Bankan Gold Project
Siguiri Basin, Guinea
Technical Report

Effective Date: 31 July 2025

Authors

Philip Jankowski, MSc, FAusIMM

Ross Cheyne, BEng (Mining), FAusIMM

Julian Broomfield, BEng (Mining), FAusIMM

Peter O'Bryan, BEng (Mining), MEngSc (RockEng), MAusIMM(CP)

Pieter Labuschagne, MSc (Hydrogeology), MIAH, Pr.Sci.Nat 400386/11

Stewart Watkins, BEng (Chem), FAusIMM

Predictive Discovery Limited
ABN 11 127 171 877
Suite 8, 110 Hay Street, Subiaco WA 6008
T +61 8 9216 1020

www.predictivediscovery.com

CONTENTS

LIST OF TABLES

LIST OF FIGURES

1 SUMMARY

1.1 Introduction, Location and Ownership

This Report was prepared for Predictive Discovery Limited (PDI or the Company) on the Bankan Gold Project, Guinea (the Project). This Report was prepared for the purposes of reporting on the definitive feasibility study (DFS) released to the Australian Stock Exchange on 25 June 2025 in accordance with the Joint Ore Reserves Committee (JORC) "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves" and CIM 2014 Definition Standards to align with the continuous disclosure of Exploration Results, Mineral Resources and Mineral Reserves in accordance NI 43-101.

The effective date of this Report is July 31, 2025.

The Project is located in the northeast part of Guinea, approximately 450 km east-northeast of Guinea's capital city, Conakry, in the Kouroussa Prefecture. The Project is located 75 km northwest of the regional city of Kankan and 7 km southwest of Kouroussa town.

The main Project area lies within the Peripheral Zone of the Upper Niger National Park with the NEB and BC deposits approximately 21 km and 18 km, respectively, away from the closest point of the Core Conservation Area.

The Project comprises four contiguous *Permis de Recherce Industrielle (Or)* (exploration permits), which cover a combined area of 356 km^2 and are located between 9 51'00"W and 10 03'24"W and between 10 32'26"N and 10 52'00"N.

PDI's four exploration permits relating to the Project and its wider exploration potential, comprise:

- Kaninko gold exploration permit, issued by order no. A/2019/5784/MMG in favour of PDI's wholly owned local subsidiary Mamou Resources SARLU (Mamou) on 3 October 2019, covering a 98.22 km^2 area.
- Saman gold exploration permit, issued by order no. A/2020/1835/MMG in favour of Mamou on 11 June 2020, covering a 99.78 km^2 area.
- Bokoro gold exploration permit, issued by order no. A/2020/2561/MMG in favour of PDI's wholly owned local subsidiary Kindia Resources SARLU on 9 September 2020, covering a 99.98 km^2 area.
- Argo gold exploration permit, issued by order no. A/2018/7628/MMG in favour of Argo Mining SARLU on 24 October 2018 (in which PDI is a shareholder and has the right to progressively earn 90% by payment of US$100,000 and acquire the remaining 10% at a decision to mine in exchange for a 2% net smelter royalty), covering a 57.54 km^2 area.

The main Project area, and all the Mineral Resources on which this DFS is based, are situated on parts of the Kaninko and Saman exploration permits.

On 31 January 2025, PDI and Mamou submitted exploitation permit applications for 50% of the Kaninko and Saman permit areas to the Ministry of Mines and Geology (MMG) and Centre for the Promotion of the Development of Mining (CPDM) in accordance with Guinean mining law. PDI has indicated that the applications are at an advanced stage and are still being processed. PDI is not aware of any immediate obstacles to the granting of the exploitation permits.

PDI submitted renewal applications for the Argo and Bokoro exploration permits in 2021 and 2023 respectively, and has relied on Article 78 of the Guinean Mining Code that allows for permits to be extended automatically until the date of renewal. PDI has been made aware that, on 26 May 2025, the MMG announced the revocation of over 100 exploration permits, including the Argo exploration permit (which hosts the Fouwagbe and Sounsoun Deposits) and the Bokoro exploration permit. PDI has not received any formal communication from the Guinean government on the matter and intends to work diligently with the MMG to achieve the granting of the renewals.

1.2 History

In late 2018 PDI commenced work in the Kaninko area in the Siguiri Basin. Field visits identified widespread artisanal workings consisting of extensive pitting into weathered bedrock with shallow surficial workings in lateritic cover material extending for hundreds of metres away from the pitted areas, in what were later to be identified as the NEB and BC deposits.

PDI's initial field work included BLEG stream sediment geochemistry, rock chip sampling and geological mapping followed by twelve vertical channel samples. This initial program was followed up by a second program of systematic channel sampling of saprolite exposures.

In early 2020, a program of 3,178 m of shallow power auger drilling and 490 lineal metres of trenching was completed at NEB and BC, with mineralisation identified across a broad zone This program was followed up by an aircore and reverse circulation drilling program, with further auger drilling extending the strike length of NEB.

A maiden mineral resource estimate was completed for the Project in September 2021 comprising an Inferred Mineral Resource of 72.8 Mt at 1.56 g/t Au for 3.65 Moz of contained gold. On the 1st of August 2022 additional drilling was used to update the Inferred Mineral Resource estimate to 79.5 Mt at 1.63 g/t Au for 4.2 Moz of contained gold.

Based on an infill drilling program through the second half of 2022 the mineral resource estimate confidence was improved and an updated mineral resource estimate was announced on 6 February 2023 including an Open Pit Indicated Mineral Resource of 42.7 Mt at 1.27 g/t Au for 1.75 Moz contained gold at NEB along with a further Open Pit Inferred Mineral Resource of 24.7 Mt at 2.23 g/t Au for 1.77 Moz contained gold and an Underground Inferred Mineral Resource at NEB of 2.2 Mt at 4.75 g/t for 335 koz contained gold. An Inferred Mineral Resource of 7.2 Mt at 1.42 g/t for 331 koz of contained gold was announced for the BC deposit.

Continued drilling at the Project led to an announcement on 7 August 2023 an increase to these mineral resources to an estimated 100.5 Mt at 1.66 g/t Au for 5.4 Moz of contained gold with approximately 77% being in the Indicated Mineral Resource category. Based on this mineral resource estimate, PDI completed a pre-feasibility study (PFS) which included the announcement of a maiden Mineral Reserve for the Project, consisting of open pit and underground ore from NEB and open pit ore from BC, of 57.7 Mt at 1.64 g/t for 3.05 Moz of contained gold. All Mineral Reserves were in the Probable Mineral Reserves category.

Other than artisanal scale gold mining, which is not material to the Mineral Resources or Mineral Reserves, there has been no production from the Property.

1.3 Geological Setting and Mineralisation

The Project is hosted by greenstones in the southwest margin of the Siguiri Basin, in upper Guinea. The Siguiri Basin contains metasediments and related volcanic and plutonic rocks of the early Proterozoic Birimian supergroup, which hosts most of West Africa's gold deposits. The gold deposits within the region are principally orogenic lode deposits. Prolonged weathering has led to extensive lateritic duricrusts and deep saprolite profiles. Vertical remobilisation of gold during lateritic weathering is common, and primary gold deposits are often overlain by lateritic or supergene gold deposits.

The Project area is deeply weathered, with a thick saprolite and a pisolitic and nodular lateritic cover which hosts remobilised gold, generally above the primary deposits or dispersed a few tens of metres laterally. Outcrops are sparse, and the underlying bedrock geology is known largely from regional scale geophysics and drilling completed by PDI.

Regionally, mineralisation has been focussed on the intersection of north-northwest striking and northwest striking structures on the margin of a regional granitic batholith. Numerous anastomosing north-northeast striking structures have been interpreted from the aeromagnetic data. Smaller granitic intrusions in the greenstones are structurally controlled and provide evidence for significant heat and fluid flow late in the orogenic history, likely to be part of the gold mineralisation process.

These granitic intrusions partially host the two Project main deposits. NEB has been developed at the hanging wall contact of a small tonalitic intrusion, structurally controlled by a north-northwest striking shear (main shear zone or STMZ), which is part of a network of anastomosing north-northwest to north-northeast striking structures. The NEB deposit includes a small satellite deposit, GBE, located approximately 250 m north of the main NEB deposit.

In the footwall, a very well developed second order shear, 3 m to 5 m thick, (STSZ01) has very similar structure and alteration characteristics to the STMZ and forms a step over, or jog, from the STMZ to a more weakly developed structure and hence it is a locus for dilation and fluid flow associated with mineralisation. The STSZ01 nearly outcrops, whereas the STMZ terminates below the surface above its intersection with STSZ01. This fault duplex is interpreted to represent a soft-linked overlapping shear system, where a component of strain is accommodated by rotation or folding between the main bounding shear segments, as well as at the termination of the segments.

Below the STSZ01 shear, four other parallel structures have been interpreted with similar relationships to the STMZ, however, these are less well constrained by drilling and, hence, have a greater degree of uncertainty in their location and extent.

Higher grades are found in and on the immediate footwall of the STMZ, with lower grade mineralisation in both the tonalitic footwall and the greenstone hanging wall. Mineralisation comprises wide zones of structurally controlled chlorite, silica and sericite alteration with associated pyrite and quartz veining.

Sulphide mineralisation largely comprises pyrite with minor chalcopyrite. In the altered felsic igneous rocks, the sulphide mineralisation is generally associated with the later stage veining, with minor amounts disseminated through the rock texture. In NEB, higher grade mineralisation is characterised by higher pyrite and covellite, and arsenopyrite and sphalerite contents. Low-grade mineralisation lacks covellite, galena, sphalerite, and bismuth species. Other sulphides that have been noted include tennantite-tetrahedrite, hessite, gersdorfitte, bornite and cobaltite. Generally sulphide content is low.

BC is hosted in the carapace of a small tonalitic intrusion, which has intruded a structurally complex greenstone sequence of clastic and carbonate metasediments, volcanics and marbles. The structural controls for BC are much less well understood. From the drillhole logging, two shears have been interpreted. A major one dipping moderately to the southwest and a second order structure dipping moderately to the northeast. These appear to constrain both the small tonalite intrusion and the mineralisation that is localised in the carapace of the intrusion. Foliations generally dip parallel to the major shear, whereas the veins have several preferred orientations and a greater scatter than the veins at NEB. Bedding planes and contacts broadly dip parallel to the foliations and shears.

The weathered profile in the Project area comprises:

- Cemented ferricrete layer, composed of in-situ or transported ferruginous concretions in a ferruginous matrix.
- Mottled clay layer, composed of variably ferruginous residual clays formed by intense weathering and consequent profile collapse.
- Saprolite zone, composed of highly weathered bedrock, where there has not been sufficient leaching to initiate the collapse of the profile, and original rock textures are recognisable even though most original rock forming minerals have been weathered to clays. There may be a transition or saprock zone at the base of the saprolite zone into the fresh zone, where weathering is either patchy or restricted to favourable structures. Levels greater than 40% fresh rock defines this saprock zone.
- Underlying essentially un-weathered fresh zone.

The complete laterite profile is preserved at NEB under a ridge capped with resistant ferricrete. At BC, recent erosion has incised the currently active river valley and the mottled zone and saprolite are largely exposed at the surface in the artisanal workings with a thin veneer of transported soil and alluvium elsewhere. A few small patches of remnant ferricrete have also been identified.

1.4 Exploration

Due to the deep weathering, transported cover and lack of outcropping rock, the most effective exploration methods have proved to be geophysical and geochemical vectoring, followed up by drill sampling.

Following the NEB discovery, PDI completed a series of early-stage exploration programs, including broad spaced auger drilling and a helicopter-borne magnetic and radiometric survey. The aeromagnetics identified a major 35km-long north-northwest structural corridor with the potential to host multiple orogenic gold discoveries. Structural targets identified using the aeromagnetics have been progressively followed up with power auger and aircore (AC) drilling. The strategy to date has been to undertake wide-spaced auger drilling covering the structural targets, typically 320 m by 80 m spacing, followed by closer spaced infill where encouraging gold results have been obtained (generally plus 0.25g/t composite values in saprolite to depths of around 20 m). AC drilling has then followed up the encouraging auger results, typically with pairs of scissor holes to help assess the orientation of the gold mineralisation.

These samples are useful for producing geochemical anomalies, however due to the open hole and non-representative sampling, are not used for resource estimation.

PDI has also completed a comprehensive petrophysics and ground geophysics program at NEB. The petrophysics study of the NEB drill core was designed to calibrate the detailed ground geophysical orientation program.

The ground geophysical techniques selected were gradient array induced polarisation ("GAIP") and pole dipole induced polarisation ("P-DIP"), magnetics and gravity with P-DIP methods proving the most effective in geophysically finger-printing the NEB deposit, with elevated chargeability (attributed to sulphide mineralisation) and elevated resistivity (attributed to silica alteration).

1.5 Drilling

Drilling completed at the Project comprises aircore (AC), reverse circulation (RC), reverse circulation grade control (RCGC) and diamond core (DDH) holes, with some deeper diamond holes having a RC pre-collar in expected waste and core thereafter. For the Mineral Resource Estimate for the NEB and BC deposits, announced in August 2023, only the DDH and RC holes were used as AC samples are not considered representative. Drillhole spacing is variable, typically 40 m spacing on 40 m sections in the upper parts of the deposits and spacings as much as 100 m at the lower fringes.

The total drilling incorporated into the Mineral Resource estimates for the NEB and BC deposits comprises:

- NEB Deposit:
  - 26,341 m RC across 209 holes.
  - 84,162 m DDH or RC pre-collar with DDH across 202 holes.
- BC Deposit:
  - 2,321 m RC across 20 holes.
  - 11,536 m DDH or RC pre-collar with DDH across 59 holes.

An additional 394 AC holes for 18,684 m which were used to supplement the RC and DDH holes for geological interpretation.

Drilling results after the cutoff dates for the mineral resource estimates have been reviewed but not yet modelled. This includes infill drilling at both NEB and BC. The results of the additional data are in line with the resource models and are not expected to significantly change the mineral resource estimates or classification.

1.6 Sample Preparation, Analysis and Security

Samples have been assayed by fire assay at a range of commercial laboratories in West Africa with most of the recent samples having been assayed at SGS in Bamako, Mali. PDI has implemented a quality assurance/quality control (QAQC) program for exploration and resource evaluation drilling and sampling at the Project, comprising monitoring of:

- Analytical data accuracy using certified reference materials (CRMs) and umpire laboratory assaying.

- Analytical data precision using field and laboratory duplicate and repeat samples.
- Potential for contamination during sample preparation using blanks.

No significant issues were noted with the CRMs, blanks, laboratory duplicates or umpire assaying. From the field duplicates, the precision of the sampling is reasonable, with the poorest precision in the core duplicate pairs, suggesting that there is a moderate to high fundamental nugget factor in the mineralisation.

Based on the data assessment, the Qualified Person considers the entire dataset acceptable for resource estimation subject to the preceding comments regarding the analytical accuracy and precision.

1.7 Data Verification

PDI has been developing the resources since 2019. The Qualified Person has visited the site on four occasions, from the 10th to the 15th June 2022, from the 10th to the 21st November 2022, from the 11th to the 27th January 2023 and from 28th August 2024 to the 5th September 2024. During these visits, the following were inspected:

- General site layouts.
- DDH, RC, AC and auger drilling.
- Drillhole setup.
- DDH core orientation and markup.
- DDH core logging and sampling.
- Density measurement procedure.
- Point Load Test measurement procedure.
- X-Ray Fluorescence measurement procedure.
- RC, RC and auger logging and sampling.
- Sample dispatch.
- DDH core and RC retention bag storage.
- Pulp storage.
- Review of selected core intervals and comparison with assaying results.

Detailed technical discussions with PDI staff were also conducted.

The Qualified Person has checked a selection of the original assay certificates against the database and not identified any errors.

The drilling, sampling, assaying, quality assurance, sample security and data handling procedures at the Project are well designed and are well implemented; they are capable of producing a reliable dataset that is fit for purpose for Mineral Resource estimation.

1.8 Mineral Processing and Metallurgical Testing

Metallurgical testwork for the Project has been completed across three programs:

- Program 1, preliminary testwork managed by Mintrex in 2021.
- Program 2, interim testwork managed by Independent Metallurgical Operations (IMO) in 2024.
- Program 3, testwork carried out as part of the DFS managed by Dhamana Consulting, with input from DRA, in 2024/25.

The process definition testwork was largely completed in the first two of these programs, where the work was focused on defining the key process design criteria, which included:

- Grind size, selecting a grind size of (P_{80}) 75 µm.
- Gravity gold recovery, which demonstrated that there is material gravity recoverable gold in the ore.
- CIL or carbon in pulp (CIP), which demonstrated that hybrid CIL is suitable due to minimal preg-robbing by the ore.
- Leaching residence time, demonstrating that industry typical residence time of 24 hours was suitable.
- Oxygen versus air for oxygen for the leaching, showing the use of air is comparable to oxygen in the cyanide leaching process, therefore the more cost-effective use of sparged air is justified.
- Leaching of the gravity tail with a range of cyanide concentration demonstrating that there is no loss of gold extractions when leaching commenced with an initial cyanide concentration of 500 ppm then allowing it to decrease to 120 ppm during the leach.
- Comminution testing to provide early estimates of ore hardness and grindability.

Program 3 used the earlier work that defined the general process flowsheet and conditions to complete a variability program and bulk testwork with the aims of better defining the range of process performance and design envelope, and to define carbon loading and cyanide destruction performance and design parameters, focussing on the sizing, or requirements for, the following aspects of the process:

- Pre-leach thickener sizing and performance across a range of feed blends.
- Tailings filtration performance across a range of feed blends.
- Paste-fill feed requirements and the requirement for desliming, also providing representative feed for the testing of paste properties.

Materials handling testwork was also completed in this program across the range of lithologies present in the ore body.

Across the three programs, 82 individual mineralised samples were selected to represent the range of lithologies, grades and deposits.

In addition to the mineralised samples, bulk samples of non-mineralised laterite and saprolite were used with residual mineralised fresh samples for testwork focused on the physical properties of the ore such as materials handling, thickening and filtration.

The sample suite was biased towards fresh ore (mafic, tonalite and shear) from the NEB open pit and underground resources as these represent the majority of the Mineral Resource to be mined.

