694**	**1,169,583**
Total Cost	**107**	**188,660**	**1,985,679**

Table 21.35: Detailed Process Plant Labour Cost Breakdown

Position Description	Employee Category	Total Number of Employees	Rate/Employee	Total Cost	Total Cost
			US$/month	US$/month	US$/a
Plant Management					
Plant Manager	E1	1	18,593	18,593	223,116
Production Superintendent	E2	1	7,437	7,437	89,244
Plant Metallurgist	E2	1	4,958	4,958	59,496
Shift Foreman	E2	4	3,646	14,583	174,996
Maintenance Manager	E2	1	7,437	7,437	89,244
Subtotal – Plant Management		**8**		**53,008**	**636,096**
Plant Operational Labour					
Control Room					
Shift Operator	E2	4	1,881	7,523	90,276
Crushing, Mill Feed Storage and Reclaim					
Shift Operator	E2	4	1,215	4,861	58,332
Attendant	L	4	627	2,508	30,096
Milling and Gravity					
Shift Operator	E2	4	1,215	4,861	58,332
Attendant	L	4	627	2,508	30,096
Classification					
Attendant	L	4	627	2,508	30,096
CIL					
Shift Operator	E2	4	1,215	4,861	58,332
Attendant	L	4	627	2,508	30,096
Acid Wash, Elution, Regeneration and Electrowinning					
Shift Operator	E2	4	1,215	4,861	58,332
Attendant	L	4	627	2,508	30,096
Carbon Safety and Detoxification					
Attendant	L	4	627	2,508	30,096

Position Description	Employee Category	Total Number of Employees	Rate/Employee	Total Cost	Total Cost
			US$/month	US$/month	US$/a
Gold Room					
Gold Room Supervisor	E1	1	7,437	7,437	89,244
Day Shift Operator	E2	2	1,215	2,430	29,160
Attendant	L	2	627	1,254	15,048
Sample Preparation (Grade Control and Plant)					
Attendant	L	8	627	5,016	60,192
Reagents					
Attendant	L	4	627	2,508	30,096
Forklift Driver/Operator	E2	4	1,215	4,861	58,332
Other					0
Overhead Crane Operator	E2	4	1,215	4,861	58,332
Light Motor Vehicle Driver	E2	4	1,215	4,861	58,332
Subtotal – Plant Operational Labour		**79**		**98,430**	**902,916**
Plant Maintenance Labour					
Fitter	E2	3	1,215	3,646	43,749
Boilermaker	E2	2	1,215	2,430	29,166
Boilermaker Attendant	L	2	627	1,254	15,047
Electrician	E2	2	1,881	3,762	45,140
Instrumentation Technician	E2	2	1,881	3,762	45,140
Subtotal – Plant Maintenance Labour		**11**		**14,853**	**178,242**
TSF and Return Water Ponds					
TSF Superintendent	E1	1	15,000	15,000	180,000
Operator	L	4	1,215	4,861	58,332
Attendant	L	4	627	2,508	30,093
Subtotal – TSF Labour		**9**		**22,369**	**268,425**
TOTAL		**107**		**188,660**	**1,985,679**

21.3.6.7 Maintenance Parts and Supplies

The plant maintenance parts and supplies annual costs for the Kobada Project were estimated at **US$857,776** when treating oxide ore and **US$1 270,041** when treating sulphide ore. Plant maintenance and supplies costs refer to the costs of operating spares, lubricants, and other maintenance-related consumables for the plant. It has been assumed that the plant will experience a moderate amount of wear. An average annual cost was calculated using the maintenance cost factors as shown in Table 21.36 for the various commodities (see Table 21.37).

Table 21.36: Plant Maintenance Cost Factors

Description	Maintenance Factor (%)
Mechanical Equipment Cost Base Case	5.0
Piping and Valves	2.5
Electricals	2.5
Instrumentation	1.0

Table 21.37: Plant Maintenance, Parts and Supplies OPEX

Description	Unit	Oxide	Sulphide
		Quantity	Quantity
Machinery and Equipment			
Mechanical Equipment Capital Cost	US$	12,383,420	20,225,024
Factor	%	5.0%	5.0%
Total Annual Cost	US$	619,171	1,011,251
Piping and Valves			
Piping and Valves Capital Cost	US$	4,721,257	4,789,243
Factor	%	2.5%	2.5%
Total Annual Cost	US$	118,031	119,731
Electricals			
Electrical Infrastructure Capital Cost	US$	4,245,587	4,872,580
Factor	%	2.5%	2.5%
Total Annual Cost	US$	106,140	121,815
Instrumentation			
Instrumentation Capital Cost	US$	1,443,352	1,724,396
Factor	%	1.0%	1.0%
Total Annual Cost	US$	14,434	17,244
Maintenance Parts and Supplies Cost	**US$/a**	**857,776**	**1,270,041**
Maintenance Parts and Supplies Cost	**US$/t feed**	**0.29**	**0.42**

21.3.6.8 Assay

The annual assay operational cost was obtained from a contractor quotation and was estimated at **US$512,434**.

The assay costs consist of the mining/grade control and process plant assay requirements. The process plant assay operating costs were determined by identifying the samples to be collected, the frequency of the sampling, and the type of analysis required. The number of metal accounting samples was based on a shift cycle of three 8-hour shifts per day. The metallurgical control samples would be collected as and when required. An allowance of the number per annum has been made to account for these.

The number of grade control samples was determined based on the mining plan, to identify how many samples and where in the pit they would be collected. These sample numbers and required tests were submitted to analytical laboratory contractors for quotations.

The assay costs for the plant and mining are summarised in Table 21.38.

Table 21.38: Assay OPEX Summary

Description	Cost US$/month	Cost US$/a
Labour	13,524	162,288
Skills Development and Employee Development Training	717	8,601
Establishment	458	5,495
Fixed Consumables	3,704	44,446
Replacement spares	1,398	16,771
Process consumables	16,771	201,250
Insurance and Finance Costs	354	4,245
Overheads	5,778	69,338
Total Cost	**42,703**	**512,434**

21.3.7 General and Administration (G&A) Costs

The G&A annual operating costs were estimated at **US$6,626,990**. The costs were determined from first principles and by using information from SENET's in-house database for similar projects from the same locality.

A summary of the LOM G&A operating costs is shown in Table 21.39.

Table 21.39: LOM G&A Cost Summary

Description	Unit	TOTAL	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16
Salaries and Wages	US$	29,921,968	989,520	1,979,040	1,979,040	1,979,040	1,979,040	1,979,040	1,979,040	1,979,040	1,979,040	1,979,040	1,979,040	1,979,040	1,979,040	1,979,040	1,979,040	1,225,892
Camp Food Costs	US$	20,715,187	685,052	1,370,103	1,370,103	1,370,103	1,370,103	1,370,103	1,370,103	1,370,103	1,370,103	1,370,103	1,370,103	1,370,103	1,370,103	1,370,103	1,370,103	848,693
Maintenance	US$	8,940,842	295,674	591,348	591,348	591,348	591,348	591,348	591,348	591,348	591,348	591,348	591,348	591,348	591,348	591,348	591,348	366,303
Off-Site Offices and Travel	US$	7,798,727	257,904	515,808	515,808	515,808	515,808	515,808	515,808	515,808	515,808	515,808	515,808	515,808	515,808	515,808	515,808	319,511
Supplies and Spare Parts	US$	6,243,300	206,466	412,932	412,932	412,932	412,932	412,932	412,932	412,932	412,932	412,932	412,932	412,932	412,932	412,932	412,932	255,786
Security	US$	6,350,164	210,000	420,000	420,000	420,000	420,000	420,000	420,000	420,000	420,000	420,000	420,000	420,000	420,000	420,000	420,000	260,164
Other Administration Costs	US$	15,784,693	522,000	1,044,000	1,044,000	1,044,000	1,044,000	1,044,000	1,044,000	1,044,000	1,044,000	1,044,000	1,044,000	1,044,000	1,044,000	1,044,000	1,044,000	646,693
Environmental and Social	US$	4,441,486	146,880	293,760	293,760	293,760	293,760	293,760	293,760	293,760	293,760	293,760	293,760	293,760	293,760	293,760	293,760	181,966
Total G&A Cost	**US$**	100,196,366	3,313,495	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	6,626,990	4,105,007

21.3.7.1 G&A Labour

The labour cost for G&A personnel was estimated using the same principles and labour rates as outlined in the process plant labour section. The G&A labour cost summary is shown in Table 21.40, and the detailed G&A labour cost summary is given in Table 21.41.

Table 21.40: G&A Labour Cost Summary

Description	Total Number of Employees	Total Cost	Total Cost
		US$/month	US$/a
General Management	7	24,350	292,200
Health, Safety and Environment	4	15,772	189,261
Finance	6	24,172	290,067
Community Relations	3	3,598	43,179
Vehicles Maintenance	7	9,896	118,757
IT and Communications	4	10,601	127,209
Security	5	22,523	270,280
Human Resources Management	4	14,875	178,495
Procurement and Logistics	9	14,989	179,872
Office and Domestic Services	1	1,094	13,125
Facilities Maintenance	11	23,050	276,595
Total	**61**	**164,920**	**1,979,040**

Table 21.41: G&A Detailed Labour Cost Breakdown

Position Description	Employee Category	Total Number of Employees	Rate/Employee	Total Cost	Total Cost
			US$/month	US$/month	US$/a
General Management					
General manager	E1	1	20,000	20,000	240,000
Secretary	E2	1	1,215	1,215	14,583
Receptionist	L	1	627	627	7,523
Driver	L	4	627	2,508	30,093
Health, Safety and Environment (HSE)					
HSE manager	E2	1	7,437	7,437	89,247
Safety officer	E2	2	2,344	4,689	56,265
Environmental scientist	E2	1	3,646	3,646	43,749
Finance					
Accounting finance manager	E1	1	15,000	15,000	180,000
Accountant	E2	2	3,646	7,291	87,497
Accounts clerk	L	3	627	1,881	22,570
Community Relations					
Community relations manager/officer	L	1	2,344	2,344	28,132
Community relations assistant	L	2	627	1,254	15,047
Vehicles Maintenance					
Vehicle maintenance manager	E2	1	4,958	4,958	59,498
Vehicle mechanic	L	2	1,215	2,430	29,166
Vehicle mechanic assistant	L	4	627	2,508	30,093
IT and Communications					
IT manager	E2	1	4,958	4,958	59,498
IT technician	E2	3	1,881	5,643	67,710

Position Description	Employee Category	Total Number of Employees	Rate/Employee	Total Cost	Total Cost
			US$/month	US$/month	US$/a
Security					
Security manager	E1	1	15,000	15,000	180,000
Security supervisor	L	4	1,881	7,523	90,280
Security personnel	L	0	627	0	0
Human Resources Management					
Human resources manager	E2	1	4,958	4,958	59,498
Human resources officer	L	1	4,958	4,958	59,498
Training officer	E2	2	2,479	4,958	59,498
Procurement and Logistics					
Procurement and logistics manager	E2	1	4,958	4,958	59,498
Buyer	L	2	1,881	3,762	45,140
Expeditor	L	2	1,881	3,762	45,140
Stores attendant	L	4	627	2,508	30,093
Office and Domestic Services					
Camp manager	E2	1	1,094	1,094	13,125
Facilities Maintenance					
Engineering manager	E1	1	15,000	15,000	180,000
Fitter	E2	1	929	929	11,145
Boilermaker	E2	1	929	929	11,145
Fitter operative	E2	2	627	1,254	15,047
Fitter assistant	L	2	627	1,254	15,047
Electrician	E2	2	1,215	2,430	29,166
Electrical operative	L	2	627	1,254	15,047
TOTAL		**61**			**1,979,040**

21.3.7.2 Camp Food

The camp food costs were derived from quotations from a local catering company. Expatriate labour will be on a seven-weeks-in and three-weeks-out roster. During the seven-week period that the expatriates will be on duty, they will be accommodated in the main camp where food will be provided at a cost of US$26.00 per person per day. Daily meals will be provided for local shift workers and day-shift labourers who do not reside in the camp. The daily meals will be provided at a cost of US$6 per person per day.

The estimated camp food and catering costs were based on the total number of people staying at the camp as well as local personnel for the estimated workdays per year as outlined in Table 21.42.

Table 21.42: Camp Food Costs per Person per Day

Description	Cost/Person/Day US$	Number of People	Days/Year
E1	18	22	280
E2	13	174	287
L	4,8	256	287
Total		**452**	

Using the estimated number of workdays per person, the expatriate shift roster, and the rate per person per day, the annual camp food cost was calculated. The annual camp catering staff cost was estimated at US$814,800/a. The camp food cost summary is shown in Table 21.43.

Table 21.43: Camp Food Cost Summary

Description	Total	G&A	Mining	Process Plant
Number of Expatriates (E1)	11	4	3	4
Number of Expatriates (E2)	87	23	10	54
Number of Locals (L)	128	34	26	68
E1 Cost per Year (US$/a)	56,210	20,440	15,330	20,440
E2 Cost per Year (US$/a)	324,597	85,813	37,310	201,474
L Cost per Year (US$/a)	174,496	46,351	35,445	92,701
Total G&A Food Cost per Year	**555,303**	**152,604**	**88,085**	**314,615**

21.3.7.3 G&A Maintenance

The G&A maintenance costs cover the following main items:

- On-site and off-site infrastructure
- Vehicle running costs

The annual maintenance costs for the main camp, plant infrastructure buildings, and in-plant roads were estimated by assuming a factor of 1 % of the respective estimated capital costs.

The off-site infrastructure maintenance cost was derived by applying a factor of 1 % to the total off-site infrastructure capital cost.

The estimated annual vehicle operating costs were determined by taking into account the projected servicing and fuel costs for each vehicle type. The costs are summarised in Table 21.44.

Table 21.44: Infrastructure and Vehicle OPEX Summary

Description	Cost	Cost
	US$/month	US$/a
Main Camp	8,722	104,662
Infrastructure Buildings	4,146	49,757
Project Roads	4,146	49,757
Off-Site Infrastructure	4,146	49,757
Vehicle Cost – Maintenance	9,860	118,322
Vehicle Cost – Fuel	18,258	219,092
Total	**49,279**	**591,348**

21.3.7.4 Off-Site Offices – Bamako Offices

An annual allowance of **US$180,000** has been made to cover the Bamako office costs for staff salaries and other office amenities for people providing logistics. The costs are summarised in Table 21.45.

Table 21.45: Bamako Offices Cost Summary

Description	Cost	Cost
	US$/month	US$/a
Logistics Manager	4,000	48,000
Secretary	1,200	14,400
Office Rental	4,000	48,000
Telephone	600	7,200
Electricity	400	4,800
Water	300	3,600
Rates and Taxes	600	7,200
Stationery	1,200	14,400
Insurance	400	4,800
Vehicle Costs	800	9,600
Fuel	500	6,000
Miscellaneous Expenses	1,000	12,000
Total	**10,011**	**180,000**

21.3.7.5 G&A Travel

The expatriate personnel travel costs were estimated based on a rotational work roster of seven weeks on and three weeks out. Based on this roster, each expatriate will be off site at least six times per year. The costs are summarised in Table 21.46.

Table 21.46: Travel OPEX Summary

Description	Total	G&A	Mining	Process Plant
FLIGHTS				
Number of Expatriates E1 (Business)	11	4	3	4
Weeks on Duty	40	40	40	40
Number of Flights per Expatriate	6	6	6	6
Cost per Flight (Return) (US$)		4,988	4,988	4,988
Total Flights Cost (US$/a)	**329,208**	**119,712**	**89,784**	**119,712**
Number of Expatriates E2 (Economy)	87	23	10	54
Weeks on Duty	40	40	10	50
Number of Flights per Expatriate	6	6	6	6
Cost per Flight (Return) (US$)		700	700	700
Total Flights Cost (US$/a)	**365,400**	**96,600**	**42,000**	**226,800**
ACCOMMODATION AND FOOD				
Number of Expatriates	11	4	3	4
Weeks on Duty	40	40	40	40
Number of Flights per Expatriate	6	6	6	6
Costs in Transit (US$)	2,100	700	700	700
Number of Days	2	2	2	2
Food Cost per Person per Day (US$)	50	50	50	50
Total Cost for Accommodation and Food (US$)	6,600	2,400	1,800	2,400
Total Cost for Expatriate Travel (US$)	**335,808**	**122,112**	**91,584**	**122,112**

21.3.7.6 Security

A security cost of **US$420 000** per annum was determined from a quotation received from a security company, Securicom Protect.

21.3.7.7 Supplies and Spare Parts

The supplies and spare parts costs were determined to be **US$394 932** per annum based on quotations and assumptions made from SENET's projects database where applicable.

The costs are summarised in Table 21.47.

Table 21.47: Supplies and Spare Parts Costs

Description	Cost US$/month	Cost US$/a
Administration and HR	1,000	12,000
Accounting	1,000	12,000
Safety Supplies	20,000	240,000
Warehouses	2,000	24,000
Medical Contract	3,411	40,932
Medical Doctor/Consultations	5,000	60,000
Security	2,000	6,000
Total	**34,411**	**394,932**

The medical cost was determined based on a contract solution that included personnel, emergency services, clinic maintenance, and medical supplies.

21.3.7.8 Environmental and Social

The environmental and social cost of **US$293,760** per annum was estimated based on quotations from the ESIA consultant, ABS Africa.

21.3.7.9 Other Administration

An administration cost of **US$1,044,000** per annum was allowed for based on SENET's projects database. The costs are summarised in Table 21.48.