1.8.1 Comminution

Saprolite samples were found to not be competent enough to enable selection of sub-samples that were suitable for standard comminution testwork. As such, the samples selected for comminution testwork were dominated by fresh lithologies (mafic, tonalite, shear) and two laterite samples. This was considered appropriate since the harder lithologies are the dominant consideration for comminution circuit design and sizing.

From a circuit design perspective, whilst the saprolite breakage characteristics are not deemed relevant, the observed “sticky” nature of the samples mean that the materials handling properties are a greater consideration for the comminution circuit design.

The various comminution programs consisted of SMC testing, Bond comminution testing and Geopyora testing. with a total of 27 samples contributing to the comminution dataset.

The comminution testwork results are summarised in Table 1.1.

Table 1.1: Comminution Testwork Summary

	Laterite		Fresh	
	Range	85th Percentile	Range	85th Percentile
Number of Samples	2		25	
Axb	140-171	144	17.9-121.8	23.2
BBWi (kWh/t)	10-11	10.8	9.9-25.5	23.6
BRWi (kWh/t)	-	13	21.4-26.3	24.4
Ai	-	0.4	0.25-0.50	0.37
SG	2.5-2.7	2.6	2.57-3.04	2.78

1.8.2 Leach Testwork

Gravity recovery followed by bottle roll leach tests in Program 1 and 2 defined many of the optimised leach conditions discussed in the introduction and applied in the variability testing in Program 3. The variability program combined gravity and leaching on 23 samples to define the geometallurgical relationships and design parameters for the processing facility.

Key relationships developed from the variability program include the following:

- Gravity recovery is variable but is expected to average 32%.
- Cyanide consumption in the leach is strongly correlated with the presence of soluble copper, but not lithology. The median cyanide consumption (0.40 kg/t) from the variability dataset was selected as the most appropriate value to represent the resource.

- Lime consumption is significantly different for fresh and weathered ore, and the average of the dataset is used to estimate leach lime consumption as 0.33 kg/t and 2.06 kg/t for fresh and weathered ore respectively.
- Gold recovery via gravity and leaching is most effectively estimated using a linear relationship against the gold head grade as per the relationship below:

Au Recovery (%) = 0.5145 x [Au head, g/t] + 91.533

1.8.3 Bulk Leach for Carbon Loading and Tailings Detoxification

Bulk leaching of composite samples was conducted to generate samples for carbon loading and tailings detoxification testwork.

Triple carbon contact tests achieved greater than 1,500 ppm gold loading on carbon and equilibrium loading carbon tests with varied carbon concentration achieved greater than 3,000 ppm gold loading on carbon.

The solution from the triple carbon contact tests was then detoxified successfully to less than 5 mg/L weak acid dissociable cyanide using typical conditions for cyanide destruction.:

1.8.4 Thickening and Filtration

Inspection of the samples as received suggested that saprolite was likely to be the most difficult component of an ore blend to dewater because of “sticky clay like” appearance. As such, composites were created ranging from 25% to 100% saprolite blended with fresh ore for dynamic thickening and filtration testwork, carried out by Metso.

The thickening testwork varied the solids loading from 0.25 to 1.5 t/m^2h and acceptable performance (greater than 55% solids) was achieved with less than 75% saprolite in the blend at 1.0 t/m^2h.

Pressure filtration achieved acceptable product cake consistency on all samples from a 40mm chamber, however the moisture in the cake was significantly improved to less than16% moisture for the samples with less than 50% saprolite in the blend. Notably, as the percentage of saprolite reduced to the 25%, the lower moisture was achieved at an improved rate.

1.8.5 Other Testwork

Materials handling and rheology testwork was specifically completed to assess the risk the saprolite ore may pose throughout the dry part of the process (ore handling, crushing, crushed ore storage and transfer points) and as a slurry where increased densities are desired (thickening, leaching).

The materials handling testwork concluded that fresh and fresh/laterite blends can be handled together in the dry part of the circuit with typical designs for transfer points and stockpiles. However, the saprolite ore warrants a separate circuit that minimises the potential for blockages that may impact operations. Saprolitic ores require moisture control, minimised drop energies, chute angles of greater than 74 degrees from horizontal and low friction resistance liner materials (e.g. Matrox) to contend with difficult materials handling properties of saprolite ores.

Rheology testing concluded that blends with less than 75% saprolite can operate at densities of approximately 50% solids in the leach without specific design to accommodate slurry rheology.

In addition, a bulk sample of simulated tailings (ground to a P_{80} of 75µm but not leached) was subjected to paste testwork. The testwork demonstrated that partially deslimed paste was cohesive and homogenous and the rheology is not overly sensitive to paste solids content.

Although relatively poor strengths were achieved with general purpose (GP) cement paste mixes for the blends tested (up to 12% cement), the strengths measured are suitable for all but the horizontal exposure application which is anticipated to require as much as 20% GP cement to achieve acceptable strength.

1.9 Mineral Resource Estimate

The conceptual geological model for NEB is a set of anastomosing shear structures in a greenstone sequence focused at the hanging wall contact of a felsic intrusion into an older mafic/metasedimentary sequence. Leapfrog Geo software was used to create models of these lithologies to fill the resource model space. The weathering surfaces were modelled as digital terrain model surfaces and other lithologies were modelled as solid wireframes. This lithological information was then coded into the resource block model.

Leapfrog grade shells were produced using downhole composite assay files as domains for the resource estimates. Smoothing parameters were chosen in an iterative process after reviewing preliminary shells to establish appropriate mineralisation continuity criteria.

For NEB, three nested grade domains were defined in the saprolite and fresh mineralisation using Leapfrog software, at nominal 2 g/t (high-grade), 0.4 g/t (medium-grade), 0.3 g/t (GBE) and 0.2 g/t (low-grade) cut-offs from 3 m downhole composites. For the laterite mineralisation, a 0.5 g/t cut-off domain was defined from 1 m downhole composites. A similar process was followed at BC.

The downhole composite files were intersected with the final domain wireframes to create the resource estimation dataset. High-grade cuts were applied to composites to reduce the influence of extreme outliers. These values are determined by statistical analysis, including a review of coefficient of variation (CV) values, histograms, log-probability plots, and mean-variance plots. The aim of choosing top-cuts was to reduce the CV without affecting the overall mean grade of the various mineralised domains.

Experimental variograms were produced from the mineralised domain composite datasets. For all domains, a normal scores transformation was applied to remove short-scale statistical noise and help model the underlying variability. After modelling variograms, the results were back-transformed into sample space, and the final variogram models were used for grade estimation. In general, the variograms are moderately well structured, with moderate to high nuggets and short ranges.

Gold grades were estimated into the flagged domain blocks using ordinary kriging. The kriging estimation parameters were chosen from the kriging neighbourhood analysis.

The Mineral Resource was classified as Indicated and Inferred based on the level of geological understanding of the mineralisation, quality of samples, mineralisation continuity evident between drillholes and drillhole spacing.

To constrain the resource models for reporting, open pit optimisations were completed using a gold price of US$1,800/oz and otherwise largely generic optimisation inputs. Cost inputs were based on similar scale projects, metallurgical recoveries are based on the preliminary testwork, and pit slopes based on analogous open pit operations. The NEB and BC optimised pits have surficial dimensions of 1600 m by 900 m and 500 m by 400 m, respectively, and are located approximately 2.5 km apart. The entire high-grade domain interpreted below the optimal pit shell for the NEB underground resource is reported.

In April 2025, PDI announced Mineral Resource estimates for the Fouwagbe and Sounsoun deposits, both of which are situated north of the main Project area on the Argo permit. These Mineral Resource estimates are reported at a 0.5g/t Au cut-off, within a preliminary open pit optimisation assuming a gold price of US$2,300/oz and otherwise generic optimisation inputs. Four composites of saprolite ore were leach tested, two from Sounsoun and two from Fouwagbe. All four had good gravity recovery, and cyanidation extractions at 24 hours were between 88% and 99%. These results suggest that relatively high recoveries may be achievable using standard CIL technology. The deposits are also within trucking distance of the Project's proposed processing plant.

The Mineral Resource estimates for the Project are tabulated in Table 1.2 and Table 1.3. The Mineral Resource estimates have been classified in accordance with the JORC Code (2012) and CIM 2014 Definition Standards and have effective dates of 7 August 2023 and 23 April 2025 respectively. The Mineral Resource Statements are global estimates of in-situ tonnes and grade. It is suitable for reporting as a global resource, however, the relatively wide sampling grid have produced models with only moderately well estimated individual blocks. No reliance should be placed on individual block grade estimates, and additional close-spaced drilling will be required to enable detailed open pit and underground detailed production planning.

At the time of the Technical Report, the Qualified Persons are not aware of any environmental, legal, socio-economic, marketing or other relevant conditions, that would materially affect the estimated Mineral Resources of the Bankan Gold Project.

Table 1.2: Mineral Resource Estimate (NEB and BC)

Deposit	Type	Classification	Cut-off g/t	Tonnes Mt	Grade Au g/t	Contained Metal koz Au
NEB	Open Pit	Indicated	0.5	78.4	1.55	3,900
		Inferred	0.5	3.1	0.91	92
	Underground	Inferred	2.0	6.8	4.07	896
BC	Open Pit	Indicated	0.4	5.3	1.42	244
		Inferred	0.4	6.9	1.09	243
Total Indicated				**83.7**	**1.54**	**4,144**
Total Inferred				**16.8**	**2.28**	**1,231**

Notes:

1. The Mineral Resources is estimated with all drilling data available on 29th July 2023 (NEB and BC).
2. The Mineral Resource is reported in accordance with the JORC Code 2012 Edition and CIM 2014 Definition Standards and has an effective date of 7 August 2023.
3. The Mineral Resources (NEB and BC) are inclusive of Mineral Reserves.
4. The Qualified Person is Phil Jankowski FAusIMM of ERM.
5. The open pit Mineral Resources are constrained by optimised pit shells using a metal price of US$1,800 per ounce Au (NEB and BC) and process recovery of 94%. The NEB underground Mineral Resource is constrained by the high-grade domain below the NEB optimised pit shell.
6. The NEB open pit Mineral Resource is reported using a notional maximal open pit option. Current reserve planning is for a smaller open pit and an accelerated underground operation, where a higher grade (>2g/t) part of the open pit Mineral Resource is mined from underground. If this option is chosen, some of the lower grade open pit Mineral Resource may be effectively sterilised.
7. Rounding may lead to minor apparent discrepancies.

Table 1.3: Mineral Resource Estimate (Fouwagbe and Sounsoun)

Deposit	Type	Classification	Cut-off g/t	Tonnes Mt	Grade Au g/t	Contained Metal koz Au
Fouwagbe	Open Pit	Inferred	0.5	2.2	1.68	119
Sounsoun	Open Pit	Inferred	0.5	0.9	1.19	34
Total Inferred				**3.1**	**1.54**	**153**

Notes:

1. The Mineral Resources is estimated with all drilling data available at 25th of February 2025 (Fouwagbe and Sounsoun).
2. The Mineral Resource is reported in accordance with the JORC Code 2012 Edition and CIM 2014 Definition Standards and has an effective date of 23 April 2025.
3. The Qualified Person is Phil Jankowski FAusIMM of ERM.
4. The open pit Mineral Resources are constrained by optimised pit shells using a metal price of US$2,300 per ounce Au (Fouwagbe and Sounsoun) and process recovery of 94%.
5. Rounding may lead to minor apparent discrepancies.
6. The Mineral Resources reported at Fouwagbe and Sounsoun are not considered further in the DFS.
7. PDI has been made aware that, on 26 May 2025, Guinea's MMG announced the revocation of over 100 exploration permits, including the Argo (which hosts the Fouwagbe and Sounsoun deposits) and Bokoro exploration permits held by PDI group companies. The applications for extension of these permits were submitted to the MMG in 2021 and 2023, respectively. PDI has not received any formal communication from the Guinean government on the matter and intends to work diligently with the MMG to achieve the granting of the renewals.

1.10 Mineral Reserve Estimate

The Mineral Reserve developed by Orelogy as part of the DFS was announced by PDI on 25 June 2025 (Predictive Discovery Limited, 2025). PDI and Orelogy confirm that they are not aware of any new information or data that materially affects the information included in the corresponding market announcement, and that all material assumptions and technical parameters underpinning the Mineral Reserve estimates in the market announcement continue to apply and have not materially changed.

The Mineral Reserve was estimated from the Mineral Resource after consideration of the level of confidence in the Mineral Resource and taking account of material and relevant modifying factors including mining, processing, infrastructure, environmental, legal, social and commercial factors available at the time. The Probable Mineral Reserve estimate is based on Indicated Mineral Resources only. No Inferred Mineral Resource was included in the Mineral Reserve. The Mineral Reserve represents the economically mineable part of the Indicated Mineral Resources. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

It should be noted that the Mineral Resource on which the Mineral Reserve is based was constrained by a large optimisation shell to determine a reasonable prospect for eventual economic extraction (RPEEE). However, the final NEB pit design generated for the Open Pit Mineral Reserve was considerably smaller as it was based on an open pit / underground trade-off optimisation study.

Consequently, the portion of Open Pit Indicated Resource below this smaller pit shell was considered for the Underground Mineral Reserve.

The total Probable Mineral Reserve is estimated at 51.6 Mt at 1.78 g/t Au with a contained gold content of 2,953 koz.

The Mineral Reserve for the Project is reported according to the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (Joint Ore Reserves Committee, 2012) and CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM Standing Committee on Reserve Definitions, 2014) and has an effective date of 31 July 2025. The Mineral Resource was converted by applying modifying factors. The Probable Mineral Reserve estimate is based on the Mineral Resource classified as Indicated. Table 1.4 presents a summary of the Mineral Reserves on a 100% Project basis at a US$1,800/oz gold price.

Table 1.4: Bankan Gold Project Mineral Reserve

Deposit	Mining Method	Classification	Tonnes (Mt)	Grade (g/t Au)	Contained Metal (koz Au)
NEB	Open Pit	Probable	40.2	1.36	1,751
	Underground	Probable	7.9	3.95	1,002
	Total		48.1	1.78	2,753
BC	Open Pit	Probable	3.5	1.78	200
Total Open Pit		Probable	43.7	1.39	1,951
Total Underground		Probable	7.9	3.95	1,002
Total Project		**Probable**	**51.6**	**1.78**	**2,953**

Notes:

1. The Mineral Reserve conforms with and uses the JORC Code (2012) and CIM (2014) definitions and has an effective date of 31 July 2025.
2. The Mineral Resources (NEB and BC) are inclusive of Mineral Reserves.
3. The Mineral Reserve was evaluated using a gold price of US$ 1,800 per ounce.
4. The Mineral Reserve was evaluated using variable cut-off grades as described in Sections 15.6.4 and 0.
5. Ore block grade and tonnage dilution was incorporated into the model.
6. All figures are rounded to reflect appropriate levels of confidence.
7. Apparent differences may occur due to rounding.
8. The Qualified Person responsible for the Open Pit component of the Mineral Reserve is Mr Ross Cheyne, Principal Consultant with Orelogy Consulting Pty Ltd. Mr Cheyne visited the site in January 2025.
9. The Qualified Person responsible for the Underground component of the Mineral Reserve is Mr Julian Broomfield, Principal Consultant with Orelogy Consulting Pty Ltd.

At the time of the Technical Report, the Qualified Persons are not aware of any environmental, legal, socio-economic, marketing or other relevant conditions, that would materially affect the estimated Mineral Reserve of the Bankan Gold Project.

The Qualified Persons are of the opinion that the proposed mine plan is technically achievable and that all technical proposals made for the operational phase involve the application of conventional technology that is widely utilized in the gold industry in West Africa. In addition, financial modelling completed as part of the DFS shows that the Project is economically viable under current assumptions and all material modifying factors (mining, processing, infrastructure, environmental, legal, social and commercial) were considered during the Mineral Reserve estimation process.

The Mineral Resources, geotechnical investigation and assumptions, hydrogeological investigation and assumption and optimisation processes that underpin the estimation of the Mineral Reserves are outlined in this Technical Report.

1.11 Mining Methods

The mining for the Projects comprises open pit mining NEB, BC and GBE along with underground mining of the lower portions of the NEB orebody.

The GBE pit is be mined during the first nine months of the two-year pre-production period. This pit is specifically designed to target the fresh rock interface adjacent to the NEB orebody, from where the underground access portal will be established.

Underground development then progresses from Month 11 of Year 1 of pre-production and pre-stripping of NEB begins in the second half of Year 2 of pre-production to ensure sustainable ore feed is available from the start of production.

Mining of BC is deferred until the end of the mine life due to the added cost and complexity of establishing surface water management, which includes the diversion of Bankan Creek.

1.11.1 Open Pit Mining

The selected open pit mining method is a conventional truck and shovel approach, which is a proven mining method for open pit mining in West Africa.

Mining areas will be cleared of vegetation and topsoil removed and stockpiled for later use in rehabilitation of the site. Topsoil ranges in depth from insignificant on the top of the hills to up to 1 m in the valley floors and therefore an average depth of 0.3 m has been assumed.

Grade control will be carried out in advance of mining utilising RC drilling. An assessment has been completed by the geological Qualified Person for the drilling density and recommended a pattern of 10 m by 7 m at 20 m vertical intervals. The mining method and grade control practises to be employed at site are aimed at mining the ore zones selectively using backhoe configured excavators on a 2.5 m flitch to minimise dilution and ore loss.

It has been assumed that all material will require drill and blast, except for the saprolite clay and mottled clay material which has been assumed as 75% and 50% free dig respectively.

The primary production fleet proposed by the selected mining contractor is provided in Table 1.5. The numbers are shown at steady state mining.