Table 21.48: Other Administration Costs

Description	Cost US$/month	Cost US$/a
Communication	15,000	180,000
Couriers	1,000	12,000
Insurances	30,000	360,000
Licence Fees for Software	1,500	18,000
Computer Hardware Update	500	6,000
Consultant Fees	10,000	120,000
Legal	2,000	24,000
Accounting and Tax Audit	5,000	60,000
Training	12,000	144,000
Recruiting	10,000	120,000
Total	**87,000**	**1,044,000**

The communication cost includes maintenance and software updates for the communications equipment. The recruiting cost is based on a staff turnover of 20 % at a recruitment cost of 10 %.

22 ECONOMIC ANALYSIS

22.1 EVALUATION PRINCIPLES AND METHOD

The financial results presented in this Report have been derived with the application of valuation methodologies that are aligned with AGG's standards and other international standards on project evaluation. No material, non-compliant issues are known to exist in the valuation of the Project.

All financial numbers are presented in constant money terms (CMT), using the US dollar (US$) over the LOM, unless otherwise stated.

The primary project evaluation method is a discounted unlevered free cash flow (DUFCF) analysis, which determines the net present value (NPV) of all expected cash inflows and cash outflows over Kobada's LOM of 16 years from start of production. Cash inflows consist of annual revenue projections while cash outflows consist of initial capital costs, sustaining capital costs, operating costs, taxes and royalties.

The DUFCF method takes into account the time value of money. It consists of estimating projected future cash flows generated by the assets, subtracting the cost of the investment, and discounting these cash flows at the company's required rate of return. The result, if positive, indicates that the investment results in an increase in shareholder wealth, as it earns more than the required rate of return. The project investment has been evaluated on a stand-alone basis, and no incremental cash flow was calculated and was done assuming 100% ownership.

22.2 ASSUMPTIONS

The economic analysis is based on several technical and economic input assumptions, as presented in Table 22.1.

Table 22.1: Financial Analysis Assumptions

Description	Unit	Assumption
Evaluation Start Date		01 January 2022
Revenue		
Gold Price	US$/oz	1,750
Refining Losses	%	0.08
Discount Rate	%	5.0
Fuel Prices		
Diesel Price	US$/L	0.797
HFO Price	US$/L	0.428
Fiscal		
Government Royalty	%	3
Tax Holiday	Years	3
Tax Rate (after tax holiday)	% of profits	30
Tax Rate (if there is a loss)	% of annual turnover	1

Description	Unit	Assumption
Dividend Tax	%	10
Depreciation	%	10 % over 10 years
Units of Measurement		
Kilograms to Ounces	kg/troy oz	32.1505
Diesel SG	t/m^3	0.85
HFO SG	t/m^3	0.97
Other Charges		
Bullion Transport and Refining Costs	US$/oz	3.70
Exchange Rates	ZAR/US$	14.50
	£/US$	0.93
	A$/US$	1.57
	C$/US$	0.81
	CFA/€	655
	CFA/US$	561

22.3 EVALUATION DATE

The NPV results presented were based on an evaluation and start date of 1 January 2022.

22.4 GOLD PRICE

For the purposes of the economic analysis, the assumed gold price for the LOM is US$1,750/oz in 2022 CMT. The gold price used is based on the review of median consensus price forecasts by large banks and brokerages, recently published technical reports, and 3-month and 12-month trailing averages. A sensitivity analysis was performed to address the impact of various financial and operating variables on the overall project economic results.

22.5 DISCOUNT RATE

For the economic analysis, a real discount rate of 5 % was applied. A discount rate of 5 % is commonly used to evaluate gold projects of this nature. Where specific debt funding will be used together with equity funding, the appropriate weighted average cost of capital (WACC) should be determined and used in the calculation of the NPV. In the absence of such information, the Project was evaluated using the real discount rate as noted above. The sensitivity of the Project NPV to a range of discount rates is provided.

22.6 ESCALATION AND INFLATION

There is no adjustment for inflation and escalation in the financial model; all cash flows are in US dollars CMT.

22.7 TAXATION AND ROYALTIES

22.7.1 Taxation

The following outlines the main taxation considerations applied in the financial model as provided by AGG with reference to Law No. 2012-015 of 27 February 2012 (2012 Mining Code).

- In terms of the Establishment Convention relating to the Kobada Mining Title, dated 6 November 2000, it is governed by the provisions of the 1999 Mining Code and is subject to the Mali corporate tax rate of 30 %.
- A tax holiday of three years from the date of production has been applied (Article 110 of the Code and Article 15.8 of the Establishment Convention of 6 November 2000 signed with Cominor on the Kobada perimeter).
- Tax allowances for initial capital costs: A capital allowance of 10 % per annum on a straight-line basis has been applied to the mining and processing initial capital costs.
- Tax treatment for stay-in-business (SIB) capital: These costs are claimed in the year that they are incurred.
- The Project was evaluated on a stand-alone basis with tax ringfencing applied.
- Dividend tax on the cash flows to AGG was not considered in this evaluation as it was assumed that the valuation is based on the Project's value to AGG. If this tax were to be considered, then the cash flows that AGG receives would decrease by 10 %.

22.7.2 Government Royalties

The main royalty considerations applied in the financial model, as provided by AGG with reference to the 1999 Mining Code, are royalties known as a "Special Tax on Certain Products" (ISCP), which is payable at a fixed rate of 3 % of the gross revenue.

22.7.3 Depreciation

Depreciation is ignored in the calculation of the NPV. It is, however, accounted for in the earnings before interest and tax (EBIT) whereby the initial capital is depreciated at 10 % per annum on a straight-line basis.

22.7.4 Working Capital

The calculation of the NPV considered the following working capital needs:

- Payables: This considered the necessity to make routine payments, cover unexpected costs, and purchase materials used in mining and processing. A conservative view was taken regarding payables with all the expenses incurred settled in the month they were incurred. The financial model accounted for the start-up working capital needs based on expected commitments.
- Receivables: The refining process will be completed within two to three working days after receipt of the bullion. Generally, 99.92 % of the gold contained in the bullion will be returned to AGG, and the sale price will be fixed on the day of the refinery outrun with a settlement of two to three working days. It is estimated that it will take approximately five days from receipt of the bullion before payment is received. Kobada

plans to despatch bullion once every week; thus, in all practical terms, it is estimated that payment from gold sales will be received in the same month as production.

22.7.5 Production Schedule and Operating Cash Costs

The production schedule and OPEX assumptions used in the model are given in detail in Section 14 of this Report and summarised in Table 22.2.

Table 22.2: Summary of Production Schedule and Operating Cash Costs

Description	Unit	LOM Totals	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
Tonnes Treated	kt	45,028	1,377	2,997	3,007	2,906	2,930	2,964	2,961	3,030	2,996	3,059	2,957	2,998	2,918	3,031	3,041	1,858
Grade	g/t	0.87	1.18	1.10	1.14	1.10	1.09	1.12	1.12	0.92	1.16	1.05	0.66	0.46	0.45	0.46	0.46	0.44
Recovery	%	95.6	96.6	96.5	96.6	96.5	96.5	96.5	96.5	96.3	95.1	94.5	94.5	94.7	94.6	93.9	92.1	91.6
Gold Production	koz	1,202	50.35	102.46	106.71	99.02	99.17	102.99	103.29	86.22	105.84	97.35	58.88	42.33	40.17	41.74	41.52	24.24
Payable Gold	%	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9
Payable Gold	koz	1,201.3	50.31	102.37	106.62	98.94	99.09	102.91	103.20	86.15	105.75	97.28	58.83	42.30	40.14	41.70	41.49	24.22
Gold Price	US$/oz	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750
Revenue	US$ thousand	2,102,300	88,039	179,154	186,588	173,148	173,413	180,090	180,606	150,761	185,065	170,232	102,956	74,021	70,247	72,981	72,606	42,391
Operating Costs																		
Mining	US$ thousand	506,451	23,387	52,350	56,463	58,311	56,531	61,965	65,534	65,952	29,325	14,197	16,239	1,470	1,325	1,377	1,229	796
Plant	US$ thousand	385,196	11,080	23,049	23,119	22,776	22,976	23,178	23,028	23,616	30,360	32,606	26,190	23,398	22,888	26,128	31,460	19,343
G& A	US$ thousand	100,196	3,313	6,627	6,627	6,627	6,627	6,627	6,627	6,627	6,627	6,627	6,627	6,627	6,627	6,627	6,627	4,105
Refining Charges	US$ thousand	4,448	186	379	395	366	367	381	382	319	392	360	218	157	149	154	154	90
Government Royalties	US$ thousand	63,069	2,641	5,375	5,598	5,194	5,202	5,403	5,418	4,523	5,552	5,107	3,089	2,221	2,107	2,189	2,178	1,272
Total Cash Costs	US$ thousand	1,059,361	40,608	87,780	92,201	93,275	91,703	97,555	100,989	101,037	72,256	58,896	52,363	33,872	33,097	36,476	41,648	25,606
Total Cash Costs	US$/oz	881	807	857	864	942	925	947	978	1,172	683	605	889	800	824	874	1,003	1,056

Production commences in August 2023 with steady state (250,000 t processed per month) achieved within two months. The average grade achieved during the first 10 years of operation is ± 1.1 g/t with the grade decreasing in the last six years of operation to an average grade of 0.5 g/t as lower-grade stocks are pulled from stockpiles.

An average recovery rate of ± 96 % is expected for the processing of high-grade ore during 2023-2032. A recovery rate of ± 94 % is expected during the last six years of operation.

The first tonnes will be processed in August 2023, and the mining operations will deliver the last tonnes in 2031. During the mining period (2023-2033), the delivered tonnes will exceed the processing capacity, resulting in a constant ore stock build-up to 2033.

The mining cost over the LOM is US$506 million (48 % of the mine operating cost (MOC)), resulting in an average of US$11.25/t processed over the LOM.

The processing cost over the LOM is US$385 million (36 % of the MOC), resulting in an average of US$8.55/t processed over the LOM. G&A costs over the LOM are US$100 million (9 % of the MOC) and are expected to remain constant over the LOM, resulting in an average of US$2.22/t processed over the LOM. The refining cost over the LOM is US$4.44 million (0.4 % of the MOC) resulting in US$3.7/oz delivered over the LOM.

The average LOM operating cost is US$881/oz and is in line with other operations treating saprolite followed by fresh/sulphide ores in West Africa. The distribution of the cash cost over the LOM is shown in Figure 22.1.

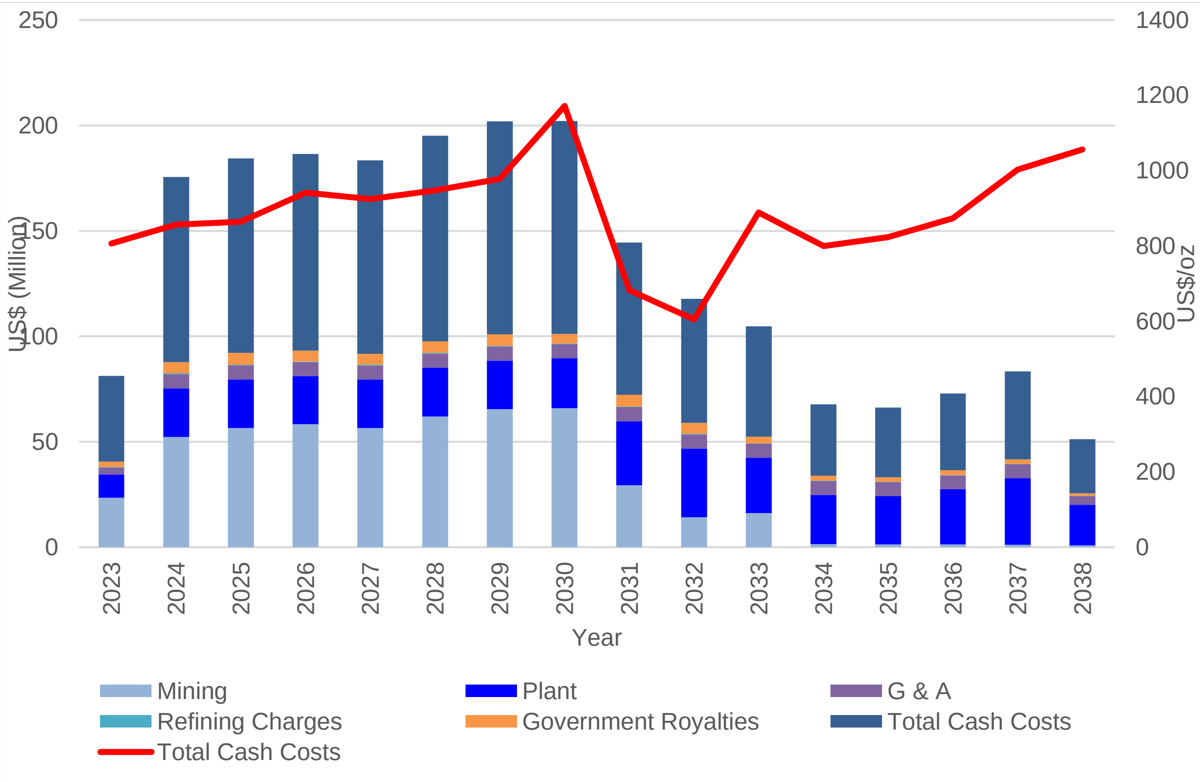


Figure 22.1: Production Schedule and Operating Cash Costs

22.8 INITIAL CAPITAL AND SUSTAINING COSTS

The initial capital required to complete the Project is US$165.95 million. The sustaining capital over the life of the Project is estimated to be US$108.67million, which includes rehabilitation and closure costs of US$31.07 million. This includes a contingency of US$10.7 million.

A summary of the capital costs is given in Table 22.3, and the capital cost schedule is given in Table 22.4.

Table 22.3: Summary of Capital Costs

Item	Value (US$ Million)
INITIAL CAPITAL	**165.95**
Mining	**28.25**
Mining – Development Costs	23.57
Mining – Other Costs	3.87
Haul Roads	0.80
Process Plant and TSF	**137.69**
Process Plant	88.67
TSF Phase 1A	29.43
IFC PS Action Plans and Readiness Assessment	0.38
Owner's Pre-Production	8.26
Resettlement	6.17
Working Capital	4.78
SUSTAINING CAPITAL	**108.67**
Mining	**7.00**
Mining – Pit Dewatering and Treatment	1.90
Mining – Contractor Mobilisation/Demobilisation	5.10
Process Plant and TSF	**89.29**
Process Plant	28.46
TSF Phase 1B to Phase 5	60.83
Mine Wide and G&A	**12.38**
Mine Wide – Resettlement	0.00
Mine Wide – Rehabilitation – Operational	6.64
Mine Wide – Decommissioning and Closure	18.80
Mine Wide – Post-Closure Costs	5.64
Process Plant Salvage Value	−18.70

Table 22.4: Capital Cost Schedule

Description	Unit	LOM	2022/12/31	2023/07/31	2023/12/31	2024/12/31	2025/12/31	2026/12/31	2027/12/31	2028/12/31	2029/12/31	2030/12/31	2031/12/31	2032/12/31	2033/12/31	2034/12/31	2035/12/31	2036/12/31	2037/12/31	2038/12/31
INITIAL CAPITAL																				
Mining																				
Mining – Development Costs	US$ thousand	23,575	0	23,575																
Mining – Other Costs	US$ thousand	3,872	0	3,872																
Haul Roads	US$ thousand	804	0	804																
Process Plant																				
Plant and Infrastructure	US$ thousand	88,671	50,627	38,044																
TSF	US$ thousand	29,425	11,034	18,391																
IFC PS Action Plans and Readiness Assessment	US$ thousand	384	384	0																
Owner's Pre-Production	US$ thousand	8,264	4,628	3,636																
Resettlement	US$ thousand	6,167	4,111	2,056																
Working Capital	US$ thousand	4,783	0	4,783																
SUSTAINING CAPITAL																				
Mining – Pit Dewatering Pumps and Piping	US$ thousand	1,089			0	0	545	0	545	0	0	0	0	0	0	0	0	1,089		
Mining – Pit Dewatering Water Treatment Plant	US$ thousand	813			0	0	406	0	406	0	0	0	0	0	0	0	0	813		
Mining – Contractor Mobilisation/ Demobilisation	US$ thousand	5,100			1,020	1,020	0	0	0	0	0	0	0	0	3,060	0	0	5,100		
Process Plant	US$ thousand	28,459			0	0	0	0	0	0	11,384	17,075	0	0	0	0	0	28,459		
TSF Phase 1B to Phase 5	US$ thousand	60,833			7,856	150	15,236	0	150	16,966	0	150	11,942	0	150	8,233	0	60,833		
Mine Wide – Resettlement	US$ thousand	0			0	0	0	0	0	0	0	0	0	0	0	0	0	0		
Mine Wide – Rehabilitation – Operational	US$ thousand	6,640			347	0	263	473	394	294	604	294	427	609	299	399	555	6,640		
Mine Wide – Decommissioning and Closure	US$ thousand	18,795			0	0	0	0	0	0	0	0	0	0	0	0	0	18,795		
Mine Wide – Post-Closure Costs	US$ thousand	5,642			0	0	0	0	0	0	0	0	0	0	0	0	0	5,642		
Equipment Salvage Value	US$ thousand	(18,698)			0	0	0	0	0	0	0	0	0	0	0	0	0	(18,698)		
TOTAL PROJECT CAPEX	**US$ thousand**	**274,619**	**70,785**	**95,160**	**9,223**	**1,170**	**16,450**	**473**	**1,495**	**17,260**	**11,988**	**17,520**	**12,369**	**609**	**3,509**	**8,633**	**555**	**274,619**	**70,785**	**95,160**

The initial capital (US$166 million) will be required for the processing facilities, infrastructure, and tailings dam construction, which will commence on 1 January 2022 and are expected to be completed on 30 September 2023 (21 months). The mining capital (US$28 million) will be required for the pre-stripping operations, which will commence in February 2023 and be completed in July 2023 (6 months). The sustaining capital consists of mining capital (US$7 million), processing capital (US$28 million), TSF capital (US$61 million), and G&A capital (net US$12 million). It is important to note that the SIB G&A cost for Year 2038 includes closure costs of US$18.8 million with salvage revenue offsets of US$18.7 million, a net cost impact of US$12 million. It is envisaged that mining operations will be extended and would, therefore, defer closure and salvage operations.

The LOM all-in sustaining costs (AISC) are US$972/oz. The breakdown of the AISC is given in Table 22.5, and a summary of the costs per ounce is shown in Figure 22.2.

Table 22.5: AISC Breakdown per Year

Description	Unit	LOM Totals	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
Operating Costs																		
Mining	US$ thousand	506,451	23,387	52,350	56,463	58,311	56,531	61,965	65,534	65,952	29,325	14,197	16,239	1,470	1,325	1,377	1,229	796
Plant	US$ thousand	385,196	11,080	23,049	23,119	22,776	22,976	23,178	23,028	23,616	30,360	32,606	26,190	23,398	22,888	26,128	31,460	19,343
G&A	US$ thousand	100,196	3,313	6,627	6,627	6,627	6,627	6,627	6,627	6,627	6,627	6,627	6,627	6,627	6,627	6,627	6,627	4,105
Refining Charges	US$ thousand	4,448	186	379	395	366	367	381	382	319	392	360	218	157	149	154	154	90
Government Royalties	US$ thousand	63,069	2,641	5,375	5,598	5,194	5,202	5,403	5,418	4,523	5,552	5,107	3,089	2,221	2,107	2,189	2,178	1,272
Total Cash Costs	US$ thousand	1,059,361	40,608	87,780	92,201	93,275	91,703	97,555	100,989	101,037	72,256	58,896	52,363	33,872	33,097	36,476	41,648	25,606
SIB Costs																		
Mining – Pit Dewatering and Treatment	US$ thousand	1,089	0	0	545	0	545	0	0	0	0	0	0	0	0	0	0	0
Mining – Contractor Mobilisation/Demobilisation	US$ thousand	5,100	1,020	1,020	0	0	0	0	0	0	0	0	3,060	0	0	0	0	0
Process Plant	US$ thousand	28,459	0	0	0	0	0	0	11,384	17,075	0	0	0	0	0	0	0	0
TSF Phase 1B to Phase 5	US$ thousand	60,833	7,856	150	15,236	0	150	16,966	0	150	11,942	0	150	8,233	0	0	0	0
Mine Wide – Resettlement	US$ thousand	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Mine Wide – Rehabilitation – Operational	US$ thousand	6,640	347	0	263	473	394	294	604	294	427	609	299	399	555	408	1,272	0
Mine Wide – Decommissioning and Closure	US$ thousand	18,795	0	0	0	0	0	0	0	0	0	0	0	0	0	0	8,676	10,120
Mine Wide – Post-Closure Costs	US$ thousand	5,642	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	5,642
Equipment Salvage Value	US$ thousand	(18,698)	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	(18,698)
Total SIB Costs	US$ thousand	108,673	9,223	1,170	16,450	473	1,495	17,260	11,988	17,520	12,369	609	3,509	8,633	555	408	9,948	-2,937
Total AISC Costs	US$ thousand	1,168,034	49,831	88,950	108,651	93,748	93,199	114,815	112,977	118,557	84,625	59,506	55,872	42,505	33,652	36,884	51,596	22,669
Total AISC Costs	US$/oz	972	990	868	1,018	947	940	1,115	1,094	1,375	800	611	949	1,004	838	884	1,243	935

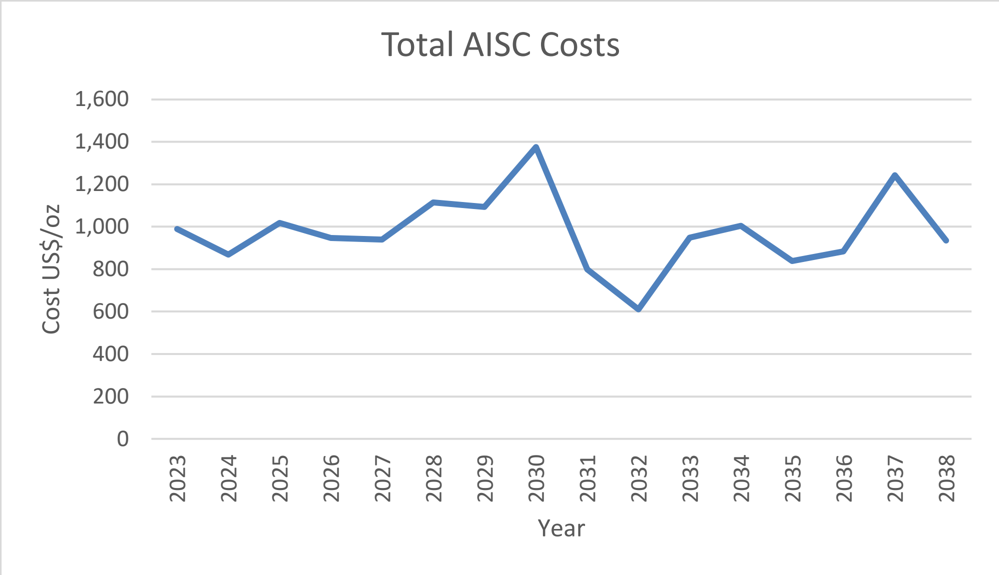


Figure 22.2: LOM AISC Breakdown per Year

Table 22.6 to Table 22.9 provides the details of the variance between the AISC value from the 2020 DFS (oxide ore only) to the 2021 DFS (includes sulphide treatment). The inclusion of sulphide ore meant an increased resource/ reserve and therefore more ore to be mined and treated.

Table 22.6: Mining OPEX Increase

Item Description	2020	2021	Variance	Variance US$/oz
Diesel	37,678,135	110,727,473	73,049,338	30
Waste & Ore Mining	127,885,468	266,497,604	138,612,136	57
Ore Rehandling	6,652,971	11,871,252	5,218,281	2
Drill & Blast	5,632,393	36,166,956	30,534,563	13
Grade Control & Overheads	43,315,404	81,187,934	37,872,530	16
Total	221,164,372	506,451,220	285,286,848	
Total- US$/oz	**304**	**421**	**118**	**118**

Part of the increased AISC cost is due to increased mining OPEX caused largely by increased fuel costs and higher tonnages of waste and ore mining when the sulphide ore is mined, resulting in cost variances of US$30/oz and US$57/oz respectively.

Table 22.7: Plant OPEX Increase – Sulphides Treatment

Item Description	Unit	Amount
Sulphides Tonnage	t	11,362,145
Oxides Power	kWh/t	14.56
Sulphides Power	kWh/t	29.68
Additional power Requirements	kWh/t	15.12
Additional power Costs	US$	35,323,694
Additional power Costs	**US$/oz**	**29.38**

The introduction of sulphide ore, results in high power consumption in the process plant, 15.12 kWh/t more than when treating oxide ore. This increase in power contributes an additional US$29.38/oz to the increased AISC cost.

Table 22.8: Power Cost Increase

Item Description	Unit	Amount
LOM Power	kWh	829,517,134
2020 Power Costs	US$/kWh	0.157
2021 Power Costs	US$/kWh	0.2056
Increase in Power Costs	US$/kWh	0.0486
Increase in Power Costs	US$	40,314,533
Increase in Power Costs	**US$/oz**	**33.53**

Due to the commodity prices increases from 2020 DFS to 2021 DFS, the cost of power has also been adversely affected, since it heavily depends on the commodity prices, the most critical being the fuel price. The fuel and other material prices increase resulted in a power unit price increase of US$0.0486/kWh increase, translates to additional AISC cost of US$33.53/oz.

Table 22.9: Sustaining and Rehabilitation Costs Increase

Item Description	2020	2021	Variance
Tailing Dams	31,773,642	60,833,324	29,059,682
Process Plant- Sulphides		28,458,836	28,458,836
Rehab & Closure (nett)	18,328,240	12,379,106	-5,949,134
Total	50,101,882	101,671,266	51,569,384
Variance US$/oz			**43**

The addition of sulphides to the resource meant additional plant equipment required to treat the sulphide as well bigger tailings dam footprint, contributing to increased AISC.

22.9 RESULTS OF FINANCIAL ANALYSIS

Using the assumptions shown in Table 22.1 and the production schedules, capital and operating costs given in detail in Sections 13 and 14, a financial analysis was carried out. Sunk costs were excluded from this analysis.

22.9.1 Summary of Financial Results

Table 22.10 summarises the key financial outputs from the financial model. Based on the DUFCF method, the decision is to accept an investment or project with a positive NPV. The Project generates an after-tax NPV of US$355 million at a 5 % real discount rate, an IRR of 37.6 % and a payback period of 2.33 years from commencement of production. The Project also generates an undiscounted free after-tax cash flow of US$549.9 million over the LOM.

The financial analysis results are summarised in Table 22.10.

Table 22.10: Financial Analysis Summary

Description	Unit	Pre-Tax	After Tax
LOM Tonnage Ore Processed	kt	45,028	45,028
LOM Feed Grade Processed	g/t	0.87	0.87
Production Period	Years	15.6	15.6
LOM Gold Recovery	%	95.6	95.6
LOM Gold Production	koz	1,202	1,202
LOM Payable Gold After Refining Losses	koz	1,201.3	1,201.3
Gold Price	US$/oz	1,750	1,750
Revenue	US$ million	2,102	2,102
LOM Operating Costs	US$/oz	881	881
AISC	US$/oz	972	972
NPV	US$ million	506	355
IRR	%	44.8	37.6
Discount Rate	%	5.0	5.0
Payback Period (from start of construction)	Years	3.91	3.91
Payback Period (from start of production)	Years	2.33	2.33
Project Net Cash	US$ million	773.1	549.9

22.9.2 Project LOM Cash Flow

The annual project cash flows are presented in Table 22.11 and Table 22.2

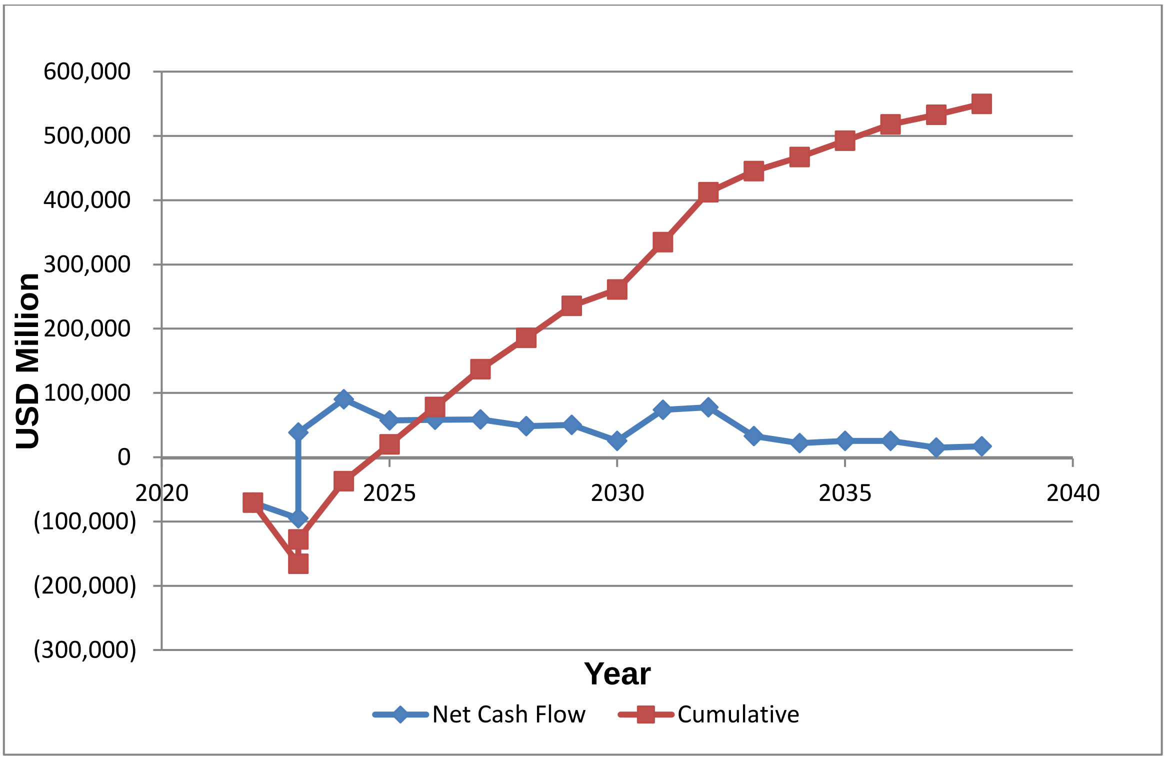


Figure 22.3: Project Discounted Cash Flow After Tax

The initial capital of US$71 million (2022) and US$95 million (2023) will result in a cash call of US$166 million over the first two years. The first tonnes will be processed in August 2023 and reach steady state in September 2023, resulting in a strong positive cash flow already in 2023. The annual net cash flow remains strong for the LOM and only declines during implementation of the sulphide phase and the last three years of production (owing to lower grade). The mining cost averages US$59 million per annum from the first year of ore extraction to 2031, after which it decreases until mining is stopped. The processing costs average US$24 million per annum, with a few spikes in the later production years, owing to high sulphide ore proportions. The cash flow is impacted by the net mine closure cost of US$5.6 million in 2038. Income taxation is absent for 2023 to 2025 due to a three-year tax holiday available for the Project.

Table 22.11: LOM Project Cash Flow

Description		LOM Totals	2022-12-31	2023-07-31	2023-12-31	2024-12-31	2025-12-31	2026-12-31	2027-12-31	2028-12-31	2029-12-31	2030-12-31	2031-12-31	2032-12-31	2033-12-31	2034-12-31	2035-12-31	2036-12-31	2037-12-31	2038-12-31
Tonnes Treated	kt	45,028			1,377	2,997	3,007	2,906	2,930	2,964	2,961	3,030	2,996	3,059	2,957	2,998	2,918	3,031	3,041	1,858
Grade	g/t	0.87			1.18	1.10	1.14	1.10	1.09	1.12	1.12	0.92	1.16	1.05	0.66	0.46	0.45	0.46	0.46	0.44
Recovery	%	95.6			96.6	96.5	96.6	96.5	96.5	96.5	96.5	96.3	95.1	94.5	94.5	94.7	94.6	93.9	92.1	91.6
Gold Production	koz	1,202			50.35	102.46	106.71	99.02	99.17	102.99	103.29	86.22	105.84	97.35	58.88	42.33	40.17	41.74	41.52	24.24
Payable Gold	%	99.9			99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9	99.9
Payable Gold	koz	1,201.3			50.31	102.37	106.62	98.94	99.09	102.91	103.20	86.15	105.75	97.28	58.83	42.30	40.14	41.70	41.49	24.22
Gold Price	US$/oz	1,750			1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750
Revenue	US$ thousand	2,102,300			88,039	179,154	186,588	173,148	173,413	180,090	180,606	150,761	185,065	170,232	102,956	74,021	70,247	72,981	72,606	42,391
Operating Costs																				
Mining	US$ thousand	506,451			23,387	52,350	56,463	58,311	56,531	61,965	65,534	65,952	29,325	14,197	16,239	1,470	1,325	1,377	1,229	796
Plant	US$ thousand	385,196			11,080	23,049	23,119	22,776	22,976	23,178	23,028	23,616	30,360	32,606	26,190	23,398	22,888	26,128	31,460	19,343
G&A	US$ thousand	100,196			3,313	6,627	6,627	6,627	6,627	6,627	6,627	6,627	6,627	6,627	6,627	6,627	6,627	6,627	6,627	4,105
Refining Charges	US$ thousand	4,448			186	379	395	366	367	381	382	319	392	360	218	157	149	154	154	90
Operating Costs Excluding Royalties	US$ thousand	996,292			37,967	82,405	86,603	88,080	86,501	92,152	95,571	96,514	66,704	53,789	49,274	31,651	30,989	34,286	39,470	24,334
Government Royalties	US$ thousand	63,069			2,641	5,375	5,598	5,194	5,202	5,403	5,418	4,523	5,552	5,107	3,089	2,221	2,107	2,189	2,178	1,272
Total Cash Costs	US$ thousand	1,059,361			40,608	87,780	92,201	93,275	91,703	97,555	100,989	101,037	72,256	58,896	52,363	33,872	33,097	36,476	41,648	25,606
Total Cash Costs	US$/oz	881			807	857	864	942	925	947	978	1,172	683	605	889	800	824	874	1,003	1,056
After-Tax Cash Flow																				
Cash Flow Before CAPEX	US$ thousand	1,042,939			47,432	91,374	94,387	79,874	81,710	82,535	79,617	49,724	112,810	111,335	50,593	40,149	37,150	36,506	30,958	16,785
Initial Capital																				
Mining	US$ thousand	28,251	0	28,251	0															
Process Plant	US$ thousand	88,671	50,627	38,044	0															
TSF	US$ thousand	29,425	11,034	18,391																
Pre-Production	US$ thousand	14,815	9,123	5,692																
Working Capital	US$ thousand	4,783	0	4,783																
Initial Capital Expenditure	US$ thousand	165,945	70,785	95,160	0															
Sustaining Capital																				
Mining SIB	US$ thousand	7,002			1,020	1,020	951	0	951	0	0	0	0	0	3,060	0	0	0	0	0
Processing SIB	US$ thousand	28,459			0	0	0	0	0	0	11,384	17,075	0	0	0	0	0	0	0	0
TSF SIB	US$ thousand	60,833			7,856	150	15,236	0	150	16,966	0	150	11,942	0	150	8,233	0	0	0	0
G&A SIB	US$ thousand	12,379			347	0	263	473	394	294	604	294	427	609	299	399	555	408	9,948	(2,937)
Sustaining Capital	US$ thousand	108,673			9,223	1,170	16,450	473	1,495	17,260	11,988	17,520	12,369	609	3,509	8,633	555	408	9,948	-2,937
Cash Flow After CAPEX	US$ thousand	768,320	(70,785)	(95,160)	38,209	90,204	77,937	79,401	80,215	65,275	67,629	32,205	100,441	110,726	47,084	31,516	36,595	36,098	21,011	19,722
Depreciation	US$ thousand	(84,565)			(9,038)	(9,038)	(9,140)	(9,242)	(9,337)	(9,337)	(9,432)	(9,432)	(10,570)	0	0	0	0	0	0	0
Taxable Income	US$ thousand	719,642			0	0	0	68,797	70,159	70,878	55,938	58,197	22,773	89,870	110,726	47,084	31,516	36,595	36,098	21,011
Tax Rate	%	30			0	0	0	30	30	30	30	30	30	30	30	30	30	30	30	30
Tax	US$ thousand	215,893			0	0	0	20,639	21,048	21,263	16,781	17,459	6,832	26,961	33,218	14,125	9,455	10,978	10,829	6,303
After-Tax Cash Flow	US$ thousand	552,428	(70,785)	(95,160)	38,209	90,204	77,937	58,762	59,167	44,012	50,847	14,746	93,609	83,765	13,866	17,391	27,140	25,119	10,182	13,418
Undiscounted Cumulative Cash Flow After Tax	US$ thousand		(70,785)	(165,945)	(127,736)	(37,532)	40,405	99,166	158,333	202,345	253,192	267,938	361,546	445,312	459,178	476,569	503,709	528,828	539,010	552,428
Discount Rate	%	5																		