Table 1.5: Proposed Open Pit Mining Fleet

Category	Type	Model	Specification		Number
			Unit	Value	
Loading	Excavator	Caterpillar 6015	operating weight (t)	140	3
	Excavator	Caterpillar 395	operating weight (t)	94	1
	Excavator	Caterpillar 374	operating weight (t)	74	1
	Excavator	Caterpillar 335 (w/ hammer)	operating weight (t)	35	1
	FEL	Caterpillar 988	bucket capacity (m^3)	6.5	2
Drilling	Drill	EPIROC D65	hole dia.(mm)	110-229 mm	3
Hauling	Dump Truck	Caterpillar 777	capacity (m^3)	60	13
Support	Track Dozer	Caterpillar D9	power (kW)	357	6
	Wheel Dozer	Caterpillar 834K	power (kW)	419	1
	Motor Grader	Caterpillar 16	blade length (m)	4.9	2
	Water Truck	Caterpillar 777	tank size (kl)	75	2
	Water Truck	MAN 6x6	tank size (kl)	20	2
	Roller/Compactor	Caterpillar CS78	weight (t)	18	1
	Rockbreaker	Furukawa FXJ375/ Caterpillar 335	weight (t)	2.6	1
U/G Rehandle	FEL	Caterpillar 992	bucket capacity (m^3)	105	1
	Truck	Caterpillar 777	capacity (m^3)	90	3
Total					**43**

Open pit mining costs were developed through a budget pricing enquiry that was sent to four selected contractors with experience on mining operations of similar scale to the Project in West Africa. The contractors were provided with preliminary mining schedules, waste rock dump locations, estimated haulage profiles, rehandle requirements and details of other requirements under the contract. Three of the contractors provided submissions that were withing ±5% with one of the contractors selected as the preferred submission. This single contractor was then requested to update their submission on the basis of the final designs, production schedules and haulage profiles.

1.11.2 Underground Mining

The underground mining philosophy for the Project is designed to maximise resource recovery, optimise operational efficiency, and align with processing constraints, while effectively managing geotechnical risks. Given the high-grade nature of the orebody, the selected method combines transverse and longitudinal long hole open stoping with engineered paste fill to enable large-scale stope extraction without leaving behind stabilising pillars. A bottom-up mining sequence has been adopted to support this approach, as it allows for larger stope designs and reduces the number of required stope cycles, thereby improving productivity and lowering unit costs. The use of paste fill plays a critical role in achieving this strategy, serving as structural support to enable the maximum stope recovery while minimising ground stability risks. Careful attention has been given to sequencing to limit dilution from exposure to paste-filled voids.

The underground operation will adopt a conventional mechanised mining approach, in line with industry best practices. Lateral development will be carried out using twin-boom jumbo rigs, which will also install in cycle ground support. In areas requiring increased ground control, such as sill pillar drives, the application of fibrecrete will provide additional reinforcement.

Vertical development will be executed using a combination of raise boring equipment for larger diameter infrastructure (e.g., return air raises) and long hole drilling techniques for smaller openings such as slot raises, secondary ventilation, escapeways and service holes.

Once footwall access drives are completed, grade control drilling using RC rigs will be undertaken to define ore boundaries and support detailed stope design. Stopes will be designed to maximise recovery while controlling dilution and will be drilled using long hole production rigs. Blasting activities will utilise emulsion explosives and electronic detonators to ensure precise rock fragmentation and controlled breakage. Broken ore will be bogged by load-haul-dump (LHD) units and stored in level stockpiles before being transferred to underground trucks and hauled to the surface stockpile. From there, surface haulage trucks will deliver the material to the ROM pad for processing.

A ventilation study was undertaken for the underground mining operation based on Western Australia Work Health and Safety (Mines) Regulations 2022 allowing for the equipment in use underground, fuel usage, working area minimum airflow requirements and for leakage. Main peak ventilation requirements are 435 m^3/s which is serviced by four surface connections at the main portal, a ventilation drive (FAD/RAD), a fresh air raise (FAR) dedicated to mine cooling, and a main return air raise (RAR), which will serve as the primary exhaust pathway for the underground mine. Secondary ventilation will be by means of auxiliary fans installed in strategic locations connected with vent bags and auxiliary fans installed in the FADs to deliver fresh air into levels

As the underground mine progresses deeper, the heat load from equipment and rock will increase and auxiliary cooling will be required. Three 1.5 MWBAC modular cooling units will be progressively installed to cool air drawn into the mine. Modelling has shown that the auxiliary cooling will only be require approximately six months of the year when surface temperatures are at their hottest.

Paste fill for the underground operations will be prepared from filtered, deslimed plant tailings and cement binder. Individual stopes will have a tailored paste blend to meet the particular strength requirements determined primarily on whether a stope is undercut or non-undercut. Paste will be delivered from the paste plant, located between the GBE and NEB open-pits, via paste fill lines to the stopes.

1.11.3 Mine and Production Schedules

The mining schedule was developed with a philosophy of maximising high-grade material from the underground and then meeting the milling capacity requirements from the open pit mines. In addition, blending of the various lithologies to a minimum fresh material percentage and a maximum saprolite material percentage was applied to meet processing requirements.

The overall integrated Project LOM production schedule, based on Mineral Reserves only, is presented by source in Figure 1.1. The process plant feed is presented by lithology in Figure 1.2 and by source in Figure 1.3 in terms of contained gold.

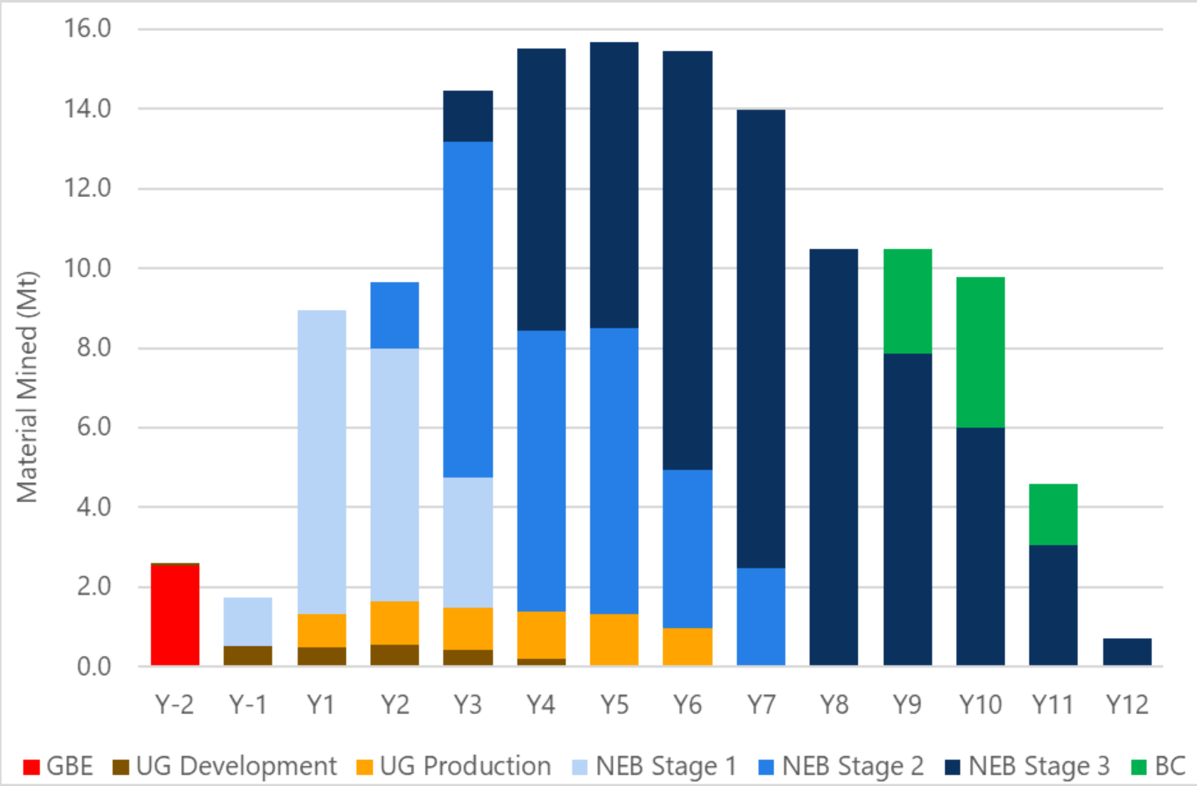

Figure 1.1: Project LOM Schedule – Total Material Mined

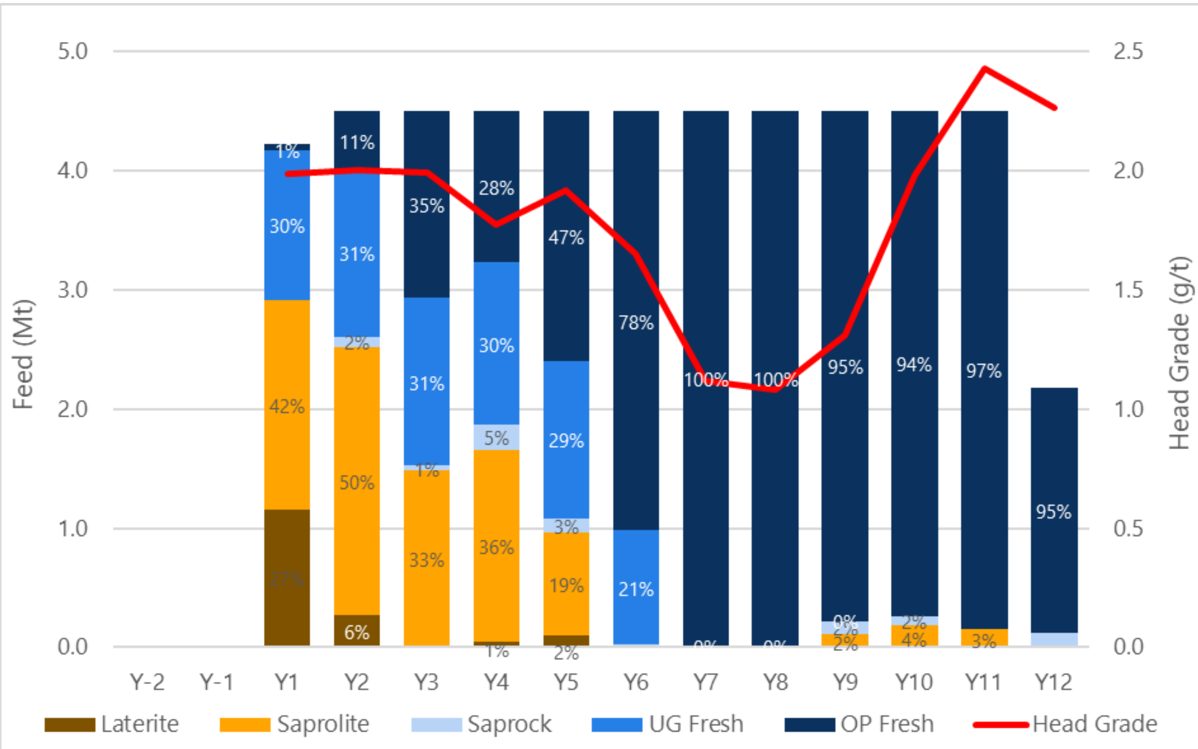

Figure 1.2: Project LOM Schedule – Mill Feed by Lithology and Grade

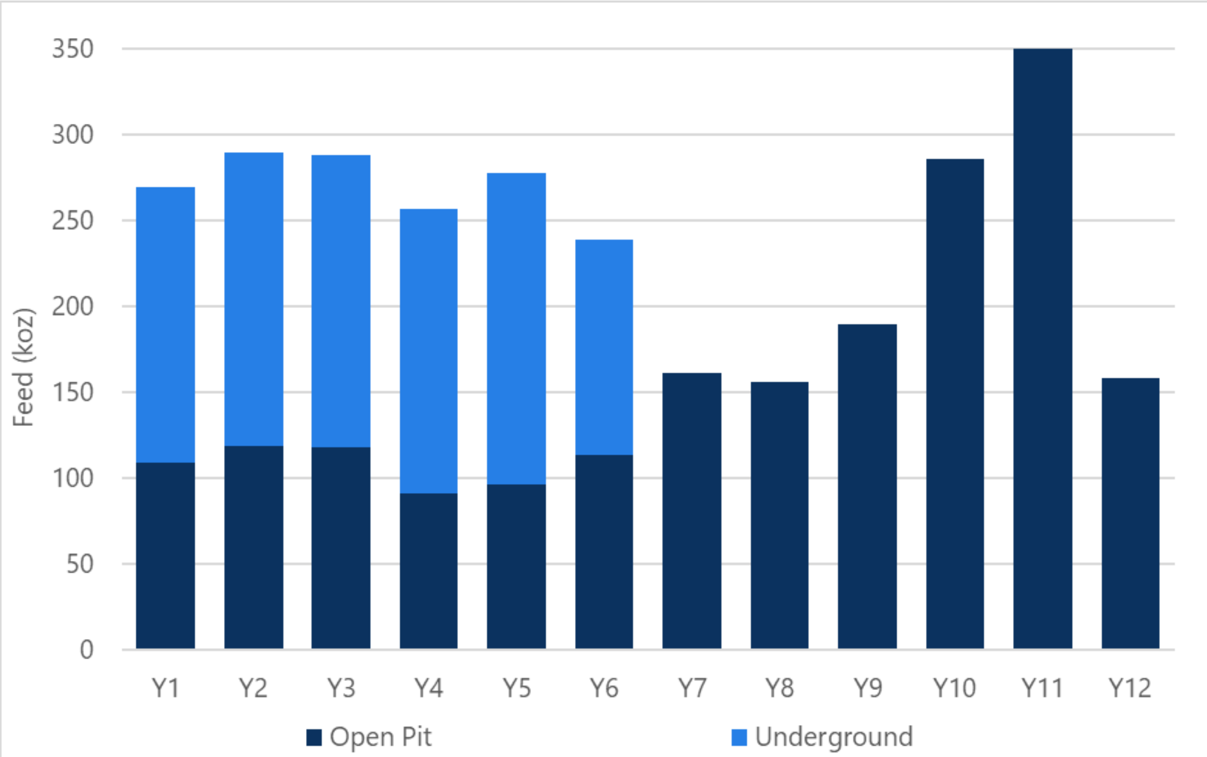

Figure 1.3: Project LOM Schedule – Mill Feed Contained Gold by Source

1.12 Recovery Methods

The selected process flowsheet will include primary crushing of fresh (hard) ore and soft (weathered) ore in separate circuits using a primary jaw crusher and mineral sizer respectively. Crushed fresh ore will report to a crushed ore stockpile, while crushed weathered ore will report directly to SAG mill feed.

The grinding circuit will consist of a SAG mill with a pebble crusher and a ball mill operating in closed circuit with the grinding circuit cyclones.

A leach feed thickener will be included in the flowsheet to provide a consistent thickened feed to the hybrid leach-CIL train. Loaded carbon will be eluted using the Zadra method, with simultaneous electrowinning of gold and silver from the eluate solution. Precious metal sludge will be recovered from the electrowinning cells and cathodes, then filtered and dried. Dry precious metal sludge will be mixed with fluxes and smelted to produce doré bullion bars.

CIL tailings will be detoxified for cyanide destruction using sodium meta-bisulphite and air prior to being filtered using pressure filtration to produce filter cake for disposal in the TSF. A portion of the tailings will be deslimed in cyclones for separate filtration to produce feed for the paste backfill plant.

Key design parameters for the processing plant include:

- Throughput, 4.5 Mtpa.
- Comminution parameters:
  - Fresh ore, Axb, 23.2, Bond ball mill work index 23.6 kWh/t.
  - Laterite and saprock ore, Axb, 144, Bond ball mill work index 10.8 kWh/t.

– Saprolite ore, Axb, 150, Bond ball mill work index 3.0 kWh/t.

- Primary grid size, 75 µm.
- Leach and adsorption, 1 leach tank, 6 CIL tanks for a nominal 24 hour residence time.
- Elution column capacity, 10 t carbon.

Figure 1.4 provides the gold production by year. Production average 251 koz/a over the first six years of production before dropping to an average of 156 koz/a for the following three years before a peaking at an average of 295 koz/a of the last two years of full production (Year 10 and 11).

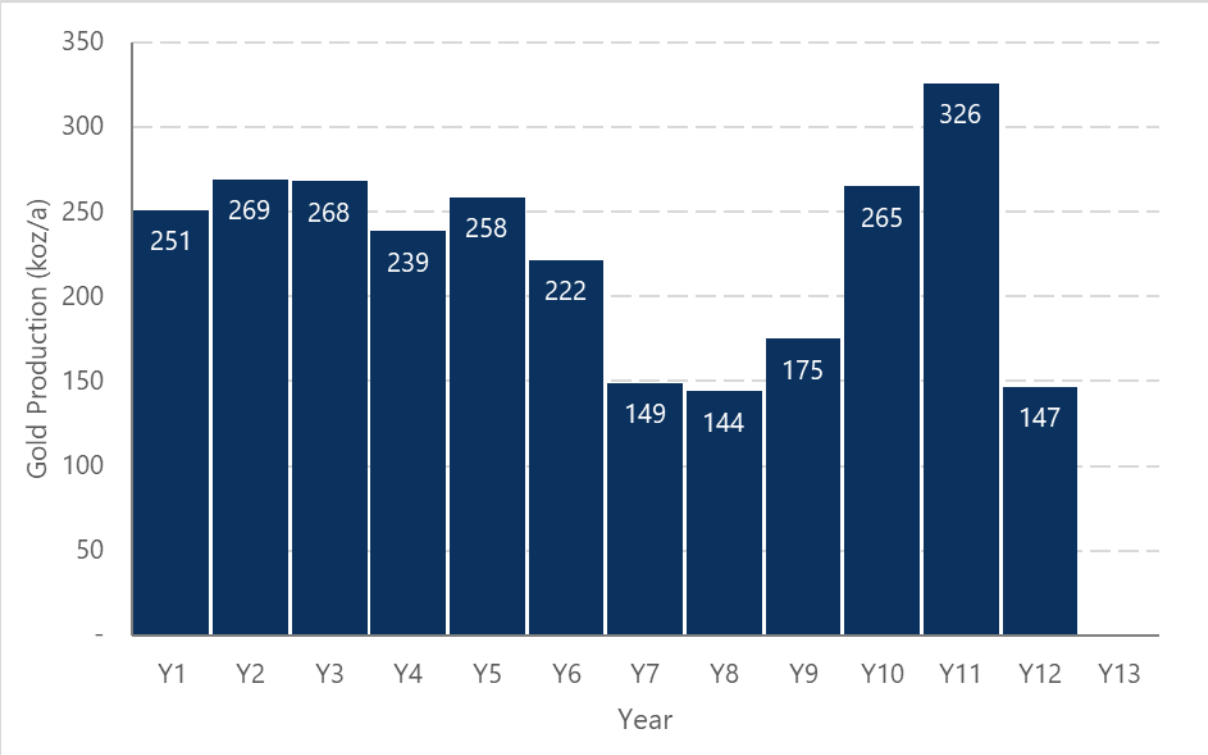

Figure 1.4: Project LOM Schedule – Mill Feed Recovered Gold

1.13 Project Infrastructure

A range of non-process infrastructure (NPI) will be required to enable operations at the Project. This will include the following:

- Access roads to the site from the N2 highway running to Kouroussa.
- Offices, warehouses, workshops and other buildings generally located at the process plant site.
- Accommodation village for expatriate and senior staff with the majority of the Project workforce living residentially in Kouroussa and the surrounding villages.
- Filtered tailings storage facility and TSF water storage dam.