Description		LOM Totals	2022-12-31	2023-07-31	2023-12-31	2024-12-31	2025-12-31	2026-12-31	2027-12-31	2028-12-31	2029-12-31	2030-12-31	2031-12-31	2032-12-31	2033-12-31	2034-12-31	2035-12-31	2036-12-31	2037-12-31	2038-12-31
Discounted Cash Flow After Tax	US$ thousand	338,499	(67,414)	(88,086)	34,657	77,922	47,139	45,721	43,990	34,463	33,957	16,356	45,110	45,318	18,353	11,700	12,938	12,155	6,738	7,484
Cumulative Discounted Cash Flow After Tax	US$ thousand		(67,414)	(155,500)	(120,844)	(42,922)	4,217	49,938	93,929	128,392	162,349	178,705	223,815	269,132	287,485	299,184	312,122	324,277	331,015	338,499
All-in Sustaining Costs (AISC)																				
AISC	US$ thousand	1,168,034			49,831	88,950	108,651	93,748	93,199	114,815	112,977	118,557	84,625	59,506	55,872	42,505	33,652	36,884	51,596	22,669
AISC	US$/oz	972			990	868	1,018	947	940	1,115	1,094	1,375	800	611	949	1,004	838	884	1,243	935
Discount Rate	%	5																		
Post-Tax NPV	(US$ thousand)	355,294																		
Post-Tax IRR	(%)	37.6																		
Undiscounted Payback Period from Start of Production	Years	2.33																		

22.9.3 Maximum Funding Requirement

The maximum funding requirement is US$165.9 million until August 2023, 21 months from the start of capital expenditure as shown in Figure 22.4.

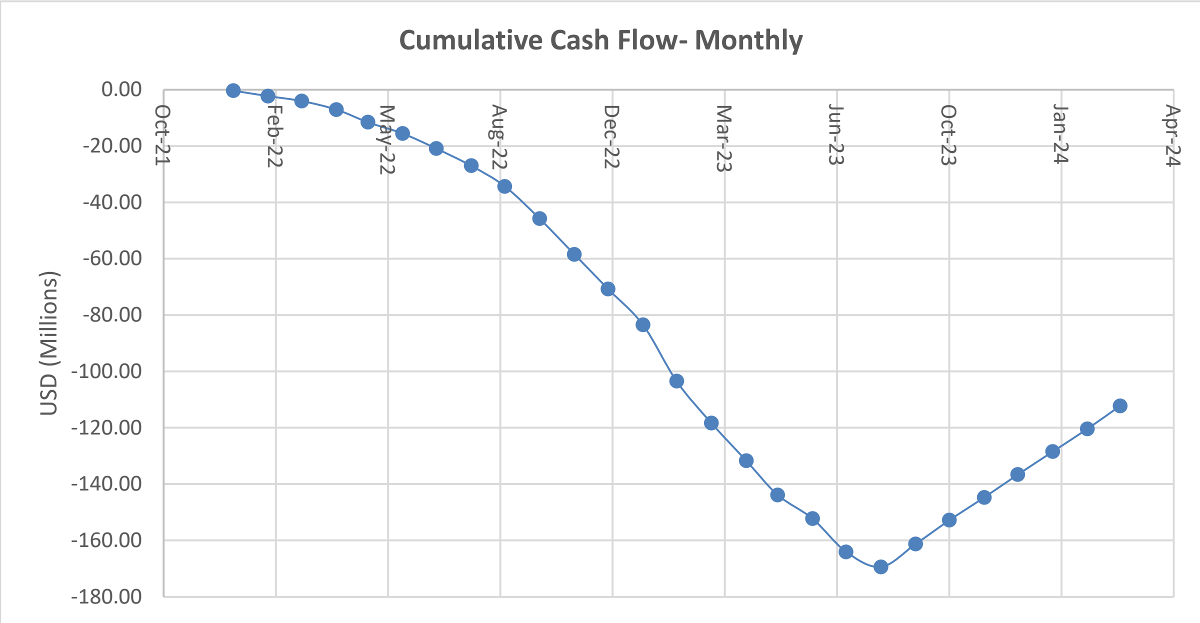


Figure 22.4: Maximum Funding Requirement

22.9.4 Sensitivities

A sensitivity analysis was performed on each individual variable, measuring the impact of each input variable independently on the target forecast (NPV).

An overall sensitivity analysis was also performed, considering all the variables on the target forecast.

22.9.5 Key Project Metric Sensitivity to Gold Price

Table 22.12 assesses the sensitivity of some of the key financial indicators of the Project to variations on the average gold price. The results reflect a robust project even with a low gold price, with more than 20 % IRR achieved. At the time of the study, the gold price was fluctuating at approximately US$1,780/oz. A strong gold price, which is anticipated in the short to medium period (three to five years), will significantly improve the NPV and IRR.

Table 22.12: Key Project Metric Sensitivity to Gold Price

Description	Unit	Change in Gold Price (%)				
		−15	−10	0	+10	+15
		Average Gold Price (US$/oz)				
		1,488	1,575	1,750	1,925	2,013
NPV at 5 % – After Tax	US$ million	191	246	**355**	465	520
IRR	%	23.2	28.1	**37.6**	47.0	51.7
Cash Flow Payback Period	Years	3.87	3.22	**2.33**	1.76	1.55
Maximum Funding	US$ million	170.29	169.99	**169.37**	168.76	168.46
NOTE: The values in bold indicate the base case NPV.						

22.9.6 Gold Price and Discount Rate Sensitivity Analysis

In assessing the combined impact of the gold price and the discount rate on the value of the Project, the two variables were varied concurrently, and the results of their combined sensitivity analysis are shown in Table 22.13. Considering the spot, three-year average and long-term median broker forecast gold prices, the economics of the Project are robust.

Table 22.13: Gold price and Discount Rate Sensitivity Analysis

Gold Price	NPV		
US$/oz	0 % Discount Rate	5 % Discount Rate	10 % Discount Rate
1,488	324	191	109
1,575	399	246	150
1,750	550	**355**	234
1,925	700	465	317
2,013	776	520	359
NOTE: The value in bold indicates the base case NPV.			

22.9.6.1 Gold Price and Head Grade Sensitivity Analysis

In assessing the combined impact of the gold price and the head grade on the value of the Project, these two variables were varied concurrently, and the results of their combined sensitivity analysis are shown in Table 22.14. The results show that the Project can accommodate a low head grade and low metal price.

Table 22.14: Gold Feed Grade and Gold Price Sensitivity

Head Grade	NPV at Varying Gold Prices (US$/oz)				
g/t	1,488	1,575	1,750	1,925	2,013
0.738	46	93	189	281	328
0.782	95	145	244	343	392
0.868	191	246	**355**	465	520

Head Grade	NPV at Varying Gold Prices (US$/oz)				
g/t	1,488	1,575	1,750	1,925	2,013
0.955	286	346	466	587	648
0.999	333	396	522	648	712
NOTE: The value in bold indicates the base case NPV.					

22.9.6.2 Gold Price and OPEX Sensitivity Analysis

In assessing the combined impact of the gold price and the total OPEX on the value of the Project, these two variables were varied concurrently, and the results of their combined sensitivity analysis are shown in Table 22.15.

Table 22.15: OPEX and Gold Price Sensitivity

Change in OPEX	NPV at Varying Gold Prices (US$/oz)				
%	1,488	1,575	1,750	1,925	2,013
−15	272	327	436	546	601
−10	246	300	409	519	574
0	191	246	**355**	465	520
10	136	192	301	411	466
15	108	164	275	384	439
NOTE: The value in bold indicates the base case NPV.					

22.9.6.3 Gold Price and CAPEX Sensitivity Analysis

In assessing the combined impact of the gold price and the total CAPEX on the value of the Project, these two variables were varied concurrently, and the results of their combined sensitivity analysis are shown in Table 22.16.

Table 22.16: CAPEX and Gold Price Sensitivity

Change in CAPEX	NPV at Varying Gold Prices (US$/oz)				
%	1,488	1,575	1,750	1,925	2,013
−15	214	268	377	487	542
−10	206	261	370	480	535
0	191	246	**355**	465	520
10	177	231	341	450	505
15	169	224	333	443	498
NOTE: The value in bold indicates the base case NPV.					

22.9.6.4 Gold Price and Gold Recovery Sensitivity Analysis

In assessing the combined impact of the gold price and the gold recovery on the value of the Project, these two variables were varied concurrently, and the results of their combined sensitivity analysis are shown in Table 22.17.

Table 22.17: Recovery and Gold Price Sensitivity

Recovery	NPV at Varying Gold Prices (US$/oz)				
%	1,488	1,575	1,750	1,925	2,013
80.6	49	96	192	285	332
85.6	96	147	246	345	394
95.6	191	246	**355**	465	520
96.6	201	256	366	477	532
98.6	210	265	377	489	545
NOTE: The value in bold indicates the base case NPV.					

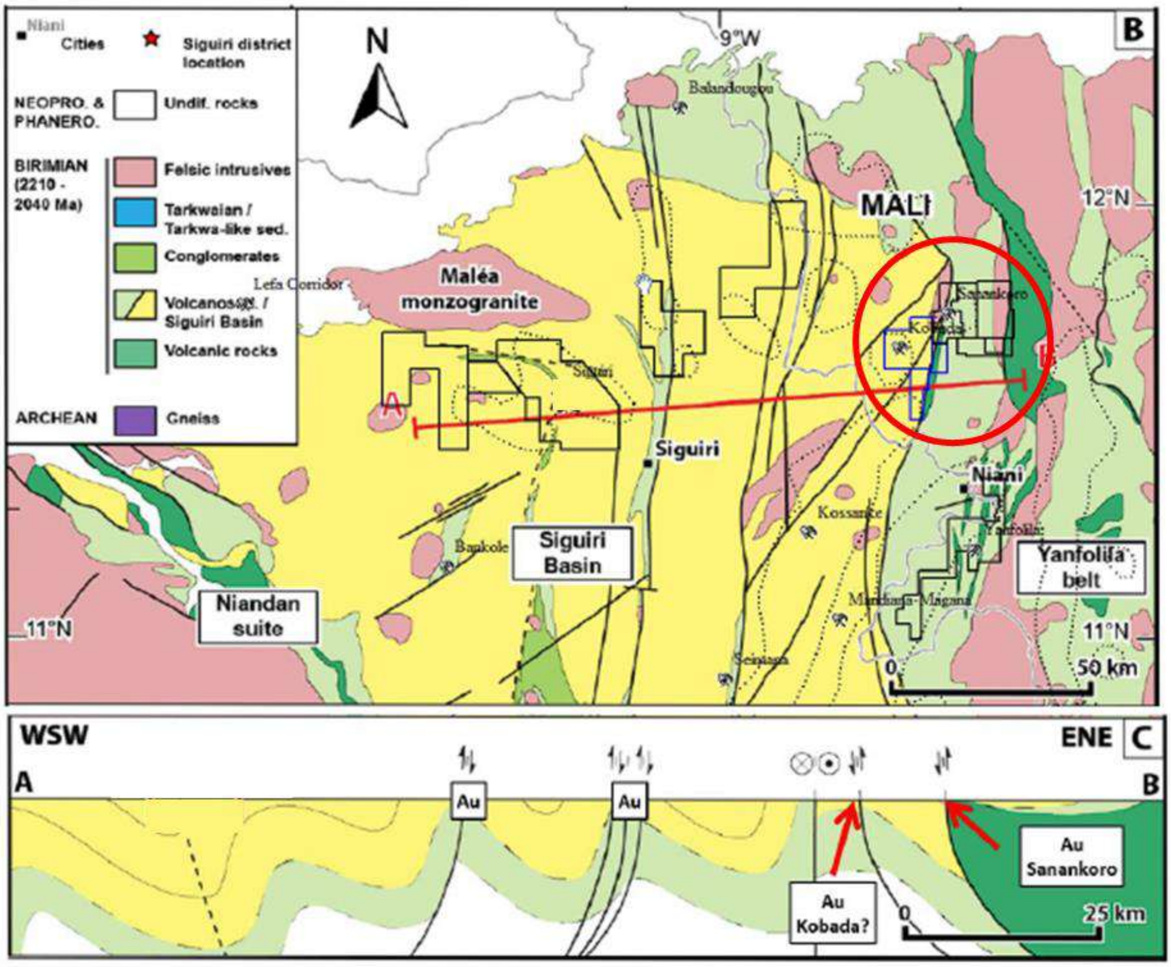
22.9.7 Discussion and Opportunities

When ranked, the sensitivity analysis indicates that the Project is most sensitive to gold price, followed by head grade, recovery, OPEX and then CAPEX.

23 ADJACENT PROPERTIES

Apart from widespread artisanal mining in the immediate area and in the Kobada Project Area, there are no adjacent properties to Kobada that currently mining operation

However, Cora Gold has recently announced an Inferred Mineral Resource of 265 koz (5 Mt at 1.6 g/t) on their Sanankoro Project to the northeast of the AGG Kobada Project. They are currently drilling target shear zones with a planned 22,000 m drilling programme. This project has been interpreted as an extension of the Kobada Est shear zone (Bell, 2020) running through the Kobada Project (see Figure 23.1).



Source: Adapted from Bell (2020)

Figure 23.1: Regional Geology showing the Kobada Project and Neighbouring Sanankoro Project

24 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT SCHEDULE

The project implementation schedule has been compiled to ensure that the engineering, procurement and construction management activities are aligned for successful project execution. The schedule also contributes to understanding the overall risk profile for the Project as the Project moves through its subsequent phases, so that the strategies adopted will mitigate possible problems and unforeseen circumstances. For this study update, a standalone implementation schedule was developed specifically for the sulphides circuit upgrade and is presented as a high-level Gantt along with the implementation schedule for the oxides phase of the project, as assembled in the 2020 DFS study.

24.1.1 Project Implementation Plan – Oxides

AGG will appoint an EPCM Contractor to execute or manage and oversee the execution (by appointed EPCM/EPC/turnkey subcontractors, who will report to and be managed by the EPCM Contractor).

The oxide ore project is due to start in January 2022 and the forecast is that the project will be completed in July 2023, achieving 100% plant throughput by September 2023.

24.1.1.1 Project Milestones

The Oxides project phase schedule assumes that there will be a seamless advancement between the various phases of the project evolution. It is also recognised that this is a moderately aggressive schedule and that it will require diligent progress monitoring and coordination of all the parties involved. The project milestones are given in Table 24.1.

Table 24.1: Project Execution Milestones

Project Milestone	Project Month
Commencement of detailed engineering	Month 1
Commencement of procurement and contracts administration	Month 3
Placement of orders for long-lead delivery items	Month 5
Mobilisation of earthworks and TSF construction contractor	Month 5
Mobilisation of Structural, Mechanical, Plate Work and Piping (SMPP) construction contractor	Month 11
Mill delivered to site	Month 14
ROM material ready to feed ROM bin	Month 17
TSF complete	Month 17
Solar plant available for start of production	Month 18
Construction complete	Month 17
Commissioning complete	Month 19
First gold from CIL circuit	Month 19
Ramp-up to 100 % of nameplate production	Month 21

24.1.1.2 Long-Lead Equipment

Placing the purchase orders for the long-lead equipment is crucial not only to ensure that the equipment is on site in time to allow for a seamless construction sequence and a successful project execution but also to obtain the certified information from the supply vendors on their equipment to complete the detailed engineering phase of the Project.

The key long-lead equipment for the Project is as follows:

- Ball mill
- Primary crusher
- Apron feeder
- Regeneration kiln
- Lime plant
- Gravity concentrator
- Intensive cyanidation package
- Mixers/agitators
- Elution heaters
- Elution column
- Acid wash column
- Blowers
- Cyclone cluster
- Linear screens
- Vibrating screens
- Vibrating feeders
- Gantry cranes
- Interstage screens
- Tower crane
- Pumps
- Dust suppression system
- Sewage treatment plant
- MV switchgear
- Solar farm thermal plant

24.1.2 Rain Delay

The rainy season for the geographical region of the Project is from May to October. It is important to note that no rain delay has been allowed for in the project execution schedule. The completion of the bulk earthworks and the construction of the TSF must be planned in such a way that the works are executed during the dry season. If the Project start date does not align with the above statement, additional time must be allowed to complete the bulk earthworks and TSF during the wet season.

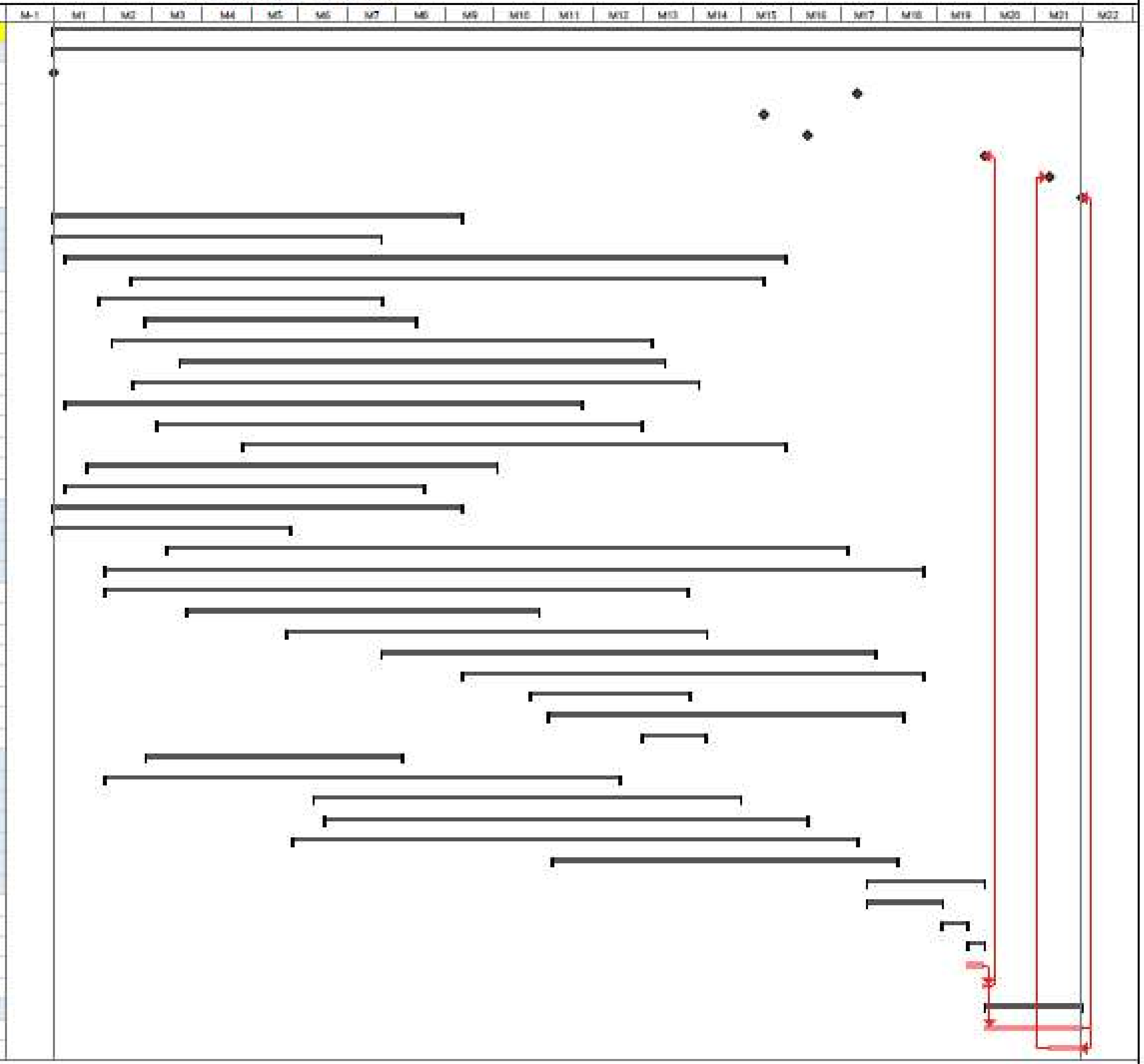
The summarised Oxides project schedule is shown in Figure 24.1.

Kobada Gold Project

Feasibility Study - Section : 16 Implementation

Project Execution Schedule

ID	Task Name	Duration
1	Kobada Gold Project - Project Execution Schedule	455 days
2	Project Milestones	455 days
3	Project start	0 days
4	Solar plant available for start of production	0 days
5	Mill delivered to site	0 days
6	Tailings dam ready to accept feed	0 days
7	1st Gold from CIL Circuit	0 days
8	Start plant performance test	0 days
9	Production to 100% of nameplate throughput	0 days
10	Engineering Design and Drawing office	181 days
736	Tailings Storage Facility Design	145 days
751	Procurement, Fabrication and Delivery to site	367 days
752	Mechanical equipment	280 days
1342	Pre-Engineered Steel Buildings	126 days
1373	Prefabricated Plant buildings	121 days
1396	Tanks	239 days
1470	Structural steel	215 days
1577	Piping packages	251 days
1716	Civil related packages	228 days
1802	Electrical equipment	215 days
1985	Control and Instrumentation equipment	241 days
2098	3 Mtpa Fuel Depot	182 days
2102	Site establishment packages	187 days
2186	Construction Contractors Appointment and Site Establishment	211 days
2218	Mining contractor appointment	105 days
2224	Solar plant contractor appointment, design and procurement	302 days
2361	Construction - Process Plant	421 days
2362	Earthworks	297 days
2420	Site establishment construction	181 days
2441	Civil Concrete Works	210 days
2522	Structural & Mechanical Erection	249 days
2859	Electrical & Instrumentation Installation	234 days
2904	Electrical Reticulation	78 days
2926	Piping Installation	183 days
2968	Power Factor Correction Equipment	30 days
2970	Construction - River Fie Low Level Bridge	135 days
2974	Construction - Access roads maintenance	264 days
2977	Construction - 3 Mtpa Fuel Depot	213 days
2980	Construction - Tailings storage facility	242 days
2982	Construction - Solar Farm	286 days
3053	Mine pit development and material stockpiling	176 days
3056	Commissioning	63 days
3057	Pre-Commision Plant	40 days
3059	Wet Commision Plant	14 days
3061	Hot Commissioning	10 days
3062	Hot Commissioning (Ore into the plant)	10 days
3063	1st Gold from CIL Circuit	1 day
3064	Production Ramp-up and Performance test	60 days
3065	Production Ramp-up to 100% of nameplate production	60 days
3066	Perform plant performance test	20 days



Page 1

Figure 24.1: Project Schedule Summary – Oxides

24.1.3 Project Implementation plan – Sulphides Phase

The mining schedule indicates that the sulphide ore will be stockpiled and ready for processing by Q3 2030, see Figure 24.2. This timeline determines that the sulphide ore project will have to commence in Q1 2029 and be commissioned by Q3 2030, ready to start treating the sulphide ore.

The sulphide ore project is estimated to take 19 months to complete.

24.1.4 Project Milestones

The project schedule assumes that there will be a seamless advancement between the various phases of the project evolution. It is also recognised that this is a moderately aggressive schedule and that it will require diligent progress monitoring and coordination of all the parties involved. The project milestones are given in Table 24.2.

Table 24.2: Project Execution Milestones

Project Milestone	Project Month
Commencement of detailed engineering	Month 1
Commencement of procurement and contracts administration	Month 1
Placement of orders for long-lead delivery items	Month 2
Mobilisation of earthworks and TSF construction contractor	Month 6
Mobilisation of SMPP construction contractor	Month 12
Mill delivered to site	Month 14
Sulphide material ready to feed ROM bin	Month 17
Construction complete	Month 18
Commissioning complete	Month 19

24.1.5 Long-Lead Equipment

Placing the purchase orders for the long-lead equipment is crucial not only to ensure that the equipment is on site in time to allow for a seamless construction sequence and a successful project execution but also to obtain the certified information from the supply vendors on their equipment to complete the detailed engineering phase of the Project.

The key long-lead equipment for the Project is as follows:

- Ball mill 39 weeks
- Cone crusher 17 weeks
- Compressed air system 12 weeks
- Cyclone cluster 16 weeks
- Screens and pan feeders 16 weeks
- Slurry Pumps 22 weeks
- Dust suppression system 16 weeks
- Conveyor belting 14 weeks
- Conveyor pulleys 12 weeks
- Conveyor drives 16 weeks

- Conveyor moving head hydraulics 14 weeks
- MV switchgear 26 weeks
- Transformers 14 weeks
- MCCs 18 weeks

The above durations are the supply/fabrication lead times ex-works from the various global fabrication locations.

The execution schedule allows an average shipping and custom clearance duration of 8 weeks from port of loading to the nominated West African port of entry. The mill shell and trunnion end shipping are estimated to take 10 weeks to reach the port of entry. A further 3 weeks are allowed for in the schedule to transport a cargo load from port of entry to the site laydown areas.

24.1.6 Mill Earthworks and Civil Foundation Construction

The construction of the mill earthworks and civil raft foundation are currently scheduled to be completed during the construction phase of the oxide ore project starting in January 2022.

The adjudicated vendor for the mill supply committed to complete the detail engineering for both the oxide mill and the mill required for the sulphide ore expansion project during H1 of 2022. They will supply the certified civil loads and layouts to SENET facilitating the completion of the detailed civil design for both mills and issuing the approved for construction drawings to the civil construction contractor.

By completing the civil foundations for the 2nd mill as part of the sulphide ore project the risk of damaging and disrupting the operational works are largely mitigated. This construction methodology will ensure that the erection of the mill access platforms will be completed, and safe access will be provided to the specialist mill installation team to install the mill mechanical equipment in an environment that will contribute to the successful completion of the project on time.

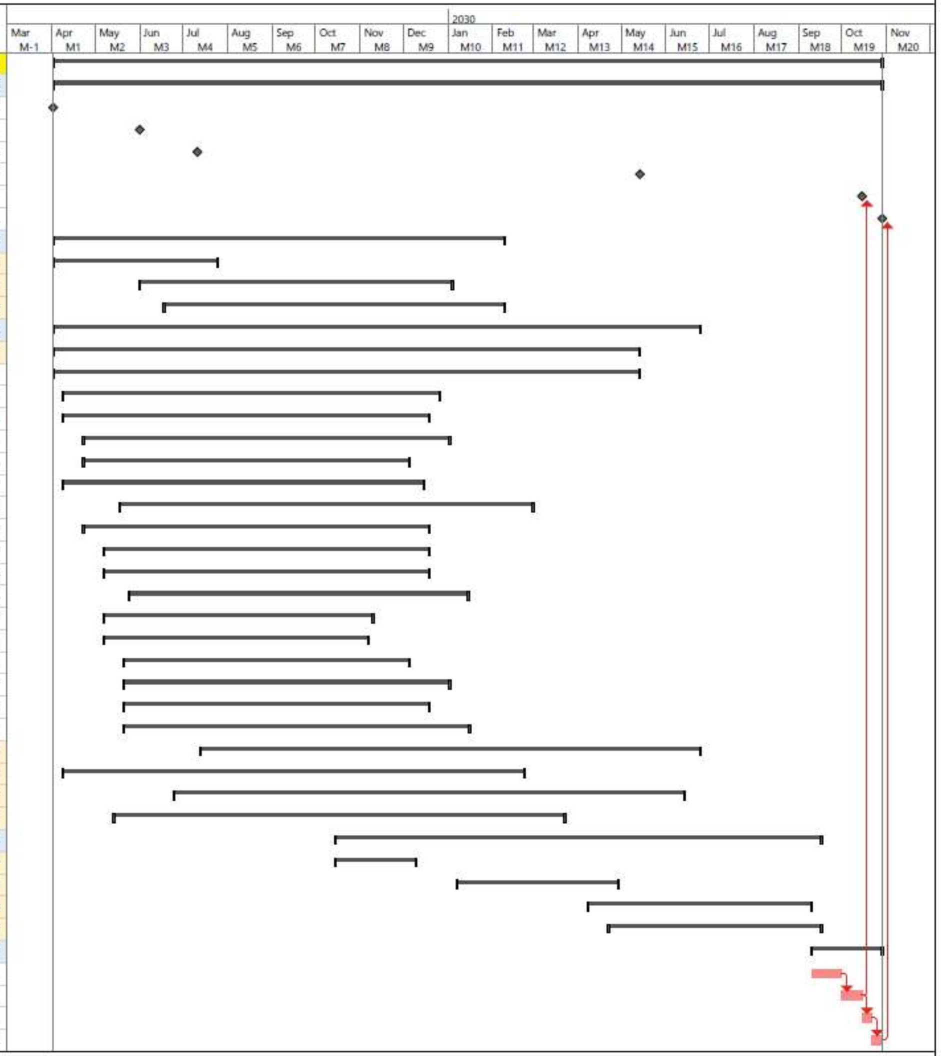
24.1.7 Rain Delay

The rainy season for the geographical region of the Project is from May to October. It is important to note that no rain delay has been allowed for in the project execution schedule. The completion of the bulk earthworks is planned in such a way that the works are executed during the dry season. The bulk earthworks for the new conveyors are estimated to take two months to complete.

SENET

Kobada Sulphide Ore Project
Execution schedule

ID	Task Name	Duration	Start	Finish
1	**Kobada Sulphide Ore Project**	**411 days**	**2029/04/02**	**2030/10/28**
2	**Key Project Milestones**	**411 days**	**2029/04/02**	**2030/10/28**
3	Project start	0 days	2029/04/02	2029/04/02
4	Issue PFD's to Client for signoff	0 days	2029/05/31	2029/05/31
5	Issue P&ID's for detail design	0 days	2029/07/10	2029/07/10
6	Mill delivered to site	0 days	2030/05/13	2030/05/13
7	Dry commissioning complete	0 days	2030/10/14	2030/10/14
8	Hot commissioning complete	0 days	2030/10/28	2030/10/28
9	**Engineering Design and Drawing office**	**225 days**	**2029/04/02**	**2030/02/08**
10	**Process Engineering**	**82 days**	**2029/04/02**	**2029/07/24**
64	**Area 3100 : Milling**	**155 days**	**2029/06/01**	**2030/01/03**
137	**Area 2310 : Secondary and Tertiary Crushing**	**170 days**	**2029/06/18**	**2030/02/08**
197	**Procurement, Fabrication and Delivery to site**	**321 days**	**2029/04/02**	**2030/06/24**
198	**Mechanical equipment**	**291 days**	**2029/04/02**	**2030/05/13**
199	**007 : BALL Mill**	**291 days**	**2029/04/02**	**2030/05/13**
210	**008 : Cone crusher**	**187 days**	**2029/04/09**	**2029/12/25**
222	**009 : Screens and Pan feeders**	**182 days**	**2029/04/09**	**2029/12/18**
234	**010 : Dust suppression compressors and associated**	**182 days**	**2029/04/23**	**2030/01/01**
246	**011 : Compressed air system**	**162 days**	**2029/04/23**	**2029/12/04**
258	**012 : Mill Cyclone cluster**	**180 days**	**2029/04/09**	**2029/12/14**
269	**060 : Slurry pumps**	**205 days**	**2029/05/18**	**2030/02/28**
280	**500 : Conveyor belting (Incl. splice kits)**	**172 days**	**2029/04/23**	**2029/12/18**
291	**501 : Conveyor pulleys**	**162 days**	**2029/05/07**	**2029/12/18**
303	**502 : Conveyor idlers**	**162 days**	**2029/05/07**	**2029/12/18**
314	**503 : Conveyor drives**	**167 days**	**2029/05/25**	**2030/01/14**
326	**504 : Conveyor belt cleaners**	**135 days**	**2029/05/07**	**2029/11/09**
338	**505 : Conveyor take-ups**	**132 days**	**2029/05/07**	**2029/11/06**
350	**506 : Conveyor belt scales**	**142 days**	**2029/05/21**	**2029/12/04**
362	**507 : Magnet and metal detectors**	**162 days**	**2029/05/21**	**2030/01/01**
374	**510 : Conveyor low speed back stops**	**152 days**	**2029/05/21**	**2029/12/18**
386	**514 : Conveyor moving head hydraulics**	**172 days**	**2029/05/21**	**2030/01/15**
398	**Piping equipment**	**247 days**	**2029/07/13**	**2030/06/24**
421	**Electrical equipment**	**230 days**	**2029/04/09**	**2030/02/22**
462	**Structural steel and platework**	**254 days**	**2029/06/25**	**2030/06/13**
499	**Construction Contractors Appointment and Site Establishment**	**225 days**	**2029/05/14**	**2030/03/22**
528	**Construction**	**241 days**	**2029/10/15**	**2030/09/16**
529	**Earthworks**	**40 days**	**2029/10/15**	**2029/12/10**
531	**Civil Concrete Works**	**80 days**	**2030/01/07**	**2030/04/29**
542	**Structural & Mechanical and Piping Erection**	**111 days**	**2030/04/08**	**2030/09/09**
566	**Electrical and Instrumentation**	**106 days**	**2030/04/22**	**2030/09/16**
593	**Commissioning**	**35 days**	**2030/09/10**	**2030/10/28**
594	Pre-commissioning	15 days	2030/09/10	2030/09/30
595	Dry commissioning	10 days	2030/10/01	2030/10/14
596	Wet commissioning	5 days	2030/10/15	2030/10/21
597	Hot commissioning	5 days	2030/10/22	2030/10/28



Page 1

Figure 24.2: Implementation Schedule Summary View: Sulphides

24.2 RISKS AND OPPORTUNITIES

24.2.1 Risks

The purpose of conducting the project risk assessment was to identify and evaluate the risks associated with the Kobada Gold Project as presented in this study. The risk assessment addresses project implementation issues including any external factors such as political risk, resource risk, financial risk (including taxation and revenues), and government legislation risk that might have an impact on the success of the Project.