- Power supply from a 32.5 MW hybrid heavy fuel oil and solar photovoltaic array to satisfy the average 27.7 MW load and site power distribution to remote loads and the underground mine.
- Water supply and management systems to manage water from dewatering bores, surface water run-off collection and the TSF.
- Mining infrastructure with the facilities other than the fuel storage and washdown faicilities provided by the mining contractors.
- Fuel storage and supply.
- Paste plant.
- Other infrastructure.

An overall site layout is provided in Figure 1.5, which shows the supporting infrastructure and services, mining locations, and processing plant infrastructure.

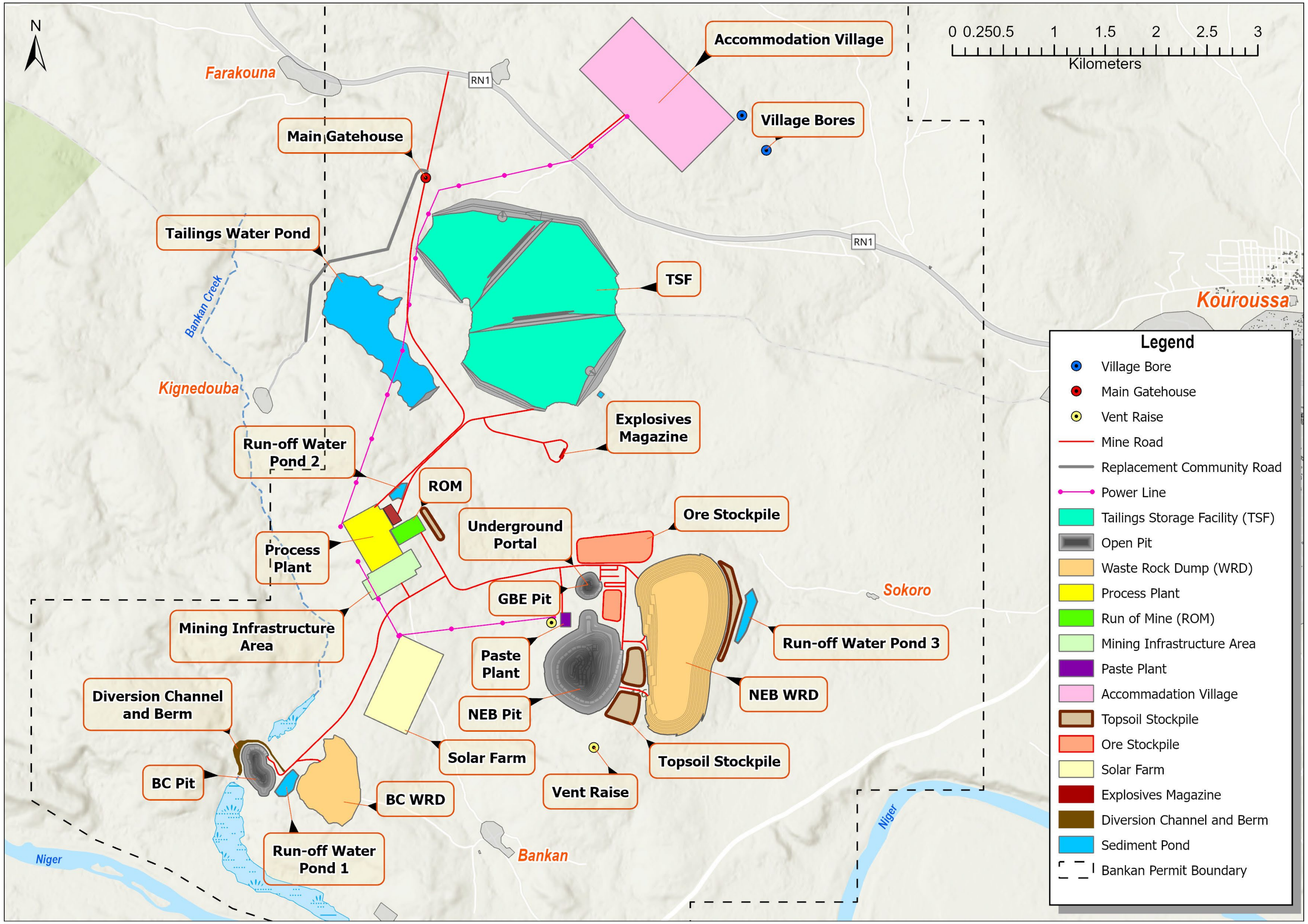

Figure 1.5: Overall Site Layout

1.14 Market Studies and Contracts

PDI intends to enter into a contract for refining doré produced at Project. Under the intended contract, the refiner will arrange transport for the doré bars to its refinery with transfer of custody for the product taking place in the secure gold room located within the processing plant facility. Charges for the transportation, insurance and refining of US$8.45 per ounce of gold have been included in the financial modelling.

As gold bullion is freely traded at prices that are widely known, a market study for the sale of gold doré. Given the freely traded nature of gold, the prospects for sale of any production are virtually assured.

The historical gold price over the last four years has shown relatively stable prices from June 2021 to February 2024 with an average of US$1,870/oz, after which the gold price has increased relatively steadily to current prices, which have averaged US$3,284/oz over the month of May 2025.

In order to forecast the long-term future gold price a consensus report was obtained from BMO Capital Markets (Chen, 2025). This forecast includes 34 individual forecasts from various sources.

The median of the long-term forecasts is US$2,400/oz which was used as a base case for the financial evaluation of the Project.

1.15 Environmental Studies, Permitting and Social or Community Impact

An ESIA, including an environmental and social management and monitoring plan (ESMMP), was developed for the Project which aligns with the national laws and regulations, and international guidelines and standards. The ESIA, including baseline studies and identifying the potential risks and impacts which may occur due to the Project, was completed in April 2024 by Environmental Resource Management (ERM), supported by local consultants Biotope and Insuco Limited. The Ministry of Environmental and Sustainable Development (MEDD) approved the ESIA and issued the ECC (CCE/00070) for the Project on 17 January 2025.

The existing physical, biological and social conditions of the area were assessed and classified as an area of influence from the Projects infrastructure, focusing on the resource to receptors that may be impacted by the Project.

The following environmental and social studies were undertaken between 2022 and 2024 to determine the existing baseline conditions:

- Ambient air quality.
- Noise.
- Surface water.
- Groundwater.
- Soils, soil quality and geology.
- Biodiversity and ecosystem services.
- Socio-economic.
- Cultural heritage.

- Landscape and visual.
- Traffic and transport.
- Human rights assessment.

The assessment of the potential environmental and social risks and impacts attributable to the phases of the Project included qualitative and quantitative (where relevant) assessments. The significance of each potential impact was identified, and mitigation measures to minimise and reduce the impacts were recommended. Cumulative impacts, particularly on communities' health and safety and on biodiversity, were also assessed.

The risks and impacts were quantified and classified as having a negligible, minor, moderate or major significance. The complete impact assessment is available in the ESIA.

A range of environmental and social management plans have been developed for the Project across the phases of development, operations and closure. The management plans incorporate measures and procedures for the short-term and long-term health and safety, environmental and social management. The management and monitoring plans were developed as a tool to be used by the Project throughout construction, operation and closure and rehabilitation. An integrated environmental and social management system (ESMS) is being developed as part of the Project to guide design and manage construction, operation and the closure and rehabilitation phases.

The environmental and social management plans developed for the Project include environmental - incorporating the management and monitoring requirements across a range of environmental aspects of the Project; socio-economic - incorporating the management and monitoring requirements across a range of social aspects of the Project; and health and safety - incorporating the management of the health and safety of workers and the surrounding communities.

These management plans all detail the risks or impacts that are to be managed, management measures that are to be implemented through the phases of the Project and monitoring measures to determine the effectiveness of the management measures. Where possible, the plans incorporate trigger-action-response plans to clearly outline requirements.

The Project's ESIA contains a conceptual plan for the rehabilitation, decommissioning and closure of the Project. The overriding intent of mine closure and rehabilitation is to return the land as close as is reasonably practical to its pre-disturbance condition, being for pastoral and agricultural activities. This will be achieved through establishment of a safe and stable post-mining land surface which supports vegetation growth and is erosion resistant over the long-term. Closure and rehabilitation activities will be undertaken during operations (progressive closure) or after operation (final closure).

The closure and rehabilitation plan includes procedures if the mine experiences temporary or sudden closure. The plan also includes maintenance and monitoring post-closure covering physical aspects of the Project such as landforms (geotechnical stability), water and biodiversity. The objective of long-term monitoring to validate compliance with closure success criteria, which includes the safety of landforms, maintaining appropriate water quantity and quality, and establishing self-sustaining ecosystems and habitats.

Based on the mine closure plan, a closure cost estimate was generated from first principles with the closure cost estimated at US$36.5m, excluding mine WRD reprofiling which is carried out as part of the mining costs. In addition, an estimate has been made for the post-closure monitoring of US$3.2m that will take place over a period of up to five years post closure.

The Project requires the acquisition of approximately 2,000 ha of land for the establishment of infrastructure. Consequently, this will result in economic displacement of the landowners and occupants, leading to the loss of livelihoods. Land that is being used for agriculture, livestock (grazing), artisanal scale mining (ASM) and areas used for ecosystem services will be acquired by the Project. Additionally, the establishment of the Project will result in a change of access to land and areas which have not been acquired.

The Project has established a Resettlement and Compensation Policy Framework (RCPF) which will serve as the basis for the development of the economic resettlement action plan and the LRP. The RCPF follows the national legislation and standards as well as international standards.

The economic resettlement action plan and LRP will aim to integrate all aspects of the planned economic resettlement related to the Project and the concurrent livelihood restoration activities into the ESMMP and, ultimately, into the Project's ESMS. This plan will serve as the basis for defining and implementing operational compliance management procedures and practices in all Project activities undertaken by the Company, its contractors, and suppliers. All aspects of this plan will be integrated into the Company's activities during the construction, operation, and closure phases of the Project.

The final planning stages for the implementation of the economic resettlement action plan and LRP are currently underway with engagement anticipated to commence following grant of the Project's exploitation permit.

1.16 Capital and Operating Costs

1.16.1 Capital Costs

Capital costs for Project have been developed based on the designs developed and described in this report. In addition, the costs have been developed in line with the implementation methodology and contracting strategy outlined. The estimate has generally been compiled based on the mechanical equipment list, specifications and material take offs (MTOs) produced for the DFS. The capital cost estimate for the Project has been developed to be generally consistent with the requirements of an AACE Class 3 estimate with an accuracy of ±15%.

In addition, operational costs incurred prior to the commencement of ore processing, particularly the underground mine development and other pre-production mining, have been included in the capital cost estimate, however, have been estimated in the same manner as operating costs in those areas.

As summarised in Table 1.6 the total pre-production costs from the commencement of execution readiness (following completion of the DFS) through to the first processing of ore, which will be shortly followed by first production of gold have been estimated at US$463.0m, including a contingency of US$34.3m. The estimate includes all the infrastructure and services required to operate the Project, pre-production mining, project management, first fills and spares and owner's costs.

Table 1.6: Capital Cost Estimate

Area	US$m[1]
Mining	
Pre-production Open Pit Mining	42.9
Pre-production Underground Mining	62.8
Mining Infrastructure	5.1
Paste Plant	6.0
Construction Costs	
Earthworks	15.0
Process Plant	146.8
Non-Process Infrastructure	30.4
Plant Buildings	3.9
Accommodation Village	13.4
Tailings Storage Facility	18.3
Power Supply and Distribution	2.5
Construction Indirect Costs	31.3
Owners Costs	
Owners Project Management	14.2
First Fills & Spares	12.0
Vehicles & Cranage	7.8
Corporate Project Costs	9.6
Pre-Production Costs	6.6
Contingency	**34.3**
Total	**463.0**

Notes:

1. Totals may not compute due to rounding

Execution readiness activities, which have been estimated at US$11.8m will be undertaken prior to final investment decision and will be funded from existing cash reserves.

Various capital and sustaining costs will also be incurred during the operations phase, totalling US$164.3m as detailed in Table 1.7.

Table 1.7: Life of Mine Sustaining and Deferred Capital Estimate

Area	US$m
Underground Development	91.7
Refrigeration Plant	4.6
BC Pit Establishment	5.7
TSF Expansion and Progressive Closure	58.5
General Sustaining Capital	3.9
Total	**164.3**

1.16.2 Operating Costs

The DFS operating cost estimate has been developed from first principles by category and cost type in line with a Class 3 estimate with a target accuracy of ±15%. The date of the estimate is the 1st quarter 2025, presented in US$. Table 1.8 presents the life of mine operating costs for the Project.

Table 1.8: Life of Mine Operating Costs

Area	LOM Cost US$m	Unit Costs	Unit Cost US$/oz
Open Pit Mining	604.3	US$4.97/t material mined	222.81
Underground Mining	442.1	US$57.39/t ore mined	163.00
Processing	873.7	US$17.08/t ore milled	322.13
Tailings Handling	174.6	US$4.54/t milled	64.38
General and Administration	164.8	US$14.3m per annum	60.77
Transport & Refining	22.9	US$8.45/oz	8.45
C1 Cash Costs	**2,282.4**		**841.55**
Royalties	390.6	6 % of Revenue	144.00
Sustaining Capital & Closure Costs	175.9	-	64.86
All-in Sustaining Costs	**2,849.9**		**1,050**

1.17 Economic Analysis

At a consensus gold price assumption of US$2,400 per oz, the Project, based on processing of the LOM production schedule, delivers a post-tax net present value at 5% discount (NPV5%) of US$1,445 million and an internal rate of return (IRR) of 46%, achieving payback in 1.8 years. Financial outcomes improve significantly at the spot gold price of US$3,300. Key project metrics are included in Table 1.9 and key financial metrics in Table 1.10.

Table 1.9: Key Project Metrics

	Units	
Mining		
Open Pit Ore Mined	Mt	43.7
Open Pit Strip Ratio	-	1.9
Open Pit Grade	g/t	1.39
Open Pit Contained Gold	koz	1,951
Underground Ore Mined	Mt	7.7
Underground Grade	g/t	3.93
Underground Contained Gold	koz	972
Total Ore Mined	Mt	51.4
Average Grade	g/t	1.77
Total Contained Gold	koz	2,924
Processing		
Mine Life	Years	11 years and 6 months
Processing Rate	Mtpa	4.5
Total Ore Processed	Mt	51.4
Average Processing Recovery	%	92.8
Total Gold Production	koz	2,712
Average Gold Production	koz pa	236
Capital Cost		
Direct Capital Cost	US$m	272.9
Owners Costs	US$m	50.2
Contingency	US$m	34.3
Pre-Production Mining	US$m	105.6
Total Pre-Production Capital Cost	**US$m**	**463.0**
Sustaining Capital	US$m	164.3
Closure Costs (excluding salvage)	US$m	39.6
Operating Cost		
C1 Cash Cost	US$/oz	842
All-in Sustaining Cost	US$/oz	1,050

Table 1.10: Key Financial Metrics

	Units	Base Case US$2,400/oz	Spot Price US$3,300/oz
Pre-Tax $NPV_{5\%}$	US$m	2,051	3,638
Pre-Tax IRR	%	57	90
Pre-Tax Payback Period	Years	1.5	1.0
Post-Tax $NPV_{5\%}$	US$m	1,445	2,571
Post-Tax IRR	%	46	74
Post-Tax Payback Period	Years	1.8	1.1

1.18 Interpretation and Conclusions

This report summarises the results of the DFS for the Bankan Gold Project, which demonstrates the technical and economic viability of the Project using a combination of open pit and underground mining for the exploitation of the NEB, BC and GBE orebodies.

1.18.1 Mineral Resources

The Qualified Person is of the opinion that the data that informs the resource estimate is considered to be representative and free of any biases or other factors that may materially impact the reliability of the sampling or significantly affect confidence in estimated resources. Checks have been undertaken to confirm the validity of the drilling database indicate that it forms a sufficiently reliable basis for resource estimation.

1.18.2 Mineral Reserves

The Mineral Reserves stated in this report are based on open pit and underground mining methods, and these have been reported separately with separate Qualified Persons. The Qualified Persons are of the opinion that the Mineral Reserves were estimated using industry accepted practices and conform to the CIM Definition Standards and the JORC Code. The proposed open pit and underground mining of the Project is considered reasonable and technically viable based on the parameters and assumptions outlined in this report.

1.18.3 Mineral Processing and Recovery

The results of the mineral processing and metallurgical testing conducted on representative samples from the deposits are considered by the Qualified Person to be of sufficient scope and quality to support the proposed plant design, performance prediction and production forecasting. The crushing circuits (with separate crushing of sticky clay material from competent ore) and SABC comminution circuit can accommodate the spectrum of material properties included in the mine production plan. The ore has been demonstrated to be amenable to conventional processing using gravity recovery, CIL and cyanide destruction. Tailings will be filtered for disposal and sufficient testing has been carried out to develop a reliable design basis for this aspect of the processing. The recovery circuit outlined in this report is conventional for gold processing.

1.18.4 Infrastructure

Infrastructure for the Project has been considered and defined to DFS level. Key enabling infrastructure includes roads, offices and other buildings, accommodation village, power station and solar array, tailing storage facility and surface water management infrastructure.

1.18.5 Environmental and Social

The environmental clearance certificate for the Project has been issued and the company has developed suitable environmental and social management plans. Detailed planning is underway for the economic resettlement action plan which will be implemented on granting of the exploitation permit to provide full access to the land required to develop the Project.