The risk assessment is based on the SENET Project Risk Management Plan and, in addition to highlighting and quantifying areas of risk, it establishes the baseline upon which measures that are installed to eliminate, mitigate or transfer risk, can be evaluated. Risks were assessed as follows:

- Identification and analysis of risk
- Evaluation of the likelihood of specific risks occurring
- Evaluation of the consequences of risk occurrence
- Selection of a risk rating using the risk scoring definitions given in Table 24.3
- Production of risk assessment tables by likelihood and severity as shown in Table 24.4

Table 24.3: Risk Scoring Definitions

Score	Description	Definition
High 5	**Likelihood**	An event that is extremely or very likely to occur.
	Severity	The occurrence of this event will impact the project's cost (and/or schedule), cause sustained production interruption or delay/reduction in cash flow that leads to negative cash flow.
Medium 3-4	**Likelihood**	An event that has a 50-50 chance of occurring.
	Severity	The occurrence of this event will cause noticeable cost (and/or schedule) increases and substantial reduction in production, resulting in reduced, but adequate cash flow for project/plant production costs.
Low 1-2	**Likelihood**	An event that is unlikely or very unlikely to occur.
	Severity	The occurrence of this event will cause small or no cost (and/or schedule) increases that, in most cases, can be absorbed by the project, minor delays in production, or a reduction in cash flow, which is easily recoverable within the next operating month.

Table 24.4: Risk Scoring Matrix

Risk Level is at the intersection of Likelihood and Severity		Severity (Impact)			
		Very Low	**Low**	**Medium**	**High**
Likelihood (Probability)	**High**				
	Medium				
	Low				
	Very Low				

24.2.1.1 General Risks

The general risks associated with the Kobada Project are listed in Table 24.5.

Table 24.5: General Risks

Risk	Description	Severity	Likelihood	Mitigating Factor
Currency Exchange Rate Fluctuations	Specifically related to the strength of the euro and United States dollar, the currencies in which commodity prices are generally quoted. Risk of these currencies gaining or losing value, which will affect commodity prices, and Project CAPEX and OPEX	Medium	High	Possible purchase of forward cover and hedging
Gold Price Fluctuations	Risk of the gold price decreasing in value, which will have a negative effect on the Project economics	High	Low	The Project has shown favourable economics. Sensitivity of gold price is investigated in detail in the Financial Section (Section 22)
Country Risk	Specifically including political unrest, economic policy changes, legislative and fiscal changes	Medium	Medium	Political risk insurance on all loans
Logistics	• The Kobada Gold Project is remotely located, with some limitations to road accesses during rainy seasons. The control of the logistics and their cost implications will be fundamental in maintaining reasonable operating costs. • The import of project equipment and essential commodities, such as diesel fuel, HFO, explosives materials, plant reagents and consumables, is highly dependent on an efficient logistics system.	Medium	Medium	• AGG has a project implementation plan that considers the potential logistics challenges such as the rainy season. A construction period of 18 months has been estimated and is considered adequate, with respect to logistics. AGG will appoint a reputable Malian transporter with experience in mine projects and operations, to ensure minimal logistics problems during construction and the operation of the Project. • AGG will have a 1-month stockholding for all reagents and consumables, ensuring enough buffer to mitigate any logistics challenges.
Fuel Price Fluctuations and Supply	• Specifically related to diesel and HFO. The risk of either of the two fuel prices increasing would affect the operating cost. • The large quantities of fuel required for the mining fleet and power plant pose a potential risk for supply shortages.	High	Medium	• Close liaison with selected fuel supplier is envisaged. • Possibility of using two fuel suppliers to reduce the risk of supply shortage. The fuel storage capacity also ensures a buffer for any potential supply shortages.

Risk	Description	Severity	Likelihood	Mitigating Factor
Raw Water Supply	Risk of insufficient raw water supply to the plant from the Niger River because of, for example, a change in water abstraction rights and/or the flow dynamics of the river system.	Low	Low	The plant will direct pit dewatering water from the dewatering borehole pumps and the pit dewatering pumps as part of the raw water supply.
Global COVID-19 Pandemic	With the Project construction phase planned to start at the beginning of January 2022, disruption could be caused by the COVID-19 pandemic. This disruption could be to both construction activity and the supply of plant and equipment.	Medium	Medium	Construction will be scheduled in a flexible way in order to move priorities according to manpower and equipment availability.

24.2.1.2 Resource Risks

The resource risks associated with the Kobada Project are listed in Table 24.6.

Table 24.6: Resource Risks

Risk	Description	Severity	Likelihood	Mitigating Factor
Bulk Density	Limited bulk density samples are available for the laterite, oxide and transition zones.	Low	Low	Additional bulk sample test work should be undertaken in the next phase. Geological losses have been applied to the Mineral Resource to account for these uncertainties.
Database is a combination of analytical methodologies	The database is a combination of diamond drilling with fire assay and RC drilling with Leachwell analysis, which could impact the estimated grade.	Low	Low	Unless geological information is required, it is recommended that future drilling should be RC drilling with Leachwell analysis and fire assay on the tails. This will achieve a larger sample size and obtain a total gold grade.

24.2.1.3 Mining Risks

The mining risks associated with the Kobada Project are listed in Table 24.7.

Table 24.7: Mining Risks

Risk	Description	Severity	Likelihood	Mitigating Factor
Commodity Price	Changes in commodity price represent a significant risk. A decrease in the gold price would decrease the mineral reserves, and affect the pit design	High	Low	• Gold has shown a stable escalating trend in recent years and has a refuge value for investors. • An optimisation run used to define the reserve pit design has been done with a price that

Risk	Description	Severity	Likelihood	Mitigating Factor
				is 10 % lower than the current spot price.
Slope Stability	The stability in saprolite zones can be affected especially by the hydrogeology conditions within that sector.	Medium	Medium	• Collect hydrogeological data and develop an overall conceptual model for the mine site area, especially in the saprolite area. • Validate the dewatering volume with a more detailed hydrogeological study. • Run finite element stability analysis including hydrogeological expected conditions
Drill and Blast	It has been estimated that approximately 5 % of saprolite and laterite will require drilling and blasting. There is a possibility that this percentage will be higher, resulting in a higher OPEX.	Low	Low	Once production progresses, blasting patterns will be adjusted to avoid increases in operating costs for this concept.
Waste Dump and Stockpile Stability	Dumps and stockpile design exceed industry standards; these may be oversized.	Low	Low	The final size of the dumps and stockpiles does not appear to be a problem, but if it is necessary to separate the material the slopes can be adjusted to optimise the space and final size.

24.2.2 Process Plant Risks

The process plant risks associated with the Kobada Project are listed in Table 24.8.

Table 24.8: Process Plant Risks

Risk	Description	Severity	Likelihood	Mitigating Factor
Ore Variability: Comminution Characteristics	Oxide saprolite ore has been tested for the study, and the results showed very soft ore. Laterite (oxide) and transition ores were not tested. Historically, laterite ore has similar comminution characteristics to the saprolite ore; however, transition ore has proven to be harder and would require more power to achieve the required CIL feed PSD.	Low	Low	The proportion of transition ore to saprolite is very low – averaging approximately 10 %, and the impact on the overall plant throughput is likely to be less. In addition, there is enough buffer in the design to handle a ROM feed blend of transition to oxide of no more than 10:90. Alternatively, the circuit can treat a reduced throughput of ROM when treating 100 % transition ore.
Ore Variability: Leach and	Oxide saprolite ore has been tested for the study, and the	Low	Low	• Test work has been recommended for laterite and

Risk	Description	Severity	Likelihood	Mitigating Factor
Recovery Process	results showed very high kinetics with dissolutions of > 90 %. Laterite (oxide) and transition ores were not tested. A potential decrease in recoveries can be expected, which would lead to a lower cash flow.			transition ores. There is an additional space allowance for an extra CIL tank to give extra residence time should it be required as per the test work results. This will ensure minimal interruption in the execution of the Project. • The recovery circuit has been designed with a buffer to handle variability.
ROM Ore Gold Grade Variability	An increase in the ROM gold feed grade that can lead to reduced recoveries.	Low	Low	Variability tests conducted indicated high gold recoveries on low and high grades The recovery circuit design allows for variability. If there is a substantial change in feed grade, a space allowance for an additional CIL tank has been made.
Arsenic and Cyanide-Contaminated Water Discharge into the Environment	There is a possibility of arsenic and cyanide-contaminated water discharge to the environment due to the TSF overflowing during the rainy season.	Medium	Medium	• The design allows for arsenic precipitation prior to discharge to the TSF. Additional space for two detoxification/arsenic precipitation tanks has been allowed for should there be a need for additional residence time. The design has allowed for a secondary detoxification process at the TSF in the event of water overflow. • Test work to confirm the arsenic removal process requirements prior to TSF deposition has been recommended.

24.2.2.1 TSF Risks

The TSF risks associated with the Kobada Project are listed in Table 24.9.

Table 24.9: TSF Risks

Risk	Description	Severity	Likelihood	Mitigation or Management Measures
Geotechnical Test Work on Soils from Geotechnical Site Investigation	Design assumptions may prove to be non-conservative and fail to satisfy minimum slope stability criteria of the tailings dam.	Medium	Low	• Once completed, use the geotechnical laboratory test work for the samples from the geotechnical site investigation to validate and confirm the design assumptions used in the DFS. Due to movement restrictions associated with the COVID-19 Pandemic, testing commenced at the beginning of June and is due to be completed mid-July 2020. • If warranted, flatten the side slopes accordingly to increase slope stability.
Geotechnical Test Work on Tailings	The design assumption for dry density may differ from the test results and implicate the storage capacity of the TSF.	Medium	Low	• Once completed, use the geotechnical laboratory test work for the tailings to validate and confirm the design assumptions used in DFS. Test work is currently under way. • If warranted, TSF storage capacity can be increased by relocating the TSF main embankment.
Geotechnical Test Work on Open-Pit Overburden Material	Open-pit material sourced for the TSF main embankment might not be deemed suitable.	Low	Medium	Material shall be sourced from surrounding borrow pits.
Geochemical Test Work	• The analyses conducted on the samples representative of the waste material indicated very little reactivity, both from an acid generation and mineral liberation perspective. The mobilisation of iron and arsenic was limited. • Material for the construction of the embankment wall shall, therefore, be sourced from borrow pits if the open-pit overburden material proves to be unsuitable for use.	Low	Low	Kinetic testing on the open-pit overburden material is required to provide a more accurate assessment of any risk posed by the material.

Risk	Description	Severity	Likelihood	Mitigation or Management Measures
TSF and Embankment Failure due to External, Side Slope Erosion	Side slope surface erosion due to surface runoff	High	Low	The TSF shall be rehabilitated during the LOM, which should mitigate erosion of the embankment outside slopes.
Overtopping due to Non-Compliance with the Operating Procedures	Tailings pond spilling over the crest of the embankment wall	High	Low	Emergency spillways have been incorporated into the design.

24.2.2.2 Sustainable Development Risks

The sustainable development risk assessment identified the following key risks and as well as the measures that can be implemented to mitigate or reduce each risk. These are outlined in Table 24.10.

Table 24.10: Sustainable Development Risks

Risk	Description	Severity	Likelihood	Mitigation or Management Measures
Licence and Permitting	The environmental permitting process is ongoing and needs to be completed before development of the project can be undertaken.	Medium	Low	Regular interaction with the authorities to identify and address any potential issues throughout the permitting process.
Water Contamination	The transportation of cyanide over the Fié River (via the barge) poses a high risk of contamination of the surface water in the event of a spill.	High	Low	• Limit transportation of cyanide and other hazardous materials along the alternative access route. • Develop the Cyanide Transportation Plan. • Include a risk assessment of the transportation route, considering water crossings, population centres, road characteristics, weather characteristics, and public infrastructure. • Implement industry best management practices.
Geochemical Test Work	• Arsenic is a significant concern. The As concentration of the sulphide ore and waste material significantly exceed the test results of	Medium	Low	• Separation of oxide and sulphide waste • Separation of oxide and sulphide ore (which the layout provides for)

Risk	Description	Severity	Likelihood	Mitigation or Management Measures
	the oxide material tested during the Oxides DFS. • The interpretation of the data generated during the static test programme supports two primary conclusions. Most of the material tested exhibited limited acid generating potential and moderate acid neutralising potential. Therefore, the risk of acidic drainage is low. • There is a risk that leachate arising from the percolation of water through tailings, waste rock impoundments and ore stockpiles could accumulate arsenic to concentrations more than the Mali standards. Kinetic tests would provide more meaningful data on the rate and extent of arsenic mobilisation from material with a larger particle size.			• Special pollution control and prevention measures for the sulphide material, such as: o Providing for an impermeable layer (such as a 500 m compacted clay liner) for the sulphide waste material o Lined Pollution control dams (PCDs) to collect surface runoff from the sulphide WRD facility o Impermeable layers for the sulphide ore stockpiles o Lined PCDs for the surface runoff from sulphide ore stockpiles o Prioritising the reuse of water from these PCDs in the plant circuit. • It is recommended that selected samples be presented for kinetic test work and the geochemical risk assessment be updated. .
Post-closure decant from the pit.	Decant risk from the main pit is considered low. It is however recommended that the long-term water quality be assessed through the development of geochemical model.	Medium	Low	• Assessing the TSF decant water quality and whether it complies with the IFC PS and Malian requirements • Conduct a detailed geochemical pit lake quality study to calculate the long-term water qualities in the pit.
Groundwater	All the wells within the Kobada village are expected to be impacted to some extent by the groundwater level drawdown cone.	Medium	Medium	• Perform additional aquifer tests on boreholes distributed throughout the Project Area as these will confirm the aquifer transmissivity range. • Additional/improved water supply would be required as part of a community development programme sourced from outside the dewatering cone zone of influence.
Air Quality and Noise	Air quality and noise – exceedance of WHO, IFC and SANS 10103 guidelines, respectively	Medium	Low	• Implement a minimum buffer of 350 m between the mining operations and any sensitive receptor locations to protect the

Risk	Description	Severity	Likelihood	Mitigation or Management Measures
				health of residents of nearby villages. A total of 1229 structures located within this recommended buffer would likely need to be relocated. • For material that will be blasted a Management Plan will be put in place.
Socio-Economic Studies and Resettlement	Several migrants moved into the Project Area after the current exploration and mining permits were issued and established settlements, likely without permission from the local and traditional leaders or permission from the mining permit holder. It is likely that more opportunistic in-migration may take place after the approval of the project.	Medium	Medium	• Develop and implement a RAP. • Undertake community interactions with local and traditional leaders. • Development of an influx management plan. • The study adopted a 500 m buffer from the mining operations for people to the resettled. Technical studies indicate that a 350 m buffer is considered adequate. The opportunity exists for the reduction in the number of structures affected by the mining operations. This will be confirmed via monitoring programmes.
Resistance to Resettlement	There may be resistance to resettlement or partial resettlement of the Kobada village, where the community insists on the complete resettlement of the Kobada Village.	Medium	Low	• Ongoing consultation • Implement CDP with a focus on parties that are not relocated in the Kobada village.
Pressure on Social Infrastructure	Increased pressure on social infrastructure due to an influx of people and potential conflict due to competition for resources.	Low	Medium	• Develop and implement influx management measures.
Community Expectations	Several community consultations have been undertaken and community expectations have been documented.	Medium	Low	• Ongoing consultation and interaction with communities and stakeholders. • Management of community expectations through clear communication on plans to improve livelihoods of the people.
Occupational and Community Health	Disruption due to illness	Medium	Medium	Develop and implement Community and Occupational Health Impact Management Plan.

Risk	Description	Severity	Likelihood	Mitigation or Management Measures
Climate Change	Climate change may affect a wide range of aspects associated with the Project, such as reliable water supply, storm water management, disruption of operations, food supply, worker health and safety, as well as community health and safety.	Medium	Low	• Develop and implement a climate-change adaptation strategy for AGG. • Undertake vulnerability risk assessments at all AGG's operations and host communities. • Develop and implement plans that respond to material climate risks. • Improve efficiencies in the use of natural resources (energy and water). • Implement community awareness and resilience strategies. • Undertake a climate change risk assessment of the project.
Artisanal and Small-Scale Mining	• The development of the project will result in the loss of certain artisanal mining sites. • A loss of income will be incurred. • The closure of some artisanal sites could contribute to increasing the population's expectations towards AGG to access salaried jobs under the Project. • The development of the Project will result in increased internal tensions and conflicts.	High	High	The impact expected on artisanal activities is limited to areas specifically earmarked for development. An artisanal mining development plan has been developed for the project. Development of a livelihood restoration plan
Blasting	Impact on structures and safety of people within blasting zone of influence.	Medium	Low	Implement controlled blast technology to reduce the negative effects of blasting. Development and implementation of the Resettlement Action Plan. While a buffer zone of 350 m is considered adequate, a 500 m buffer zone was adopted for this study. This will result in a cost and time saving on the RAP implementation.
Environmental Release of Water from Pit and TSF	The water balance reflects a positive water balance and will require the release of excess water from the pit during the wet season. The expected water quality is to be	Low	Medium	Kinetic Test Work and geochemical modelling to confirm the likely water quality of the pit lake quality. Using pit lake water to meet makeup water requirements at the plant.

Risk	Description	Severity	Likelihood	Mitigation or Management Measures
	determined to confirm whether it complies with the relevant standards.			Treatment prior to release. Containing of water in selected areas in the pit during the wet season, to reduce the volume of water to be released. Limiting the release of pit lake water into the environment to wet seasons only. No releases to be allowed when water quality does not meet relevant guidelines and standards.

24.2.3 Opportunities

The opportunities that have been identified for the Kobada Project are outlined in Table 24.11.

Table 24.11: Opportunities

Opportunity	Description
Gold Price	The Project's viability was evaluated at a gold price of US$1 530. The current gold price is > US$1 816 and increasing.
Mineral Resource	With limited additional drilling there is an opportunity to convert further Inferred oxide Mineral Resource to an indicated Mineral Resource and increase the Mineral Reserve and LOM within the current pit limits.
Exploration upside	The current mineral resource is focused on a 4 km strike of the Main Shear zone. A further 51 km of shear zones with similar structural geology have been identified on the concession through regional exploration techniques. Additional drilling within these identified shear zones may increase the size of the overall resource.
Construction Schedule Optimisation	Ordering long-lead items as early as is practically possible, even prior to detailed engineering, will greatly improve the Project schedule.
Marginal Ore	If it is decided to use a higher gold price for the COG calculation, there is a possibility of increasing the amount of marginal ore to be stockpiled and then processed in order to increase the LOM.
Underground Mine	The Kobada deposit is currently interpreted open at depth and hence underground mining could be investigated for viability after the open pit reaches a certain depth.
Transition Ore	A conservative assumption was made when considering the transition ore's characteristics and metallurgical performance. With further test work, there is an upside potential to optimise gold recovery and OPEX.
TSF Geotextile Underlying the HDPE Liner	There is an opportunity by excluding the geotextile beneath the HDPE liner.
TSF Drainage System	There is an opportunity for the optimisation of the various TSF drainage systems.

24.3 UPSIDE POTENTIAL

The upside potential for the current Mineral Resource at the Kobada Main Shear is that there are still Inferred category Mineral Resources within the laterite, oxide (saprolite), transition and sulphide zones that can be drilled further to upgrade a portion of these to Measured and Indicated Mineral Resources.

In addition to this Mineral Resource upgrade, there is a further 55 km of potential mineralised shear zones that require further exploration (see Figure 24.3).

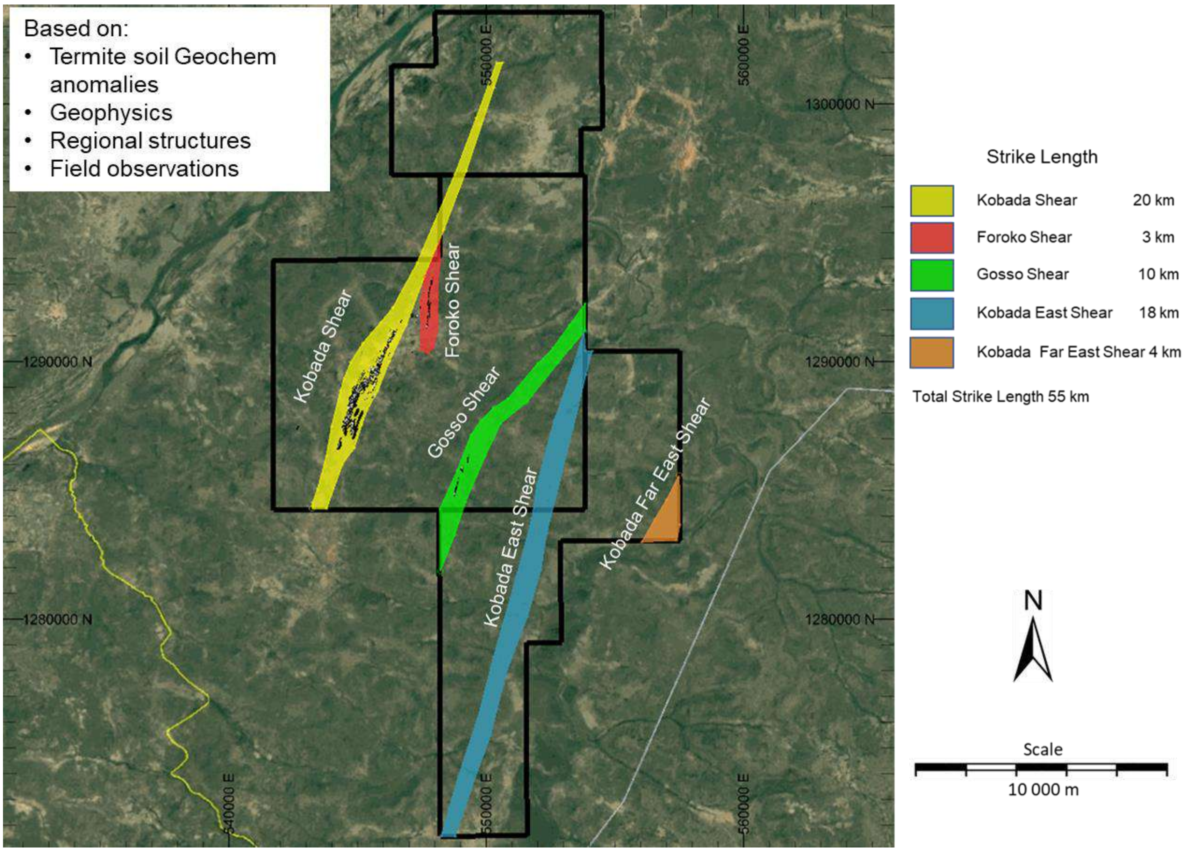


Figure 24.3: Plan Showing the Potential Mineralised Shears at the Kobada Project

A drilling programme has been planned over the various target areas, including additional drilling at the already drilled Foroko shear, which forms part of the current Mineral Resource at Domain 5 (Far North Orebody). Diamond drilling would need to be carried out across the orebody to test the geological model and the estimation.

The remaining target areas (Kobada West, Foroko, Foroko North, Diaban, Gosso and Siramana) have been subjected to limited exploration work, and further exploration of these targets could add more gold ounces to the Kobada Project (see Figure 24.3).

Historically, limited RC drilling has taken place in the Gosso Target, and in 2020 four additional diamond drillholes were drilled at Gosso.

Historically, 17 RC drillholes have been drilled at the Gosso Target (see Figure 24.4), and of these, 12 of the drillholes intersected mineralisation. The significant intersections (mineralised zones above 0.3 g/t) are summarised in Table 24.12.

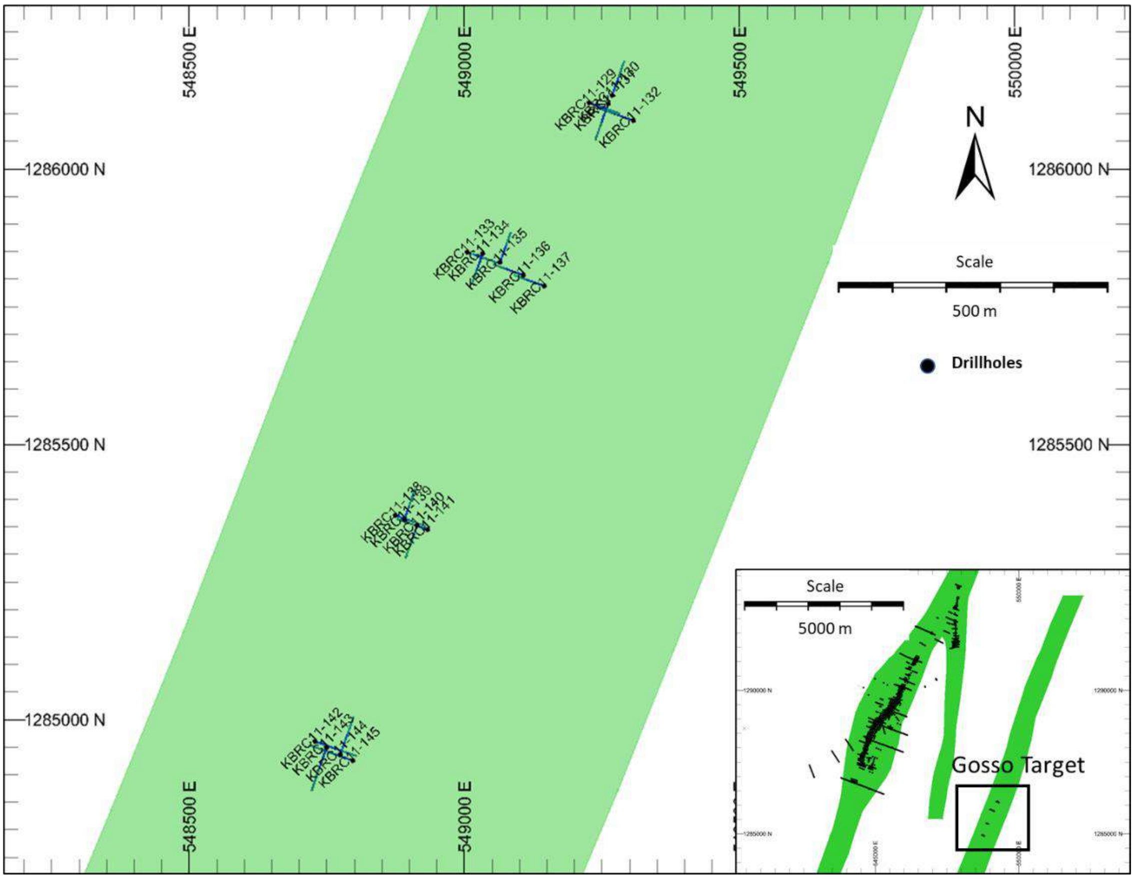


Figure 24.4: Gosso Target Drilling

Table 24.12: Significant Intersections for the Gosso Target Drillhole for the Historical Drilling

BHID	Orebody	Mineralised Zone				Includes			
		From (m)	To (m)	Composite Length (m)	Composite Grade (g/t)	From (m)	To (m)	Length (m)	Grade (g/t)
KBRC11-129	Gosso Target	2	4	2	0.53	2	3	1	0.66
KBRC11-130	Gosso Target	91	94	3	0.36	92	93	1	0.58
KBRC11-130	Gosso Target	102	114	12	0.19	108	109	1	0.96
KBRC11-132	Gosso Target	52	58	6	0.90	55	56	1	2.59
KBRC11-133	Gosso Target	6	17	11	0.55	15	16	1	2.88
KBRC11-134	Gosso Target	63	64	1	0.71	72	73	1	0.19
KBRC11-134	Gosso Target	91	93	2	0.32	91	92	1	0.50
KBRC11-137	Gosso Target	71	100	29	0.45	78	79	1	1.44
KBRC11-137	Gosso Target					80	81	1	1.32
KBRC11-138	Gosso Target	65	81	16	0.59	66	67	1	1.44
KBRC11-138	Gosso Target					70	71	1	1.99

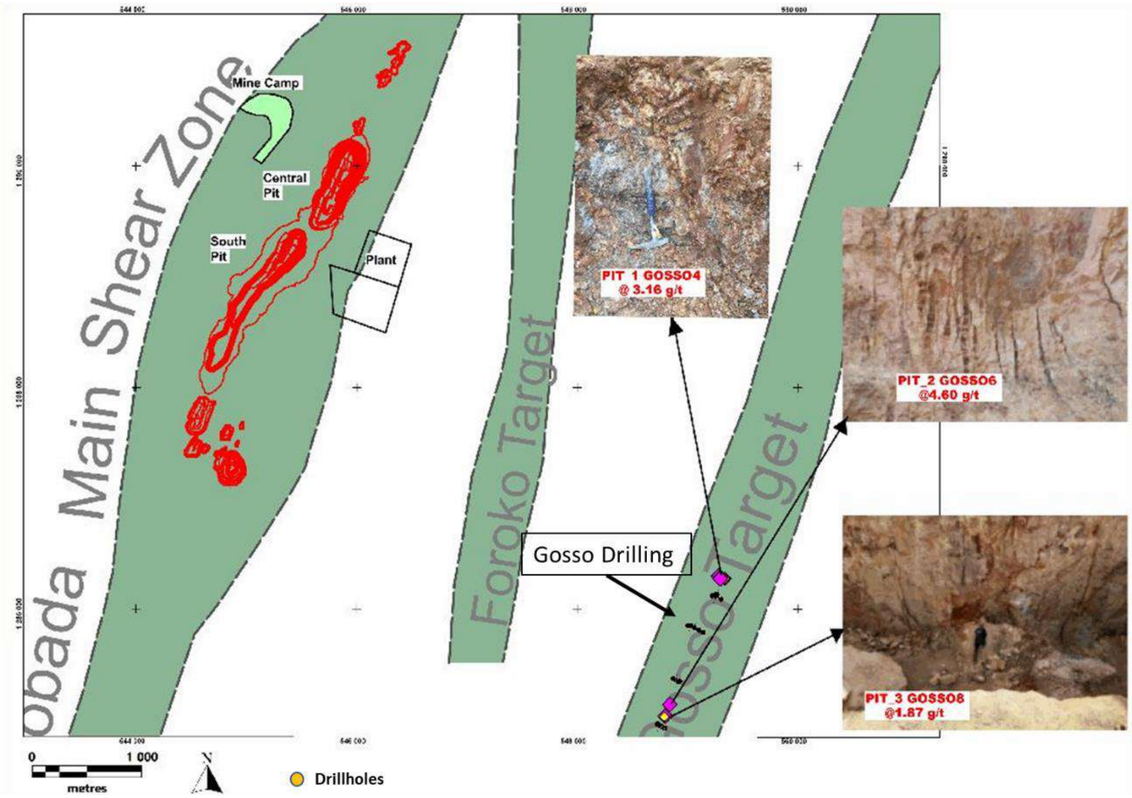
BHID	Orebody	Mineralised Zone				Includes			
		From (m)	To (m)	Composite Length (m)	Composite Grade (g/t)	From (m)	To (m)	Length (m)	Grade (g/t)
KBRC11-138	Gosso Target					71	72	1	1.35
KBRC11-138	Gosso Target					74	75	1	1.20
KBRC11-138	Gosso Target					80	81	1	1.19
KBRC11-140	Gosso Target	72	108	36	1.64	76	77	1	4.68
KBRC11-140	Gosso Target					79	80	1	1.45
KBRC11-140	Gosso Target					81	82	1	2.26
KBRC11-140	Gosso Target					82	83	1	2.63
KBRC11-140	Gosso Target					88	89	1	5.77
KBRC11-140	Gosso Target					90	91	1	3.21
KBRC11-140	Gosso Target					91	92	1	5.63
KBRC11-140	Gosso Target					94	95	1	3.11
KBRC11-140	Gosso Target					96	97	1	11.82
KBRC11-140	Gosso Target					105	106	1	6.83
KBRC11-140	Gosso Target					106	107	1	1.37
KBRC11-141	Gosso Target	44	49	5	0.42	45	46	1	0.86
KBRC11-142	Gosso Target	133	138	5	1.90	134	135	1	1.38
KBRC11-142	Gosso Target					135	136	1	4.01
KBRC11-142	Gosso Target					136	137	1	3.60
KBRC11-144	Gosso Target	1	3	2	0.75	1	2	1	0.85
KBRC11-144	Gosso Target	12	60	48	0.47	30	33	3	2.59
KBRC11-144	Gosso Target					36	37	1	3.07
KBRC11-145	Gosso Target	33	62	29	0.34	34	35	1	3.30
KBRC11-145	Gosso Target					50	51	1	1.40

The significant intersections for the four drillholes drilled at Gosso in 2020 are shown in Table 24.13. The four drillholes have an average grade of 1.11 g/t over 12 m for the cumulative mineralisation zones.