1.19 Recommendations

An execution readiness and front-end engineering and design (FEED) program has been developed and costed and is recommended to be carried out in parallel with Project funding activities. This recommended program includes the following:

- Build-up of the onwner's team, development of project execution plans and tendering of key execution contracts.
- Implementation of the economic resettlement action plan.
- Carrying out grade control drilling of the GBE deposit along with additional site investigations for mine and site geotechnical conditions to support detailed design.
- Drilling of dewatering bores in preparation for the commencement of mining.
- Completion of FEED for the processing plant, NPI and other areas of the Project.
- Commencement of procurement of long-lead items for the process plant, NPI and village.

This program has been estimated to cost US$11.8m for the execution readiness activities and US$5.3m for the FEED.

2 INTRODUCTION

2.1 Terms of Reference

This Report was prepared for Predictive Discovery Limited (PDI or the Company) on the Bankan Gold Project, Guinea (the Project). PDI is an Australian-incorporated company which listed on the Australian Securities Exchange (ASX) in December 2010 as a gold and mineral exploration company. The Company is focused on delivering long-term sustainable value to shareholders and stakeholders via the development of the Bankan Gold Project in Guinea, West Africa.

The Report complies with the requirements of NI 43-101 for reports filed under Canadian jurisdiction, for the Effective Date of the Exploration Results, Mineral Resource and Mineral Reserve estimates.

This Report was prepared for the purposes of reporting on the definitive feasibility study (DFS) released to the Australian Stock Exchange on 25 June 2025 in accordance with the Joint Ore Reserves Committee (JORC) "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves" to align with the continuous disclosure of Exploration Results, Mineral Resources and Mineral Reserves in accordance NI 43-101.

The classification categories of Measured, Indicated and Inferred under the JORC Code are equivalent to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) categories of the same names (CIM Standing Committee on Reserve Definitions, 2014). Proved and Probable Ore Reserves under the JORC Code are equivalent to Proven and Probable Mineral Reserve categories defined in CIM. For the avoidance of doubt, the term Mineral Reserves will be used for Ore Reserves and Mineral Reserves in this report. Classifications will be referred to according to the CIM nomenclature.

The effective date of this Report is July 31, 2025.

2.2 Qualified Persons and Site Visits

The Qualified Persons for this report are Mr Philip Jankowski, Mr Ross Cheyne, Mr Julian Broomfield, Mr Peter O'Bryan, Mr Pieter Labuschagne and Mr Stewart Watkins.

Philip Jankowski

Mr Philip Jankowski is a Principal Consultant for resource geology and technical mining services with ERM Australia Consultants and holds a Master of Science (Geology) from the University of Western Australia and a Bachelor of Science (Geology/Earth Science) from the Australian National University. He is a Fellow in good standing of the Australasian Institute of Mining and Metallurgy. He has over 30-years' continuous professional experience in exploration, operations and resource geology. He is familiar with NI 43-101 and, by reason of his education, experience and professional registrations, he fulfils the requirement of an independent Qualified Person as defined in NI 43-101.

Mr Jankowski visited site numerous times from 2022 to 2024 and inspected the following:

- General site layout, including the NEB and BC deposits, the Fouwagbe and Sounsoun deposits, Bankan village and surrounding areas.
- Diamond core drilling.
- Drillhole setup.

- Core orientation and markup.
- Core logging.
- Core sampling.
- Density measurement procedure.
- PLT measurement procedure.
- XRF measurement procedure.
- Reverse circulation (RC) drilling.
- RC sampling.
- Aircore drilling and sampling.
- Auger drilling and sampling.
- Sample dispatch.
- Core and RC retention bag storage.
- Pulp storage.
- Review of selected core intervals.

Detailed technical discussions with PDI staff were also conducted.

Ross Cheyne

Mr Ross Cheyne is a Principal Consultant with Orelogy Mine Consulting and holds a Bachelor of Engineering (Mining) from the University of Aukland. He is a Fellow in good standing of the Australasian Institute of Mining and Metallurgy. He has over 30-years' continuous professional experience in mine planning, mining operations, mine design and estimation of Ore and Mineral Reserves. He is familiar with NI 43-101 and, by reason of his education, experience and professional registrations, he fulfils the requirement of an independent Qualified Person as defined in NI 43-101.

Mr Cheyne conducted a site visit in January 2025 and inspected the following:

- Road access to site from Conakry.
- The general site layout, including but not limited to:
  - NEB/GBE and BC deposits.
  - Process plant.
  - Mining infrastructure.
  - TSF.
  - Magazine.
- Bankan village and surrounding areas.
- Kouroussa township.

- Diamond core drilling samples stored at site.

Detailed technical discussions with PDI staff were also conducted.

Julian Broomfield

Mr Julian Broomfield is a Principal Consultant with Orelogy Mine Consulting and holds a Bachelor of Engineering (Mining) from the Western Australian School of Mines. He is a Fellow in good standing of the Australasian Institute of Mining and Metallurgy. He has 30-years' continuous professional experience in underground mining operations, underground mine design and estimation of Ore and Mineral Reserves. He is familiar with NI 43-101 and, by reason of his education, experience and professional registrations, he fulfils the requirement of an independent Qualified Person as defined in NI 43-101.

Mr Broomfield has not visited the site and has relied on the observations of Mr Cheyne.

Stewart Watkins

Mr Stewart Watkins is a Director and Principal Consultant with Dhamana Consulting Pty Ltd and holds a Bachelor of Engineering (Chemical) from the Curtin University. He is a Fellow in good standing of the Australasian Institute of Mining and Metallurgy. He has over 30-years' continuous professional experience in process plant operations and design along with feasibility study management, project evaluation and project delivery. He is familiar with NI 43-101 and, by reason of his education, experience and professional registrations, he fulfils the requirement of an independent Qualified Person as defined in NI 43-101.

Mr Watkins conducted a site visit in January 2025 and inspected the following:

- Road access to site from Conakry.
- The general site layout, including but not limited to:
  - NEB/GBE and BC deposits.
  - Process plant.
  - Mining infrastructure.
  - TSF.
  - Magazine.
  - Access road alignment.
  - Borrow pit locations.
  - Village site.
- Bankan village and surrounding areas.
- Kouroussa township.
- Diamond core drilling samples stored at site.

Detailed technical discussions with PDI staff were also conducted.

Peter O'Bryan

Mr Peter O'Bryan is a Principal Consultant with Peter O'Bryan and Associates and holds a Bachelor of Engineering (Mining) from the University of New South Wales and a Master of Engineering Science (Rock Engineering) from James Cook University. He is a Chartered Professional Member of the Australasian Institute of Mining and Metallurgy. He has over 40-years' continuous professional experience in geomechanical engineering for underground and open pit mining.

He is familiar with NI 43-101 and, by reason of his education, experience and professional registrations, he fulfils the requirement of an independent Qualified Person as defined in NI 43-101.

Mr O'Bryan has not visited the site and has relied on the observations of qualified and experienced staff from Peter O'Bryan & Associates who have visited site acting under his supervision.

Pieter Labuschagne

Mr Peiter Labuschagne is a Principal Consultant with Australasian Groundwater and Environmental Consultants Pty Ltd and holds a M.Sc Hydrogeology from the University of the Free State, South Africa. He is a member of the Intranational Association of Hydrogeologists (IAH) and registered Scientists in South Africa (Pr.Sci.Nat 400386/11). He has over 25-years' continuous professional experience in hydrogeological assessments for the mining industry. He is familiar with NI 43-101 and, by reason of his education, experience and professional registrations, he fulfils the requirement of an independent Qualified Person as defined in NI 43-101.

Mr Labuschagne has not visited the site and has relied on the observations of qualified and experienced staff from Australasian Groundwater and Environmental Consultants who have visited site acting under his supervision.

2.3 Qualified Persons Areas of Responsibility

Table 2.1 provides a detailed list of the sections of this report and the qualified person responsible for the section.

Table 2.1: Qualified Persons Areas of Responsibility

Section	Title	Qualified Person
1	Summary	
1.2, 1.3, 1.4, 1.5, 1.6, 1.7, 1.9, 1.18.1	*History, Geological Setting and Mineralisation, Exploration, Drilling, Sample Preparation, Analysis and Security, Data Verification, Mineral Resource Estimate, Interpretation & Conclusions (Mineral Resources Only))*	Philip Jankowski
1.10 (excluding underground Mineral Reserves), 1.11 (excluding 1.11.2), 1.18.2,	*Mineral Reserve Estimate (excluding underground Mineral Reserve Estimate), Open Pit Mining, Mine & Production Schedules, Interpretation & Conclusions (Mineral Reserves Only)*	Ross Cheyne
1.10 (underground Mineral Reserves only), 1.11.2	*Underground Mineral Reserve Estimate, Underground Mining*	Julian Broomfield

Section	Title	Qualified Person
1.1, 1.8, 1.12, 1.13, 1.14, 1.15, 1.16, 1.17, 1.18 (excluding 1.18.1 and 1.18.2), 1.19	*Introduction, Location and Ownership, Mineral Processing and Metallurgical Testing, Recovery Methods, Project Infrastructure, Market Studies and Contracts, Capital and Operating Costs, Economic Analysis, Interpretations and Conclusions (excluding Mineral Resources and Mineral Reserves), Recommendations*	Stewart Watkins
2	Introduction	Stewart Watkins
3	Reliance on Other Experts	Stewart Watkins
4	Property Description and Location	Stewart Watkins
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Stewart Watkins
6	History	Philip Jankowski
7	Geological Setting and Mineralisation	Philip Jankowski
8	Deposit Types	Philip Jankowski
9	Exploration	Philip Jankowski
10	Drilling	Philip Jankowski
11	Sample Preparation, Analyses and Security	Philip Jankowski
12	Data Verification	Philip Jankowski
13	Mineral Processing and Metallurgical Testing	Stewart Watkins
14	Mineral Resource Estimate	Philip Jankowski
15	Mineral Reserve Estimate	
15.1,15.2,15.3, 15.6,	*Introduction, Mineral Reserve Statement, Mineral Resources, Open Pit Optimisation*	Ross Cheyne
15.7	*Underground Optimisation*	Julian Broomfield
15.4	*Geotechnical*	Peter O'Bryan
15.5	*Hydrogeology*	Pieter Labuschagne
16	Mining Methods	
16.1,16.2.1, 16.3, 16.4, 16.6, 16.7	*Overall Mining Strategy, Mining Method Selection – Open Pit Mining, Mine Design Basis and Optimisation, Open Pit Mine Design, Mine Schedules, Open Pit Mining Operations*	Ross Cheyne
16.2.2, 16.5, 16.8	*Mining Method Selection – Underground Mining, Underground Mining, Underground Mining Operations*	Julian Broomfield
17	Recovery Methods	Stewart Watkins
18	Project Infrastructure	Stewart Watkins
19	Market Studies and Contracts	Stewart Watkins
20	Environmental Studies, Permitting and Community Impact	Stewart Watkins
21	Capital and Operating Costs	
21.2.3.1	*Open Pit Mining Costs*	Ross Cheyne
21.2.3.2	*Underground Mining Costs*	Julian Broomfield

Section	Title	Qualified Person
21.1, 21.2 excluding 21.2.3	*Capital Cost Estimate, Operating Costs*	Stewart Watkins
22	Economic Analysis	Stewart Watkins
23	Adjacent Properties	Stewart Watkins
24	Other Relevant Data and Information	Stewart Watkins
25	Interpretation and Conclusions	
25.1	*Mineral Resources*	Philip Jankowski
25.2, 25.3	*Mineral Reserves, Mining Methods*	Ross Cheyne
25 excluding 25.1, 25.2 and 25.3	*Interpretations and Conclusions*	Stewart Watkins
26	Recommendations	
26.1	*Mine Geotechnical*	Peter O'Bryan
26.2	*Mining*	Ross Cheyne
26.5, 26.6	*Hydrogeology, Hydrology*	Pieter Labuschagne
26 excluding 26.1, 26.2, 26.5 and 26.6	*Recommendations*	Stewart Watkins
27	References	Stewart Watkins

2.4 Units and Currency

2.5 Data Sources

A list of the sources of information and data contained in this report is provided in Section 27 of this report. Project-specific information relied upon in preparing this report has been provided by PDI.

2.6 Units, Currency and Abbreviations

Unless stated otherwise, Le Système International d'Unités (SI) units have been used throughout the report (note, that some more commonly used non-metric units have been retained for ease of understanding, e.g. gold tenors are reported in troy ounces in some instances).

Currencies used in the report are US dollars, unless noted otherwise. Conversion rates from local or other currencies to US dollars used in cost estimates or financial analyses are reported in the relevant sections.

The units and symbols used in this report are provided in Table 2.2. The abbreviations used in the report and provided in Table 2.3

Table 2.2: Units and Symbols

Section	Title
%	Percent
±	Plus or minus
μm	micrometre
DS/m^{2}h	Dry solids per meter squared per hour
g	grams
g/t	Grams per tonne
GWh	Gigawatt hours
h/y	Hours per year
ha	Hectares
k/t	kilo tonnes
kg	kilograms
kg/d	Kilograms per day
kg/t	Kilograms per tonne
km	Kilometre
km^2	Square kilometres
koz	Kilo ounces, one thousand ounces
koz/a	Kilo ounces per annum
kV	Kilovolt
kVA	Kilovolt ampere
kW	Kilowatt
kWh/t	Kilowatt hours per tonne
L	Litres
L/s	Litres per second
m	Metres
m/d	Meters per day
m/month	Meters per month
m/s	Metres per second
m^2	Square meters
m^3	Cubic metres
m^3/a	Cubic metres per annum
m^3/h	Cubic metres per hour
m^3/month	Cubic metres per month
m^3/s	Cubic metres per second
mBGL	Metres below ground level

Section	Title
mg	Milligrams
mg/L	Milligrams per litre
mm	Millimetres
Moz	Million ounces
Moz/a	Million ounces per annum
MPa	Megapascals
mRL	Meters relative level
Mt	Million tonnes
Mtpa	Million tonnes per annum
MVA	Mega volt ampere
MW	Megawatt
º	Degrees
ºC	Degrees Celsius
P_{80}	80 percent mass passing
P_n	Nth percentile
pa	per annum
ppm	parts per million
t	tonnes
t/a	tonnes per annum
t/h	tonnes per hour
t/m^2h	tonnes per meter squared per hour
t/m^3	tonnes per cubic meter
US$	United States dollar
US$m	Million United States Dollars

Table 2.3: Abbreviations

Section	Title
800W	800 West deposit
AACE	Association for the Advancement of Cost Engineering
AC	Aircore drill hole
AEP	Average exceedance probability
AGE	Australian Groundwater and Environmental Consultants
Ai	Abrasion index
AISC	All in sustaining cost
AMS	Annual maximum series

Section	Title
AMSL	Above mean sea level
ANCOLD	Australian National Committee on Large Dams
AoI	Area of influence
ASM	Artisanal scale mining
ASTM	American standards
BAC	Bulk air cooling
BBWi	Bond ball mill work index
BC	Bankan Creek deposit
BESS	Battery energy storage system
BGL	Below ground level
BKS	Stope backs/span
BM	Ball mill
BOO	Build, own, operate
BRWi	Bond rod mill work index
CBR	California bearing ratio
CCTV	Closed circuit television
CIL	Carbon-in-leach
CIP	Carbon-in-pulp
CMIP6	Coupled Model Intercomparison Project
CNSS	National social security fund in Guinea
Company	Predictive Discovery Limited
CRM	Certified reference material
DCP	Dynamic cone penetrometer test
DDH	Diamond core drill hole
DFS	Definitive feasibility study
Dhamana Consulting	Dhamana Consulting Pty Ltd
DBT	Dry bulb temperature
ECC	Certificate of environmental compliance
EHS	Environmental, health and safety
EPCM	Engineering, procurement and construction management
ERM	Environmental Resource Management
ESIA	Environmental and social impact assessment
ESMMP	Environmental and social management and monitoring plan
ESMS	Environmental and social management system
FAR	Fresh air raise

Section	Title
FEL	Front end loader
G&A	General and administration
GBE	Gbengbeden deposit
GEV	Generalized extreme value
GHG	Greenhouse gas
GISTM	Global Industry Standard for Tailings Management
GP	General purpose cement
Guinea	Republic of Guinea
GWMMP	Groundwater monitoring and management plan
HDPE	High density polyethylene
HFO	Heavy fuel oil
HR	Hydraulic radius
HW	Hanging wall
ICCM	International Council on Mining and Metals
IFC	International Finance Corporation
IMO	Independent Metallurgical Operations
IPMT	Integrated project management team
IRR	Internal rate of return
JORC	Australasian Joint Ore Reserves Committee
LH	Slag based low heat cement
LRP	Livelihood restoration plan
mbs	Meters below surface
MEDD	Ministry of Environmental and Sustainable Development
MIA	Mining infrastructure area
MMG	Ministry of Mines and Geology
MTO	Material take off
NAF	Non acid forming
NAG	Net acid generation
NEB	Northeast Bankan deposit
NPI	Non process infrastructure
NPV	Net present value
OP	Open pit
PAF	Potential acid forming
PCD	Pollution control dam
PDI	Predictive Discovery Limited

Section	Title
PFS	Prefeasibility study
PMC	Project management consultant
PPA	Power purchase agreement
PV	Photovoltaic
QAQC	Quality assurance / quality control
RAR	Return air raise
RC	Reverse circulation drill hole
RCGC	Reverse circulation grade control drill hole
RI	Return interval
RMR	Mean rock mass rating
ROM	Run of mine
RTS	Income tax in Guinea
SAG	Semi autogenous grinding
SG	Specific gravity
SO	Deswick.SO software
SPT	Standard penetrometer tests
SSP	Shared socio-economic pathways
STMZ	Main shear zone
STMZ01	Second order shear zone
SWMP	Stormwater management plan
TARP	Trigger, action, response plan
TOFR	Top of fresh rock
TSF	Tailings storage facility
UG	Underground
UTPM	Technical and planned use of water resource
VIP	Very important person
VWP	Vibrating wire piezometers
WBT	Wet bulb temperature
WHO	World Health Organisation
WRD	Waste rock dump
WWTP	Wastewater treatment plant
XRD	X-ray diffraction

3 RELIANCE ON OTHER EXPERTS

In preparing this Report, the Qualified Persons have fully relied upon certain work, opinions and statements of experts and the Company concerning legal, political, environmental, or tax matters relating to the Project. The authors consider the reliance on other experts and the Company, as described in this section, as being reasonable based on their knowledge, experience, and qualifications. In addition, the Qualified Persons have relied upon previous environmental and other studies and government submissions for general background information on the Project prepared under the management of the other experts.

Table 3.1 outlines the relevant information that has been relied upon from the Company, often compiled by other experts, by the Qualified Persons in good faith in the preparation of the report.

Table 3.1: Information Relied Upon from the Company

Section	Information
4.2, 4.3	Ownership and title, legal obligations
4.4	Environmental risks, liabilities and permitting
20	Environmental studies, permitting and social or community impact
21.2.4	Guinean labour costs used in estimation of operating costs
22.1	Taxation, duties, levies etc., diesel fuel price, HFO price, sunk costs for depreciation used in estimation of operating costs and financial modelling

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Project is located in the northeast part of Guinea, as depicted in Figure 4.1. It is approximately 450 km east-northeast of Guinea's capital city, Conakry, in the Kouroussa Prefecture.

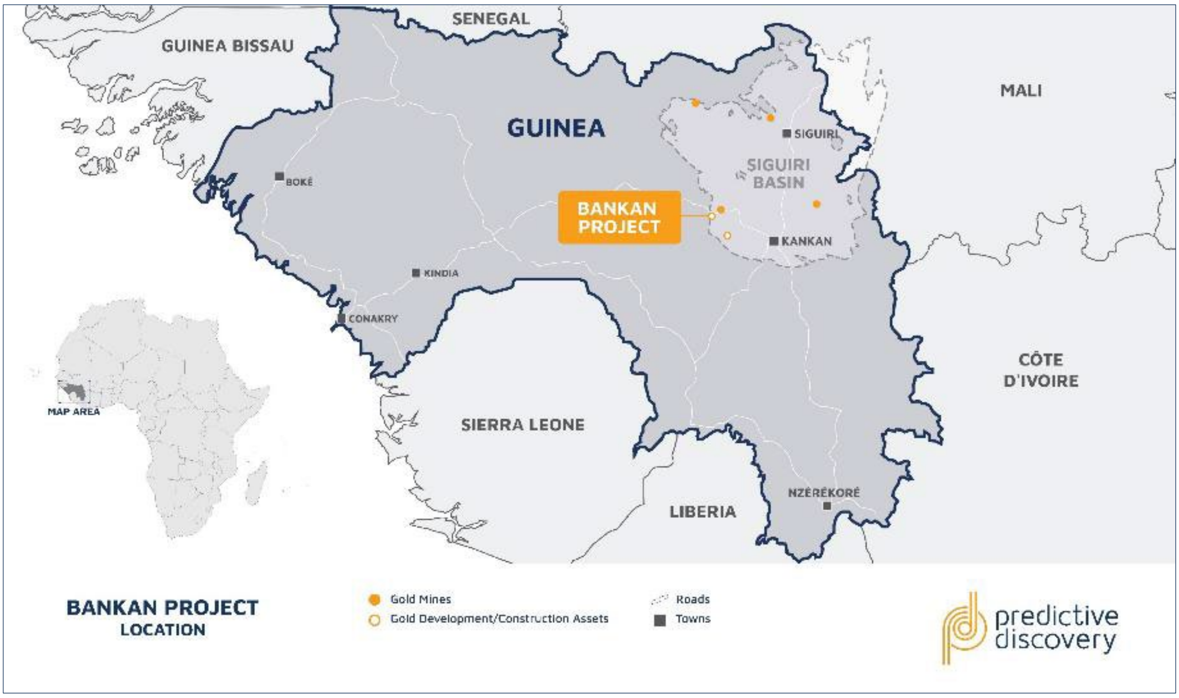

Figure 4.1: Project Location (PDI 2025)

At a regional level, as shown in Figure 4.2, the Project is located 75 km northwest of the regional city of Kankan and 7 km southwest of Kouroussa town. The main Project area, consisting of the NEB pit and processing plant is approximately 3.5 km to 4 km kilometres from the Niger River, which is the third longest river in Africa (4,200 km long). The BC pit is located approximately 1.5 km from the Niger River at its closest point.

The main Project area lies within the Peripheral Zone of the Upper Niger National Park with the NEB and BC deposits approximately 21 km and 18 km, respectively, away from the closest point of the Core Conservation Area.

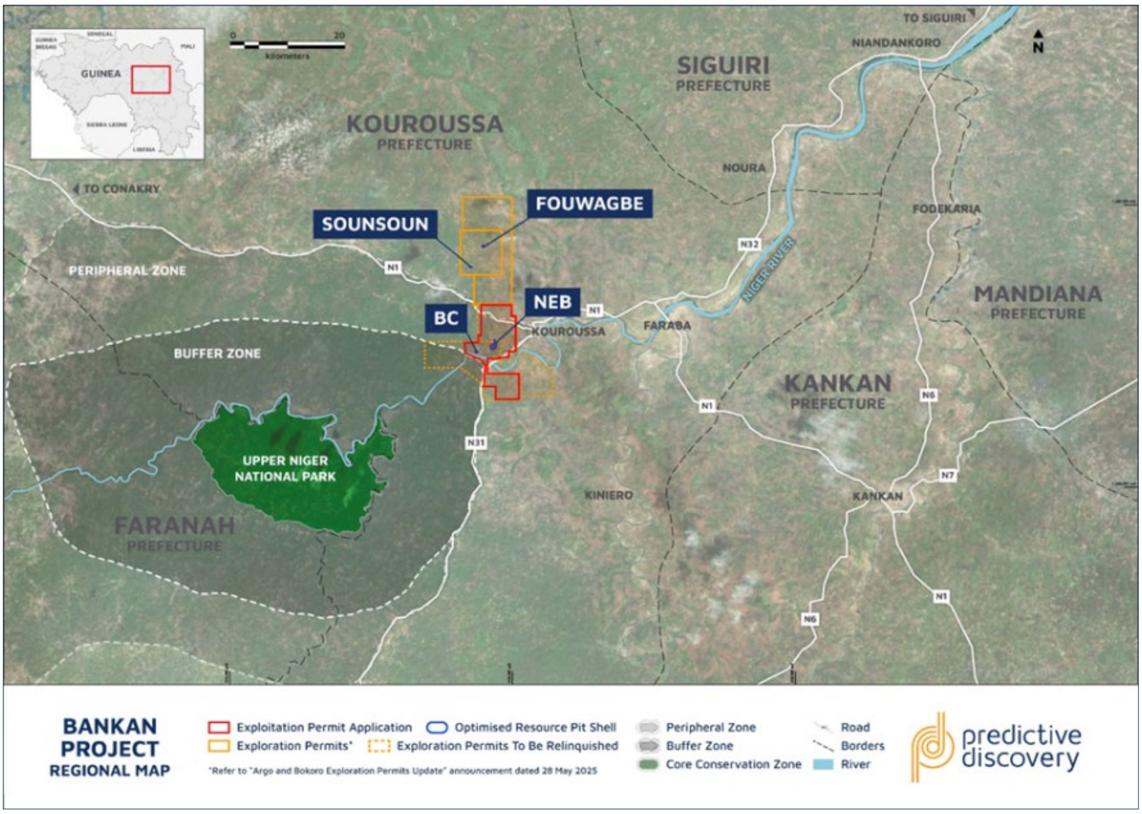

Figure 4.2: Project Region (PDI 2025)

4.2 Ownership

The Project comprises four contiguous *Permis de Recherce Industrielle (Or)* (exploration permits), which cover a combined area of 356 km^2 and are located between 9 51’00”W and 10 03’24”W and between 10 32’26”N and 10 52’00”N, shown in Figure 4.3.

PDI’s four exploration permits relating to the Project and its wider exploration potential, comprise:

- Kaninko gold exploration permit, issued by order no. A/2019/5784/MMG in favour of PDI's wholly owned local subsidiary Mamou Resources SARLU (Mamou) on 3 October 2019, covering a 98.22 km^2 area.
- Saman gold exploration permit, issued by order no. A/2020/1835/MMG in favour of Mamou on 11 June 2020, covering a 99.78 km^2 area.
- Bokoro gold exploration permit, issued by order no. A/2020/2561/MMG in favour of PDI's wholly owned local subsidiary Kindia Resources SARLU on 9 September 2020, covering a 99.98 km^2 area.
- Argo gold exploration permit, issued by order no. A/2018/7628/MMG in favour of Argo Mining SARLU on 24 October 2018 (in which PDI is a shareholder and has the right to progressively earn 90% by payment of US$100,000 and acquire the remaining 10% at a decision to mine in exchange for a 2% net smelter royalty), covering a 57.54 km^2 area.

The main Project area, and all the Mineral Resources on which this DFS is based, are situated on parts of the Kaninko and Saman exploration permits.

On 31 January 2025, PDI and Mamou submitted exploitation permit applications for 50% of the Kaninko and Saman permit areas to the MMG and CPDM in accordance with Guinean mining law. PDI has indicated that the applications are at an advanced stage and are still being processed. PDI is not aware of any immediate obstacles to the granting of the exploitation permits. Figure 4.3 shows the exploitation permit applications area and remaining portions of the Kaninko and Saman permit areas that will be relinquished upon grant of the exploitation permits.

Following the grant of the exploitation permits, PDI intends to negotiate a *Convention minière* , or mining agreement, in relation to the exploitation permits which will ultimately be ratified by the National Assembly of Guinea (currently the National Transition Council). The *Convention minière* is defined in the Mining Code as the agreement establishing the rights and obligations of the holder of an exploitation title with regard to the legal, technical, financial, fiscal, administrative, environmental and social conditions applicable to the title.

PDI submitted renewal applications for the Argo and Bokoro exploration permits in 2021 and 2023 respectively, and has relied on Article 78 of the Guinean Mining Code that allows for permits to be extended automatically until the date of renewal.

PDI has been made aware that, on 26 May 2025, the MMG announced the revocation of over 100 exploration permits, including the Argo exploration permit (which hosts the Fouwagbe and Sounsoun Deposits) and the Bokoro exploration permit. PDI has not received any formal communication from the Guinean government on the matter and intends to work diligently with the MMG to achieve the granting of the renewals.

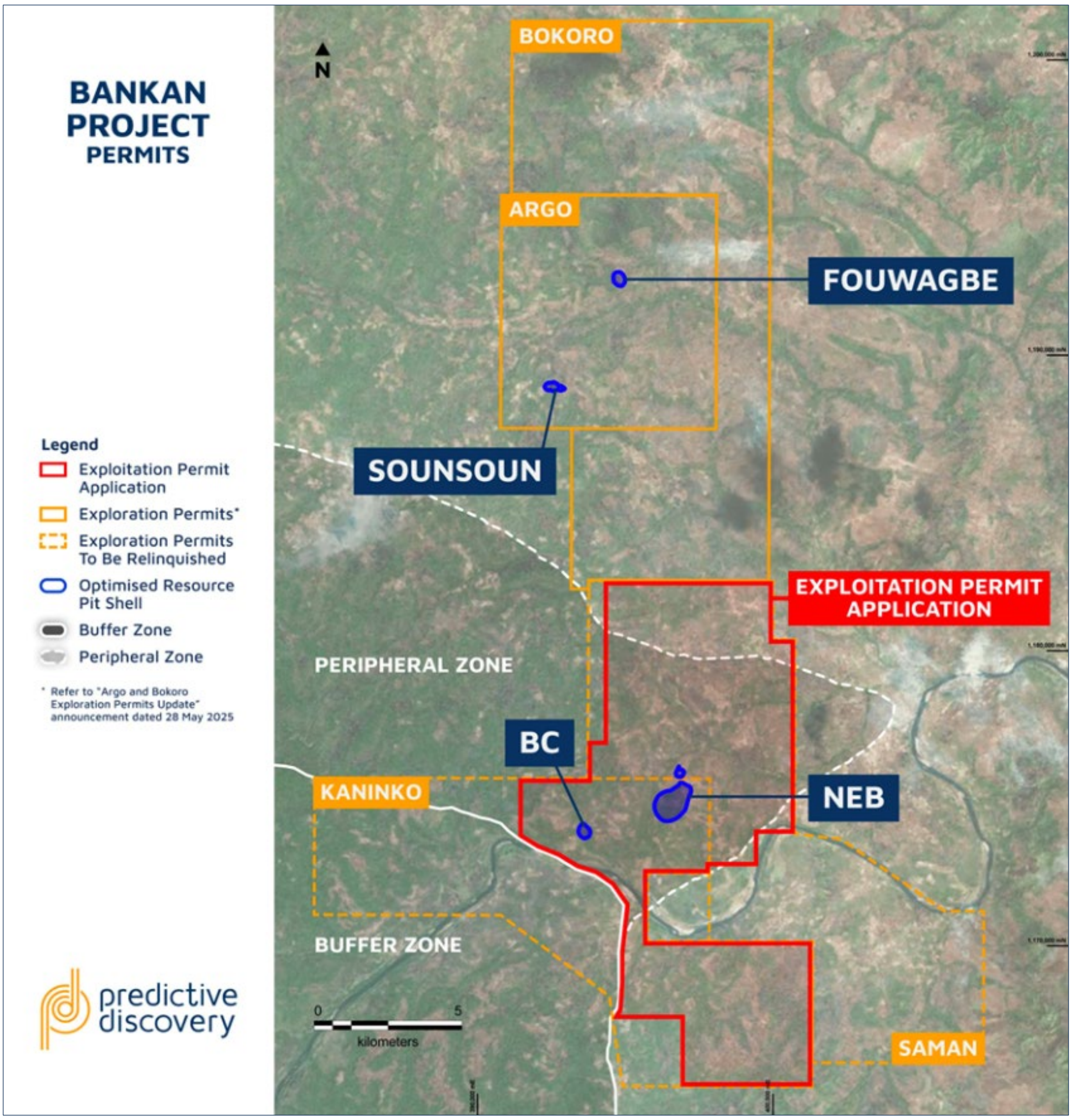

Figure 4.3: Project Permits (PDI 2025)

4.3 Legal Obligations

The Guinean legislation, regulations and national standards applicable to the Proejct include, but are not limited to:

- Mining Code (Law L/2011/006/CNT of September 2011, Amended by Law L/2013/053/CNT OF 8 April 2013).
- Water Code (Law L/ 94/ 005 /CTRN of 15 February 1994).
- Code for the Protection of Wildlife and Regulation of Hunting (Ordinary Law N° 2018/0049/AN of June 2018).
- Guinean Standards: NG 09-01-010:2012 / CNQ:2004 relating to new standards for wastewater discharges.

- Guinean Standards: NG 09-01-011:2012 / CNQ:2004 relating to new standards for air pollutant emissions.
- Guinean Standards: NG 09-01-012:2012 / CNQ:2004 relating to new standards on exposition to chemicals at work.
- Guinean Standards: NG 09-01-013:2012 / CNQ:2004 relating to new procedures for environmental inspection of industrial and commercial facilities.
- Guinean Standards on noise emissions.
- Local Government Code (1.12017/040/AN of 24 February 2017).
- The Labour Code (L/2014/072/CNT of 10 January 2014).
- Local Content Law (L/2022/0010/CNT on Local Content of 22 September 2022).
- Social Security Code (L/94/006/CTRN of 14 February 1994).
- Social Societies Law (L/2021/0017/AN of 30 April 2021).
- Law on the Protection, Conservation and Enhancement of National Cultural Heritage (L/2016/063/AN of 9 November 2016).
- Land and forestry legislation, including:
  - Land and Public Estate Code "*Code Foncier et Domanial*" (Ordonnance 0/92/019).
  - Urban Planning Code (Law L/98/017/98 of 13 July 1998).
  - Declaration of a Rural Land Tenure Policy (2001).
  - Forestry Code (Ordinary Law L/2017/ N°0038/AN of 24 April 2017).
  - Pastoral Code (Law L/95/51/CTRN of 29 August 1995).
  - Specific Laws and Regulations on Natural Protected Areas and Mining Activities (National Haute Niger).

4.4 Environmental Risks, Liabilities and Permitting

The Qualified Person has not undertaken a detailed review and assessment of the environmental risks, liabilities and permitting requirements for the Project and has relied upon representations provided by PDI pertaining to these matters.

An environmental and social impact assessment (ESIA), including an environmental and social management and monitoring plan (ESMMP), was developed for the Project which aligns with the national laws and regulations, and international regulations. The ESIA, including baseline studies and identifying the potential risks and impacts which may occur due to the Project, was undertaken in March 2024 by Environmental Resource Management (ERM) and the environmental compliance certificate (the *Certificat de Conformité Environnementale*) was approved on the 17 January 2025 (CCE/00070).

PDI is committed to complying with all relevant Guinean national laws and regulations, international standards, such as the International Financial Corporation's (IFC) performance standards on environmental and social sustainability (2012) (IFC-PS), and best practice standards in the environmental, health and safety (EHS) guidelines (2007) as well as human rights standards applicable to the project. PDI has developed a corporate governance framework aimed at aligning with the principles and recommendations of the ASX Corporate Governance Council and the World Gold Council's responsible gold mining principles. Based on the results of the ESIA and ESMMP report, the Company will develop an environmental management system, which will include an environmental policy statement.

4.4.1 Environmental Liabilities

The current environmental liabilities relating to the Project are limited to minor rehabilitation works associated with PDI's exploration activities and facilities located on the exploration permit area. Sufficient provisions have been made for this in PDI's normal course of business.

4.4.2 Permitting

In addition to the requirement for an exploitation permit which is underway, as outlined in Section 4.2, PDI will require other permits to develop and operate the project. These include, but are not limited to:

- Electricity Generation Licence from *Autorité de Régulation de l'Électricité en Guinée* (AREG).
- *Autorisation de Prélèvement d'Eau* (water abstraction permit) required if water is drawn from a natural source.
- *Autorisation de Prélèvement d'Eau Souterraine* (groundwater abstraction authorisation) from the MEDD, in coordination with the Ministry of Energy, Hydraulics and Hydrocarbons.
- *Autorisation de défrichement* (vegetation Clearing) permit authorised by MEDD as a condition for approving the ESIA.
- *Autorisation d'Installation Classée pour la Protection de l'Environnement (ICPE)* (environmental authorisation for classified facilities) required under the Environmental Code for:
  - Waste/landfill facility and to manage solid and hazardous waste (including tailings).
  - Effluent treatment facility and to discharge effluent.
  - Water treatment facility and building permit for a sewage system.
  - Water transport pipeline.
- *Autorisation d'Occupation du Domaine Public ou Privé* (land use authorisation) and an *Accord de Développement Local (ADL)* (land development agreement).
- Authorisation to build fuel storage facilities within the ICPE permit which covers the storage of diesel, gasoline and other hydrocarbons.
- *Autorisation de Construire* (construction authorisation) within the Urban Planning Code and Environmental Code, which also includes the *Autorisation d'Occupation du Sol*. This is required for the mine village and other infrastructure.
- *Autorisation de Construction et d'Exploitation de Lignes Électrique* (authorisation to construct power transmission lines within the lease boundary) from *MEDD*.