Table 24.13: Significant Intersections for the Gosso Target Drillhole for the 2020 Drilling

BHID	Orebody	Mineralised Zone				Includes			
		From (m)	To (m)	Composite Length (m)	Composite Grade (g/t)	From (m)	To (m)	Length (m)	Grade (g/t)
G20_PH3A_20	Gosso	18.10	23.30	5.20	1.55	18.10	19.10	1.00	6.38
G20_PH3A_20	Gosso	34.60	37.70	3.10	4.25	36.50	37.70	1.20	10.40
G20_PH3A_23	Gosso	91.00	104.50	13.50	0.50	100.50	102.30	1.80	1.71
G20_PH3A_23	Gosso	115.60	119.70	4.10	0.31				
G20_PH3C_16	Gosso	13.00	16.10	3.10	0.71				
G20_PH3C_16	Gosso	22.10	23.70	1.60	1.74	22.10	23.70	1.60	1.74
G20_PH3C_16	Gosso	39.00	51.50	12.50	1.15	40.00	41.30	1.30	7.19
G20_PH3C_11	Gosso	0.00	2.00	2.00	0.38				
G20_PH3C_11	Gosso	46.30	47.40	1.10	1.64	46.30	47.40	1.10	1.64

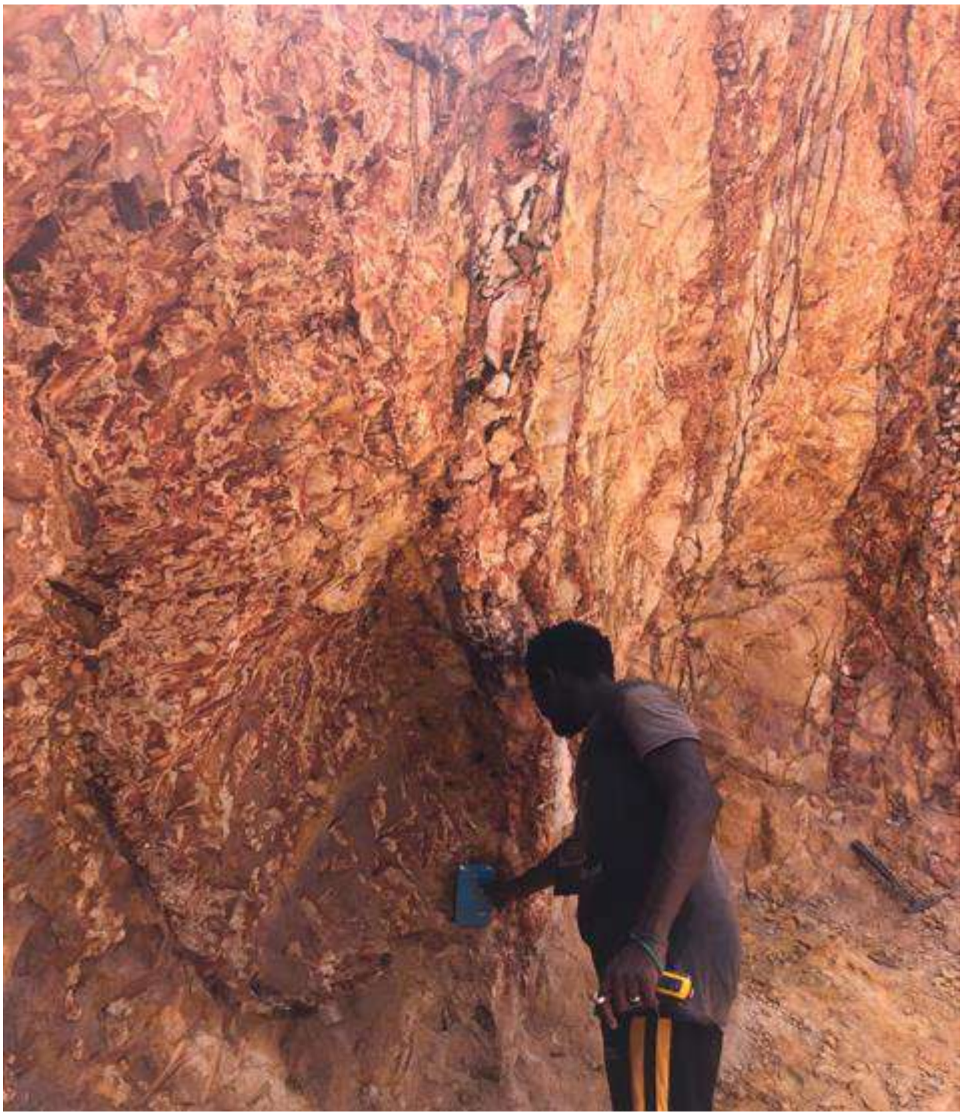
In addition to this drilling, recent grab samples from artisanal pits (see Figure 24.5) have been sent for analysis by the AGG field geologists. Eight grab samples were collected from three artisanal pits with a minimum of 0.05 g/t, maximum of 4.60 g/t, and an arithmetic mean of 1.97 g/t.



Source: AGG Field Geologists

Figure 24.5: Recent Gosso Target Grab Sampling in Artisanal Pits

Field work undertaken by the AGG geologists in early 2021 (see Section 9) at Kobada Est has highlighted the potential of this shear. The artisanal workings have exposed good examples of mineralised quartz veins, foliation and stockwork along the Kobada Est target (see Figure 24.6).



Source: AGG Field Geologists

Figure 24.6: Mineralisation in Artisanal Pits in the Kobada Est Target

25 INTERPRETATION AND CONCLUSIONS

25.1 MINERAL RESOURCE

Minxcon reviewed all the historical information in conjunction with the recent drilling campaigns and has made the observations given below regarding the Project.

25.1.1 Database

There is sufficient data density and quality to be utilised in the generation of a geological model and Mineral Resource.

The 2019 and 2020 drilling confirms the historical drilling programmes and the correlation between the old and new databases enables the use of the historical data along with the new data (see Figure 25.1).

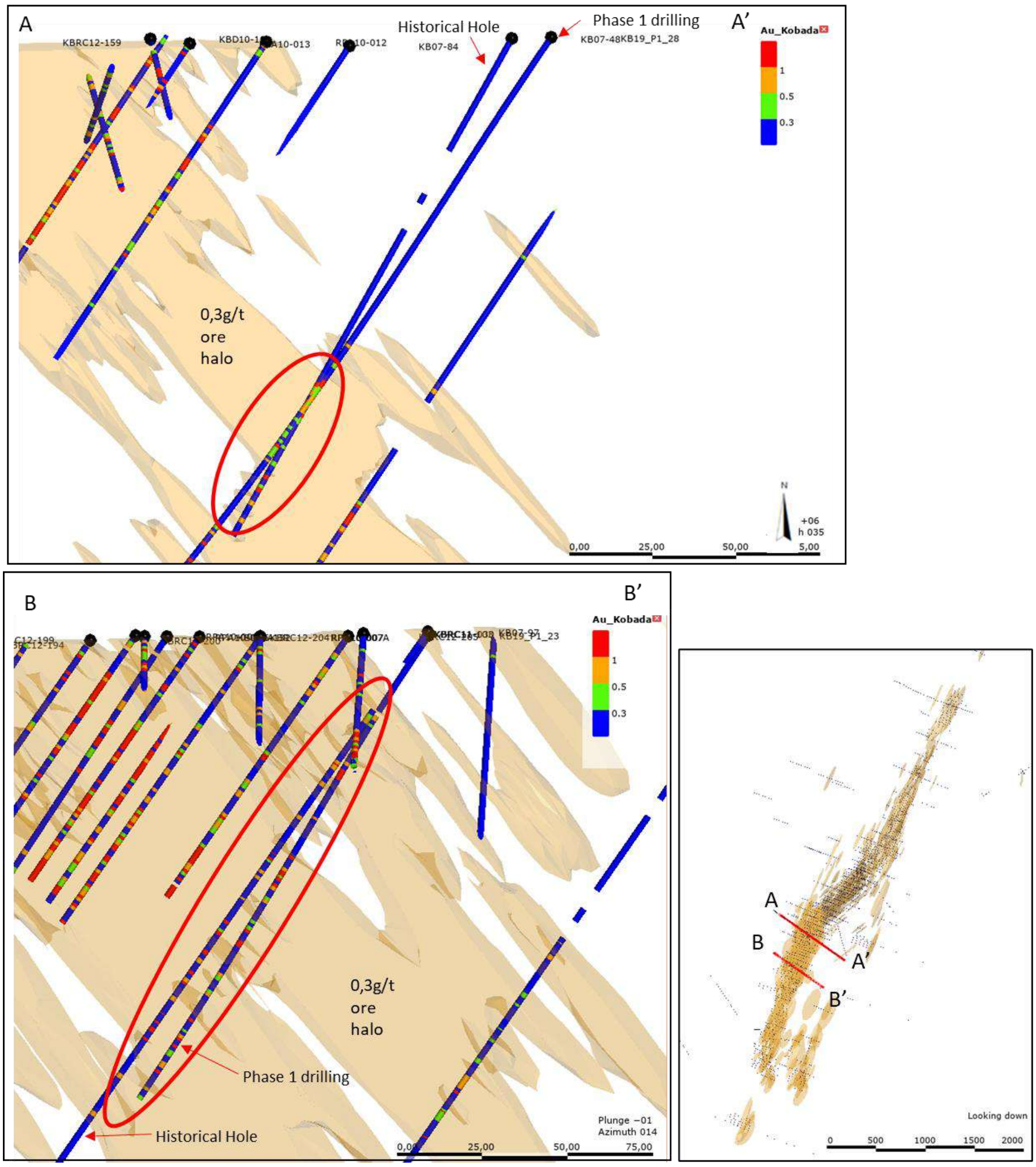


Figure 25.1: Comparison of Historical Drillholes and nearby Phase 1 Drillholes

There is, however, a concern with respect to the Leachwell sampling not providing a true representation of the grade as it is not a total gold grade but only a partial gold grade based on cyanidation leach. If the recovery is low, then the Leachwell sampling could be under-estimating the sample grade. On the other hand, the larger sample size utilised in the Leachwell process is better for the analysis of coarse gold and when there is a high nugget effect as is the case of Kobada. Therefore, diamond core samples are not ideal either. It is thus recommended that further RC drilling should be sampled and analysed using Leachwell, with a fire assay analysis done on the tails to obtain a total gold grade. This is if geological or structural data is not required as the RC samples have limited geological data. The current Kobada database is a combination of Leachwell (partial gold) and fire assay (total gold).

It is the QP's opinion that the database is reliable enough for the estimation of a Mineral Resource, but the risk is that the Mineral Resource grade might be understated as a result of sampling methodologies. It would therefore be recommended that the Leachwell analysis be done with a fire assay on the tails to achieve a larger sample size and obtain a total gold grade. An exercise comparing the pre-2018 data set (predominantly Leachwell) to the post-2018 data set (diamond drilling) has shown a 9 % difference (the Leachwell sampling had a higher grade), and an exercise comparing the Leachwell only with total gold has shown a difference of 9 %. It is, however, difficult to reliably quantify what that understatement is. As an indicative range, the above exercises would suggest that it is between 5 % and 10 %. This could be seen as a possible upside potential, i.e. the recovered grade could be higher.

25.1.2 QAQC

The historical QAQC data was reviewed to determine the nature of the database, and the following was concluded:

- Prior to 2009, limited or no QAQC data is available; thus, there is a lower level of confidence in the data and QAQC.
- From 2009 to 2012, detailed QAQC data is available; however, a negative bias is observed in the QAQC results. Thus, these samples should be considered with a medium level of confidence. However, this could be due to the erroneous CRMs being used/prepared at the time.
- 2007 and 2015 show good QAQC results and can be assigned a medium to high level of confidence (however, none of this raw data is available, only the results in reports).
- 2018 and 2019 show good QAQC results and can be assigned a high level of confidence.

The 2019 drilling campaign was designed as an infill and confirmatory drilling campaign to cover the entire Kobada Main Shear Zone from south to north. This drilling campaign was utilised to confirm the previous database in addition to its other aims of gathering additional geological data and upgrading the Mineral Resource. It is the QP's opinion that it succeeded in confirming the previous database.

The 2020 drilling campaign, which was an extension of the 2019 Phases 1 and 2, was focused on the gap area in the northern extension of the main shear (Domain 3). This drilling was successful in that a portion of the Mineral Resource could be upgraded from Inferred to Indicated. This is evident in the increase in the updated Mineral Resource. This drilling also confirmed once again the geological model that has been developed for the Kobada Main Shear.

25.1.3 Geological Model

A geological model was generated, incorporating grade shells of all the samples ≥ 0.3 g/t cut-off. These grade shells constrained the estimation. The trend and orientation of the shells were guided by existing structural knowledge and work that was performed on the property. The structural model on which the modelling process was based is a dextral shear system with the principal shear orientations trending NNE, roughly E-W striking Riedel shears and extensional veins, and approximately north-striking conjugate sets. In addition, a weathering profile was generated to represent the weathering intensity with an increase in depth. Both these models were used in the definition of the domains that were used in the estimation (see Figure 25.2). The geological model enabled a further validation of the historical (predominantly RC) versus new data sets. The revised geological model, orientations and trends that were applied based on the historical data remained unchanged when the new 2019 and 2020 drilling data was added. The new data served to update and improve the confidence in the new model and historical assay database, with no change to the interpretation of the geological model's continuity or trend.

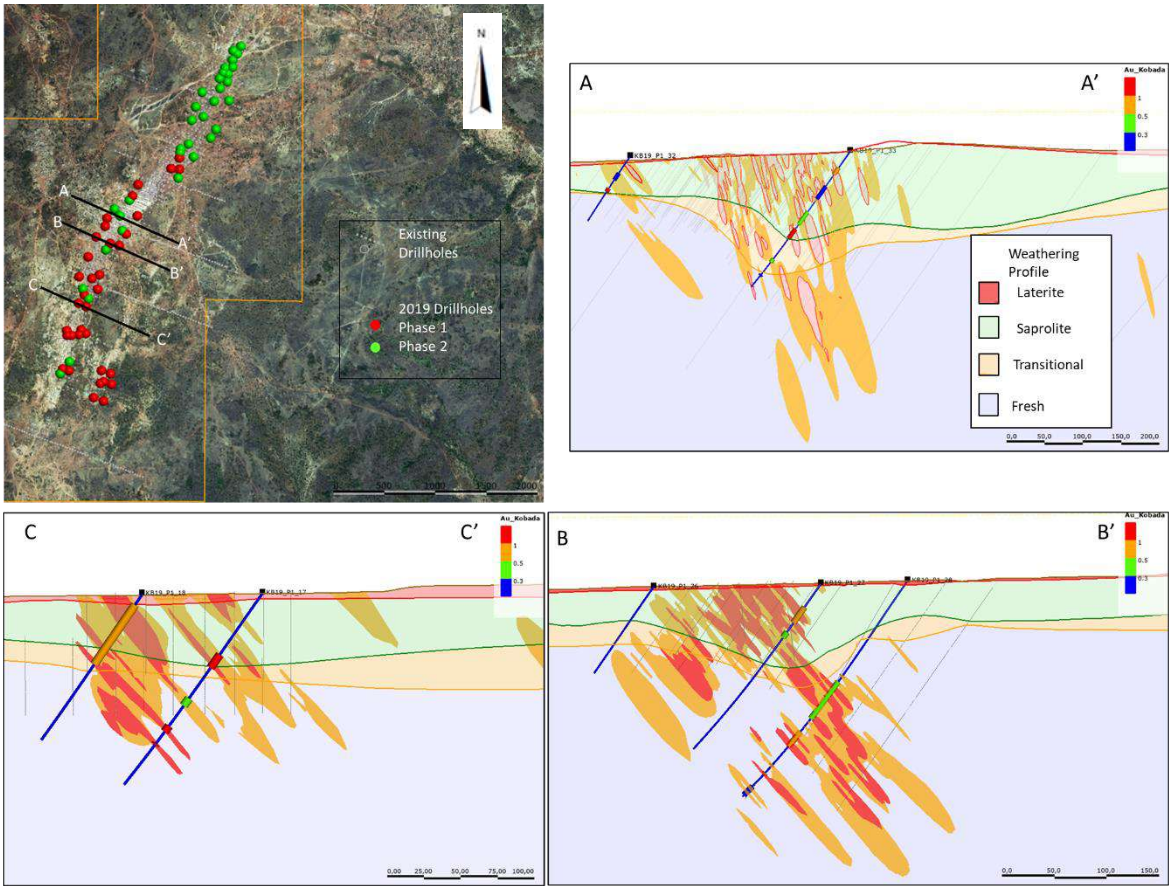


Figure 25.2: Sections through the Orebody Showing Weathering Profiles and Grade Shells and How These Were Updated to Correspond with New Drilling Information

It is the QP's opinion that the revised and updated geological model better represents the orebody when compared to the previous probability model because of the additional structural/geological parameters gathered and utilised in the modelling process. In addition, the OK model also seems to represent the database more accurately than the previous Multiple Indicator Kriging model, which resulted in an elevated grade and lower tonnage and which could result in an incorrect mining strategy.

26 RECOMMENDATIONS

The following recommendations are proposed for the Project:

- Infill drilling on the Kobada Main Shear Zone should be a priority to upgrade the Mineral Resource even more and subsequently increase the Mineral Reserve, which will in turn increase the LOM. The initial infill drilling should focus on the northern extension of the main shear (Domain 3).
- The additional exploration targets should be investigated further to understand the full potential of the Kobada Gold Project, starting with the exposed shear zone at the Gosso and Kobada Est targets.

The following recommendations are proposed for consideration and evaluation during the detailed design of the TSF:

- The geotechnical site investigation should be completed to include test pits in the revised embankment wall area in the early stages of the detail design phase.
- Undertake a select number of geotechnical drillholes in the location of the TSF main embankment wall.
- Determine the geotechnical and geochemical nature, suitability, and selected sourcing of the open-pit overburden material to be utilised in the construction of the TSF embankment walls.
- Undertake a more detailed flow slide assessment of the TSF to revalidate the zone of influence and path travelled by the pond water and liquefied tailings.
- Implementation of the Resettlement Action Plan.
- Assess possible further optimisation of the TSF drainage system.
- Assess possible further optimisation of the TSF storm diversion system.
- Assess further optimisation of closure costs.
- Appropriately size and position the TSF emergency spillway to pass the inflow design flow, considering flood routing through the dam.

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