- *Autorisation de Détention et d'Utilisation d'Explosifs* (explosive storage and use authorisation) from the Ministry of Mines and Ministry of Security and Civil Protection. This is also governed by the Mining Code and the National Security Regulations.

4.4.3 Land Access

In addition to the permitting requirements, PDI will also be required to acquire approximately 2,000 ha of land for the establishment of mining and infrastructure. Consequently, this will result in economic displacement of the landowners and occupants, leading to the loss of livelihoods. Land that is being used for agriculture, livestock (grazing), artisanal scale mining (ASM) and areas used for ecosystem services will be acquired by the Project. Additionally, the establishment of the Project will result in a change of access to land and areas which have not been acquired.

ASM is one of the main sources of income for the surrounding households surveyed. The majority of the ASM sites are located within the mining permits thus eviction from and/or restricted access to these areas will result in reduced income. The loss of agricultural land and livelihoods could lead to increased food insecurity and reduced sources of income.

In particular, the communities in Bankan and Kignédouba will lose access to land in preparation for the construction of the Project. Therefore, the impact of economic displacement will require careful and proper management pre-construction to minimise the significance of this impact through an economic resettlement action plan, livelihood restoration plan (LRP), ASM management framework and stakeholder engagement framework.

The Project will align with the relevant national and international legislation /frameworks for the acquisition of land and land access. The *Code Foncier et Domanial (Ordonnance 0/92/019)*, or land and public estate code, is a legal framework which governs the land tenure and property rights in Guinea.

The Project has established a Resettlement and Compensation Policy Framework (RCPF) which will serve as the basis for the development of the economic resettlement action plan and the LRP. The RCPF follows the national legislation and standards as well as international standards.

The economic resettlement action plan and LRP will aim to integrate all aspects of the planned economic resettlement related to the Project and the concurrent livelihood restoration activities into the ESMMP and, ultimately, into the Project's ESMS. This plan will serve as the basis for defining and implementing operational compliance management procedures and practices in all Project activities undertaken by the Company, its contractors, and suppliers. All aspects of this plan will be integrated into the Company's activities during the construction, operation, and closure phases of the Project.

The RCPF specifically aims to:

- Define the principles and procedures governing land acquisition, displacement, and involuntary resettlement caused by the Project.
- Enable the Company to define future human, technical, and financial resource needs.
- Plan future relevant engagement activities with stakeholders.

The economic resettlement action plan and LRP will aim to include the following elements:

- Avoid or minimise, as much as possible, involuntary resettlement and land acquisition by exploring all viable alternatives during the Project design.

- Improve, or at least restore, the livelihoods and living standards of people affected by displacement.
- Ensure that affected persons are consulted and can participate in all key stages of the development and implementation process of involuntary resettlement and compensation activities.
- Ensure that compensation is proportional to the impacts suffered, to verify that no person affected by the Project is disproportionately penalised.
- Ensure that affected persons, including those identified as vulnerable, are assisted in their efforts to improve their livelihoods and living standards, or at least to maintain them at their pre-resettlement level or their pre-Project level, whichever is more advantageous for them.
- Preparation of databases for stock and complaint management.
- Establish institutional framework, particularly the establishment of monitoring committees with framework agreements.
- Creation of a schedule linking the economic resettlement action plan to the mining schedule and the schedules of other stakeholders (e.g., the agricultural schedules of farmers affected by the Project).
- Design a compensation strategy in accordance with the resettlement principles outlined in Guinean legislation and international standards, particularly IFC Performance Standard 5.
- Compensation should be in kind, in land or in cash, where a person is able to choose between two or more compensation options according to their assets.
- The economic resettlement action plan process will require a grievance management mechanism, stakeholder engagement, and inclusion of non-land-related impacts from the Project.
- Livelihood Restoration – affected households will undergo initial socio-economic surveys to define the main demographic, economic, health, and social characteristics of the PAPs and their households.
- Conduct a Livelihood Impact Assessment being a qualitative and quantitative assessment of the impacts on the livelihoods of the PAPs.
- Develop a Livelihood Restoration Strategy following consultations with the PAPs in the form of focus groups and with key stakeholders (technical services, NGOs, microfinance organisations).

The final planning stages for the implementation of the economic resettlement action plan and LRP are currently underway with engagement anticipated to commence following grant of the Project's exploitation permit.

4.5 Other Factors

The Qualified Person is not aware of any other significant factors or risks that may affect access, title, or the right or ability to perform work on the Property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access

Guinea's capital city, Conakry, is serviced by direct international flights from a range of locations, including Paris (France), Brussels (Belgium), Dubai (UAE), Tunis (Tunisia), Casablanca (Morocco), Addis Ababa (Ethiopia) and various locations across West Africa.

Access to the Project from Conakry via road is on the N1 highway over a distance of approximately 570 km. The N1 is the main route from Conakry to Kankan. The N1 is a good condition bitumen highway for its entire length from Conakry to the Project site and is accessible year round. The N1 transects the Project tenure, and current access to Project area is via existing tracks directly off the N1 or off the N31 from Kouroussa to the Niger River crossing to the south of the Project. The N31 road is currently being upgraded.

The Project can also be accessed via charter flight from Conakry to the regional airport at Kankan and then by road via the N1 from Kankan to Kouroussa and the Project.

The historical Conakry to Kankan railway and an associated easement passes through the permits on a similar alignment to the N1. Discussions around re-establishing this infrastructure have been ongoing for over a decade but are unlikely to be material to the Project.

Current access within the Project area is via existing village tracks. These tracks are unsealed, and PDI has completed minor upgrade work to ensure access is possible throughout the year.

5.2 Physiography

The topography in area is characterised by low hills and plains. The highest point is located at 436 m above sea level to the north of the Project area, and the lowest point is 362 m in the Niger River valley just to the east of the Project Area. A view from near the Project area to the Niger River is provided in Figure 5.1 below, which summarises the local terrain.

Figure 5.1: View Towards the Niger River from the Project Area (PDI 2024)

The southern and western parts of the Project area have south-draining valleys directly into the Niger River, while the northern, central and eastern areas drain through shallow valleys into tributaries that pass through the town of Kouroussa before entering the Niger River. Valley slopes are generally gentle, and the interfluvial areas are flat, except to the west of the Project area, where slopes are steeper and hills more pronounced. A terrain and drainage plan is provided in Figure 5.2. The Niger River is approximately 2 km from the proposed BC Pit in an area 15 m above the river and will require a diversion around it and appropriate flood protection. Flood risk is not considered significant as the proposed NEB pit is on top of a hill.

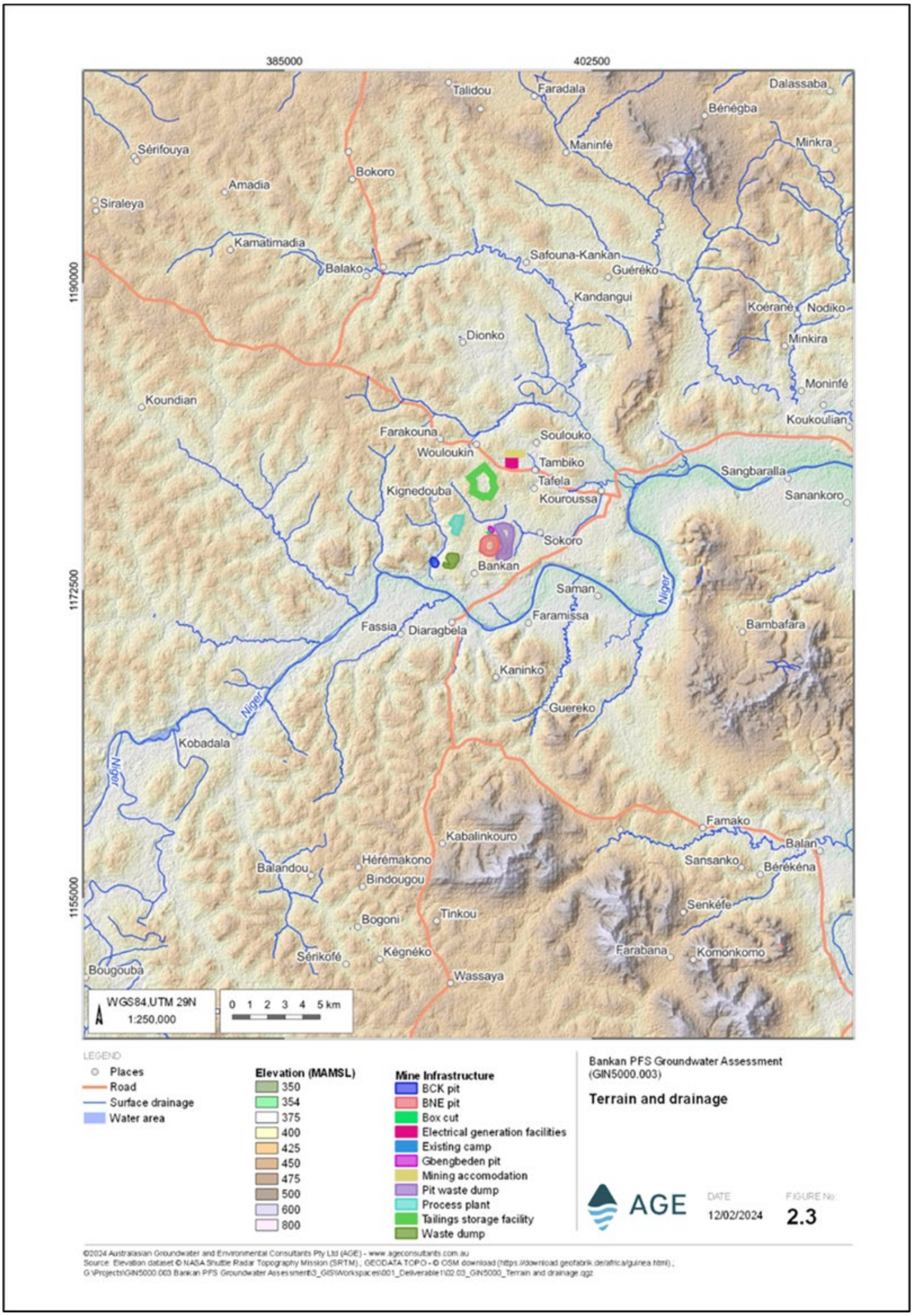

Figure 5.2: Terrain and Drainage Plan

5.3 Climate

The Project is located in the tropical Kankan region of eastern Guinea, which has a distinct wet and dry season. The rainy season runs from May to October, and the dry season from November to April. The site can be classified as having a tropical savannah climate, Köppen climate classification Aw. Table 5.1 and Figure 5.3 show that the mean annual precipitation (MAP), based on the AgERA5 dataset (Food and Agriculture Organization of the United Nations, 2025), is approximately 1,375 mm, with significant variability across the year. Evaporation is higher at about 1,820 mm per annum. The highest maximum day temperatures (around 38°C) are in March and April, and the lowest minimum temperatures (around 15°C) are in December and January.

Table 5.1: Annual Rainfall and Evaporation Data

Month	Rainfall mm	Evapotranspiration mm
January	0	176
February	0	173
March	1	201
April	11	201
May	146	161
June	194	130
July	289	115
August	288	115
September	245	109
October	186	129
November	15	141
December	0	168
Total	**1,375**	**1,819**

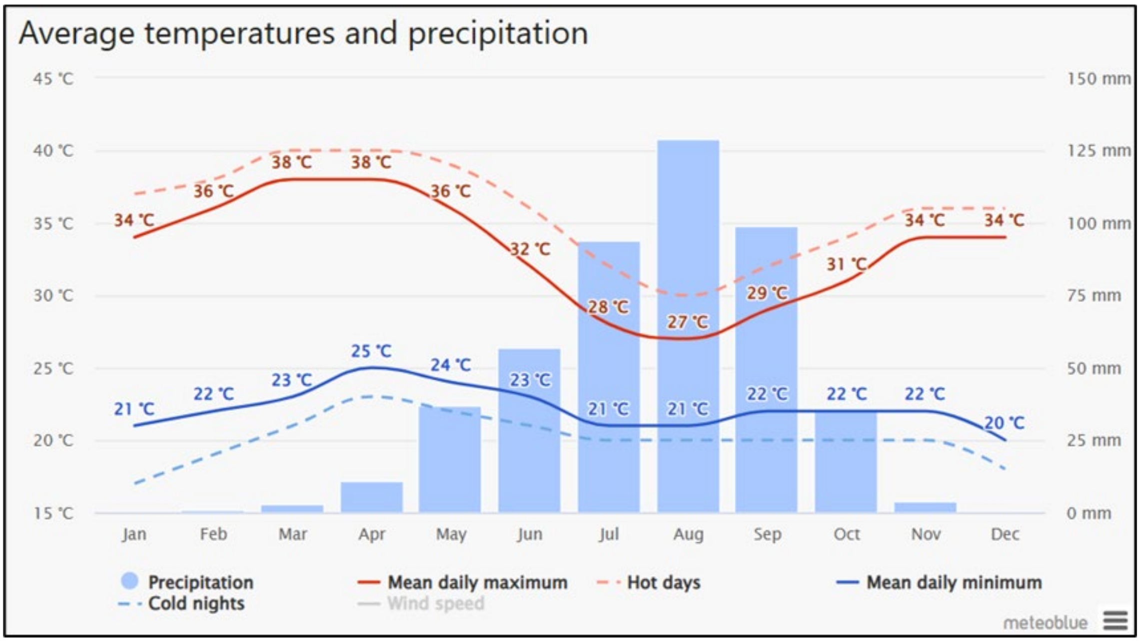

Figure 5.3: Average Rainfall and Temperature (MetoBlue 2025)

The hottest days on record have been close to 44°C (February, March and April), and the lowest recorded temperature is around 12°C in January. The dry season can be quite hot and subject to large temperature extremes and has low humidity. As demonstrated in Figure 6, winds are generally light, with the strongest breezes in the wet season. Prevailing winds typically come from the southwest.

Figure 5.4: Wind Rose for Kouroussa (MetoBlue 2025)

5.3.1.1 Design Rainfall

For the estimation of flooding and stormwater management, design rainfall is considered an important variable and the driver behind peak flows. To provide design rainfall estimates for the site the station at Kankan, approximately 75 km to the southeast was used.

A frequency analysis of the recorded daily rainfall was performed, and the generalised extreme value (GEV) maximum distribution was selected for the frequency analysis. In considering the annual maximum series (AMS) for Kankan, 31 years of data were available and a 1:62-year, return interval (RI) rainfall could be estimated with reasonable confidence. The 1:100-year RI was extrapolated from this and the 24-hour rainfall was estimated at 198.8 mm.

Considering the rainfall distribution over 24-hours, a review of rainfall trends did not conclude with any clear distribution of relevance to the region and site. The more intense (conservative with regards to flooding) SCS-SA Type III storm was thus adopted.

5.3.1.2 Monthly Rainfall

The synthetic Climatic Research Unit CRU 4.09 dataset (Climatic Research Unit, University of East Anglia, 2025) was selected for the long-term rainfall dataset compared to other datasets as it had the highest mean annual rainfall, being 1% higher than the Kankan dataset, 16% higher than Climate Hazards Centre (CHIRPS) (Climate Hazrds Centre, University of California, Santa Barbara, 2025) and 15% higher than Integrated multi-satellite retrievals for global precipitation measurement (IMERG) (NASA, n.d.).

5.3.1.3 Evapotranspiration

Estimates of mean annual reference evapotranspiration were evaluated with estimates ranging between 1,648mm to 2,090mm. As with rainfall, the average of the estimates was used to define the relevant dataset. The AgERA5 (Food and Agriculture Organization of the United Nations, 2025) was selected based on its being nearest to the annual average.

5.3.1.4 Niger River Peak Flows

For the Niger River, a highly seasonal flow from peak to near zero flow is evident in most years. Using the composite record of the Global Runoff Data Centre (Global Run off Data Centre, n.d.) and Niger-HYCOS datasets, an AMS for relevant water years was subsequently extracted containing 79 years of data which allowed a reasonable estimation of the 1:100 year RI peak flow which was estimated at 2,061 m^3/s.

5.3.1.5 Climate Change

The analysis of climate change considered the outcome of the sixth Coupled Model Intercomparison Project (CMIP6) by Seneviratne et.al (2021) with a shared socio-economic pathways (SSP) SSP5 assumed (being a scenario where fossil fuel consumption continues to rise, with a focus on economic growth and technological innovation and limited attention to environmental sustainability) due to its increased influence on flooding and rainfall.

As the mine continues operations into the early 2040s, the medium-term projection according to the CMIP6 becomes relevant (ranging from 2041 to 2060), although the near-term projection (2021-2040) predominates for the majority of the Project life. The 1-day design rainfall (and consequently the previously outlined estimates for design rainfall) is projected to increase by 11.3% and 17.7% for the near and medium terms, respectively and the total annual rainfall is projected to increase by 3.9% based on the near term predictions and 1.1% based on the medium term predictions.

In the case of the Niger River, increased peak flows are anticipated with increased extreme rainfall. Given the large area of hydrological relevance of approximately 17,120 km^2 of Niger catchment, an area weighted increase in streamflow of 9.6% and 14% is estimated for the near and medium term respectively.

5.4 Infrastructure

The town of Kouroussa is located 7 km northeast of the Project and is the capital of the Kouroussa Prefecture. Kouroussa has markets, schools, hospitals, pharmacies, hotels and 4G cellular signal. The local industry around Kouroussa is predominantly subsistence and cash crop farming, producing cotton, rice, millet, groundnuts, and vegetables. Kouroussa itself is a river port on the Niger River for small fishing vessels. There is a long history of small-scale artisanal gold mining in the region.

PDI has existing exploration facilities, including an accommodation camp, offices and a core shed. The existing accommodation camp has approximately 60 beds (with the potential to incorporate additional beds) plus supporting facilities, including two mess buildings, two laundry buildings, the site medical clinic, security, water wells and two generators to power the site. Once the Project is operational, it is expected to continue to be used for exploration activities, possibly including a new core shed facility. The core shed requires relocation from its current location for continued use in exploration, as it is within the proposed NEB waste dump footprint.

Sufficient land exists across the Project area for development of the required infrastructure when taking into account topography, surface water, geotechnical, environmental and social considerations.

Power availability in the region from gird sources is sparse although work is currently being completed on the Mali interconnector which will run approximately 30 km from the Project area. As such, power for the Project will be generated through a heavy fuel oil (HFO) power plant in combination with a solar farm to be located to the southwest of the process plant.

Water for local use is typically sourced from groundwater bores and PDI will need to be self-sufficient with its water supply for the Project. Raw water supply to the Project will be via a combination of ground dewatering bores installed around the NEB and GBE pits along with water harvested from the TSF. There will also be two bores dedicated to water supply at the accommodation village. Grey water and black water will be processed in a bacterial septic system. Most local village houses have an individual septic system or simply a pit latrine.

Unskilled labour is readily available in Guinea for the Project. Skilled labour for open pit mining operations, gold processing operations and general and administrative operations is also readily available in Guinea due to the established gold, and other commodities, mining in the country. Skilled labour for underground mining operations will be rare in Guinea as there is little underground mining in the country. Taking these factors into account, the labour cost estimates have included sufficient expatriate staff, along with the relevant on-costs, to secure the skills required for the successful operation of the Project. In addition, the use of contract mining services has been included as it will de-risk the supply of mining operations labour from local sources.

Accommodation for the Project will be predominately in Kouroussa although an accommodation village is included for the expatriate and senior staff located approximately 8.5 km from the processing plant on the northern side of the N1 highway. The accommodation village size will be sufficient for the workforce requirements and will be expanded with temporary construction facilities for construction.

5.5 Site Layout

The project layout includes:

- Open pit mines at NEB, GBE and BC with associated waste rock dumps (WRDs) at NEB and BC.
- Underground mine underneath the NEB pit with access from the GBE pit and the associated ventilation, cooling and paste plant infrastructure.
- Processing facility co-located with administration and maintenance facilities and power station.
- Solar farm for renewable power generation.
- Tailings storage facility (TSF).
- Accommodation village.
- Access roads and security infrastructure.

The facilities are described further in this report and the overall site layout is provided in Figure 5.5

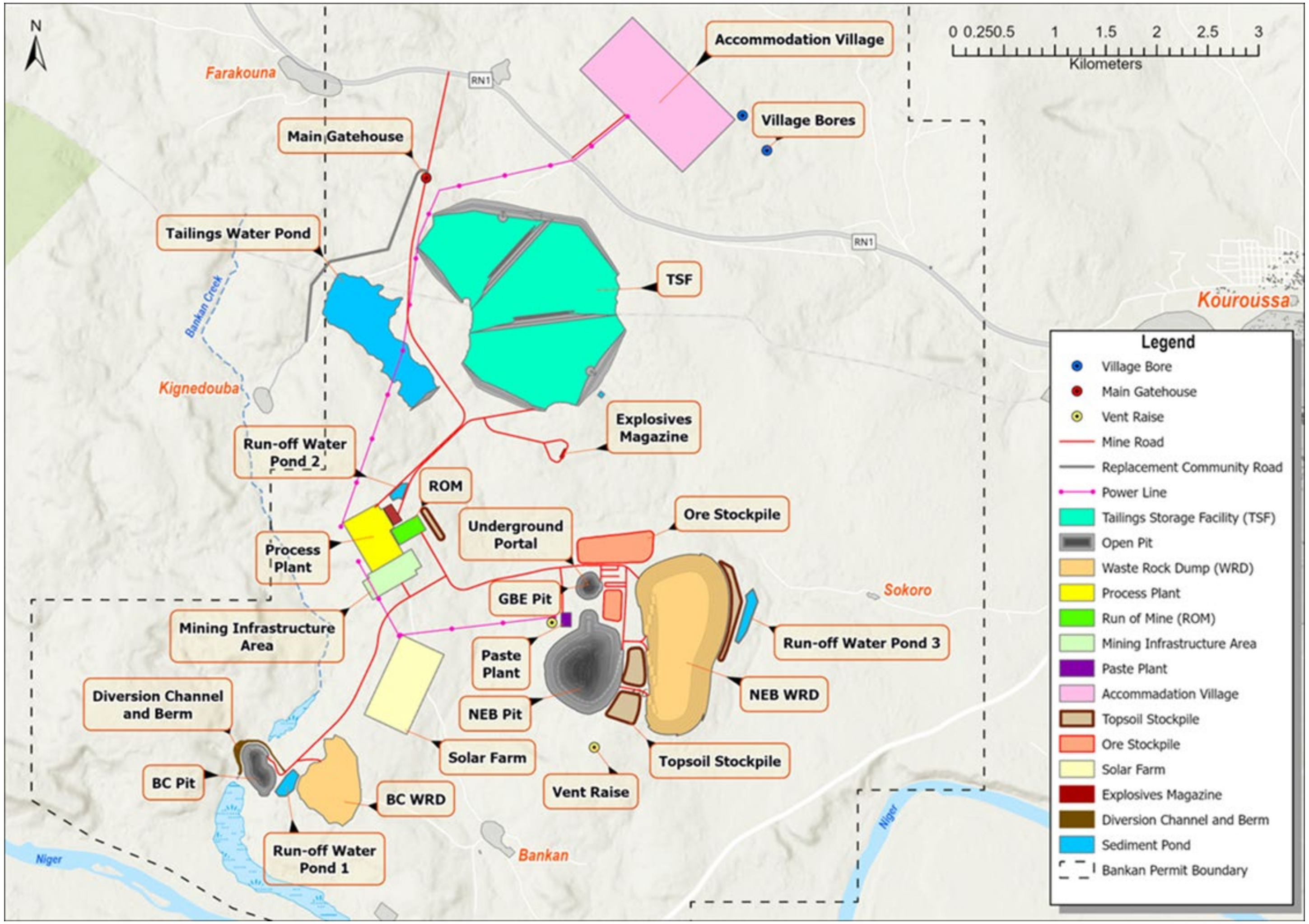

Figure 5.5: Overall Site Layout

6 HISTORY

In late 2018, the Kaninko area was highlighted by PDI during its terrain-scale assessment of the Siguiri Basin, which PDI had previously identified as being both highly prospective for gold mineralisation and underexplored.

Field visits identified widespread artisanal workings consisting of extensive pitting into weathered bedrock with shallow surficial workings in lateritic cover material extending for hundreds of metres away from the pitted areas, in what were later to be identified as the NEB and BC deposits.

PDI began field work with BLEG stream sediment geochemistry, rock chip sampling and geological mapping around the artisanal sites at BC. The first stage samples comprised twelve vertical channel samples from active artisanal mine sites. Seven samples from workings in strongly weathered bedrock returned an average value of 1.5 g/t Au with a peak value of 4.6 g/t Au.

A second program of systematic channel sampling of saprolite exposures comprised 132 samples collected from artisanal mine dumps and exposures. At BC, 49 mine dump samples were collected from an exposed saprolite zone 230 m long and up to 60 m wide, with an average grade of 0.8 g/t Au.

Twenty dump samples were also collected over a 5 ha area of surficial workings at what became the NEB prospect, and 11 samples collected from surficial workings over a 3 ha area at Bankan East.

In early 2020, a program of 3,178 m of shallow power auger drilling and 490 lineal metres of trenching was completed at NEB and BC, with mineralisation identified across a broad zone with bottom of hole samples up to 11.9 g/t Au at NEB. This program was followed up by an aircore and reverse circulation drilling program, with further auger drilling extending the strike length of NEB.

A maiden mineral resource estimate was completed for the Project in September 2021 comprising an Inferred Mineral Resource of 72.8 Mt at 1.56 g/t Au for 3.65 Moz of contained gold. On the 1st of August 2022 additional drilling was used to update the Inferred Mineral Resource estimate to 79.5 Mt at 1.63 g/t Au for 4.2 Moz of contained gold.

Based on an infill drilling program through the second half of 2022 the mineral resource estimate confidence was improved and an updated mineral resource estimate was announced on 6 February 2023 including an Open Pit Indicated Mineral Resource of 42.7 Mt at 1.27 g/t Au for 1.75 Moz contained gold at NEB along with a further Open Pit Inferred Mineral Resource of 24.7 Mt at 2.23 g/t Au for 1.77 Moz contained gold and an Underground Inferred Mineral Resource at NEB of 2.2 Mt at 4.75 g/t for 335 koz contained gold. An Inferred Mineral Resource of 7.2 Mt at 1.42 g/t for 331 koz of contained gold was announced for the BC deposit.

Continued drilling at the Project led to an announcement on 7 August 2023 an increase to these mineral resources to an estimated 100.5 Mt at 1.66 g/t Au for 5.4 Moz of contained gold with approximately 77% being in the Indicated Mineral Resource Category. Based on this mineral resource estimate, PDI completed a pre-feasibility study (PFS) which included the announcement of a maiden Mineral Reserve for the Project, consisting of open pit and underground ore from NEB and open pit ore from BC, of 57.7 Mt at 1.64 g/t for 3.05 Moz of contained gold. All Mineral Reserves were in the Probable Mineral Reserves category.

Other than artisanal scale gold mining, which is not material to the Mineral Resources or Mineral Reserves, there has been no production from the Property.

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Project Geology

The Bankan project area is largely covered by lateritic duricrusts and is deeply weathered. Outcrops are sparse, and the underlying bedrock geology is known largely from regional scale geophysics and drilling completed by PDI.

The Project area comprises four exploration permits, and is located in an area of greenstones near the southwest margin of the Siguiri Basin, surrounding the intersection of NNW striking and a NW striking structures on the margin of a regional granitic batholith as shown in Figure 7.1. Numerous anastomosing NNE striking structures have been interpreted from the aeromagnetic data. Smaller granitic intrusions in the greenstones are structurally controlled and provide evidence for significant heat and fluid flow late in the orogenic history, which in other parts of the Siguiri Basin has been demonstrated to be part of the gold mineralisation process.

At a local scale, both the NEB and BC prospects are partially hosted by these granitic intrusions. NEB has been developed at the hanging wall contact of a small tonalitic intrusion, largely structurally controlled by a NNW striking shear, which is part of a network of anastomosing NNW to NNE striking structures. Higher grades are found in and on the immediate footwall of the shear, with lower grade mineralisation in both the tonalitic footwall and the greenstone hanging wall. Mineralisation consists of wide zones of structurally controlled chlorite, silica and sericite alteration with associated pyrite and quartz veining. An interpreted structural history (Murphy, 2022) indicates E-W compression with kinematic indicators in the Bankan mylonites consistent with a reverse (oblique dextral) zone. The degree of ductile shearing suggests hundreds of meters of west over east displacement. Highest grade mineralisation is commonly ponded below and around the shear zone. Gold was deposited at an advanced stage in the deformation, overlapping with sulphide, chlorite, sericite, potassium feldspar and carbonate alteration phases. The shear zone is characterised by magnetite depletion of the overlying mafics and sediments in the hanging wall. However, magnetite occurs as clots in late-stage veins. Conglomerates in the hanging wall may relate to growth faulting, such as developed at a sub-basin position. The alteration assemblages (chlorite, sericite, epidote and sulphide) can extend high into the hanging wall sediments.

BC is hosted in the carapace of a small tonalitic intrusion, which has been intruded into a structurally complex greenstone sequence of sediments, volcanics and marbles.

In the Argo tenement, the major structures strike NE-SW between a small granitic intrusion and a granitic batholith and from the aeromagnetic data a series of tight to isoclinal SW-plunging folds in the greenstones have been interpreted.

The mineralisation at Fouwagbe has broadly developed along a main deformation corridor dipping approximately 50° to the north-west, in which three sub-parallel trends are interpreted. This deformation corridor appears to be positioned along a fold axis and is hosted by a felsic formation and is characterised by brecciated, foliated quartz veining with traces of sulphides. Along these north-west dipping trends, mineralisation is present in flattened shoots which plunge to the south-west. The structural setting and host lithology are both unclear due to exceptionally deep weathering, up to 200m deep from the current natural surface. At Sounsoun, an E-W trending shear zone is present in the northern part of the target area. The shear zone dips approximately 70° to the north and develops either in felsic intrusive formations or along a contact between felsic intrusive rocks and mafic volcanic rocks, with pyrite as the main sulphide and silica-chlorite alteration. Mineralisation appears to be preferentially developed along this E-W shear zone.

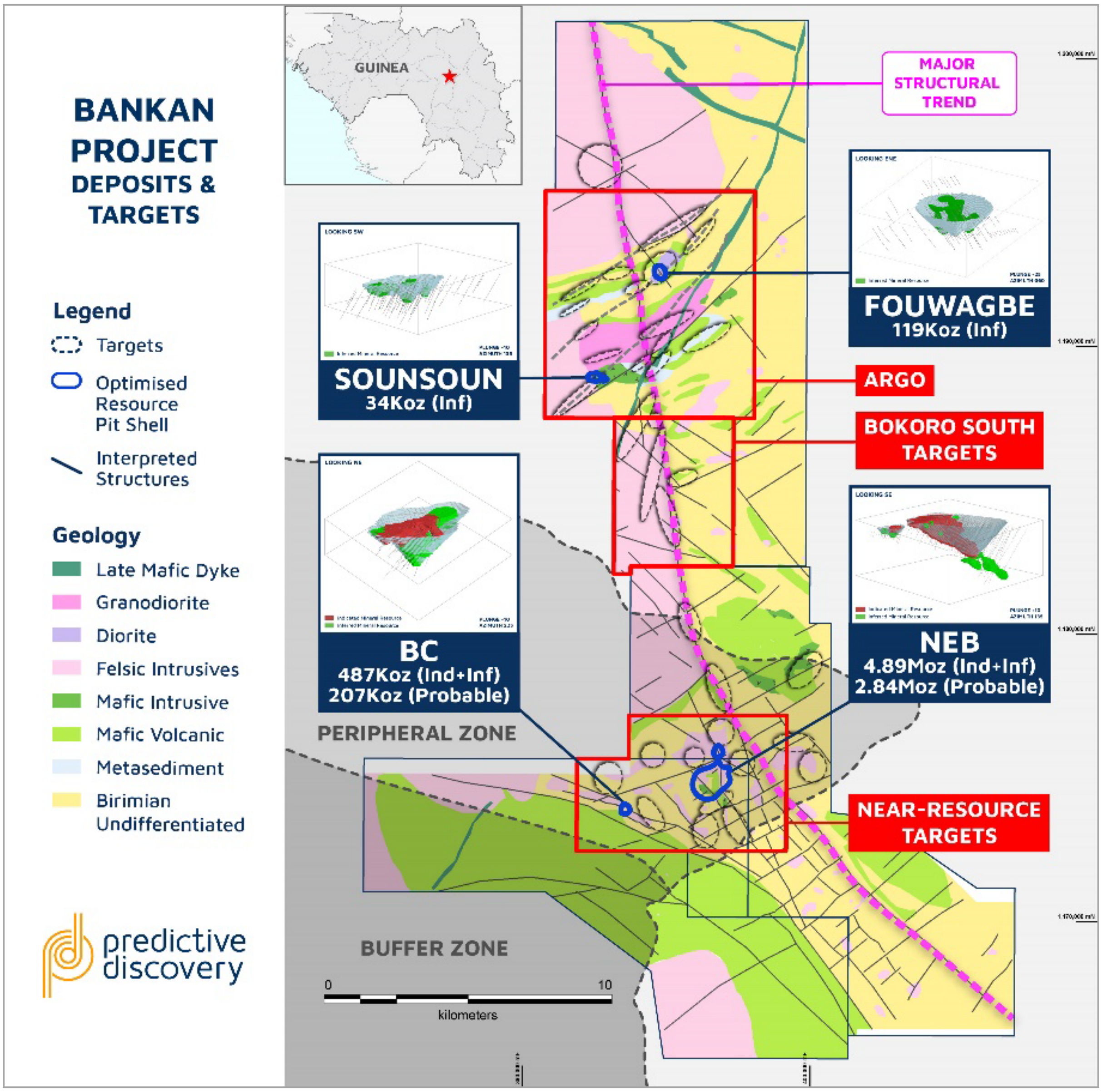

Figure 7.1: Bankan Project Interpreted Geology, Resources and Targets (PDI 2024)

7.2 Lithology and Weathering

The NEB tonalite is a coarse-grained quartz-plagioclase-K feldspar-hornblende-biotite intrusive. Minor accessory phases include muscovite and sericite; alteration minerals include carbonate, epidote, clinozoisite. The plagioclases may be extensively albitised.

The mafic sequence comprises a mixed basalt, andesite and gabbro assemblage, and is dominated by plagioclase-quartz-chlorite, with accessory biotite and hornblende; carbonate alteration and pyrite mineralisation are common.

The BC skarn is a dominated by amphibole-epidote/clinozoisite-K feldspar, with accessory plagioclase, quartz and clinopyroxene.

The Bankan project area has been subjected to long lived weathering during the Neogene (24-3 Ma), which dissected an older weathering surface and produced three lateritic pediment systems (Chardon, Grimauld, Beauvaise, & Bamba, 2018). These pediments are extensive, gently inclined slopes of transportation or erosion that truncate regolith and connect eroding slopes or scarps to areas of sediment deposition or alluvial transportation at lower levels.

The typical complete lateritic weathering profile shown in Figure 7.2 (Chardon, Grimauld, Beauvaise, & Bamba, 2018) comprises:

- Cemented ferricrete layer, composed of insitu or transported ferruginous concretions in a ferruginous matrix.
- Mottled clay layer, composed of variably ferruginous residual clays formed by intense weathering and consequent profile collapse.
- Saprolite zone, composed of highly weathered bedrock, where there has not been sufficient leaching to initiate profile collapse, original rock textures are recognisable even though most original rock forming minerals have been weathered to clays. There may be a transition at the base of the saprolite zone to the fresh zone, where weathering is either patchy or restricted to favourable structures with greater than 40% fresh rock defines this as the saprock zone.
- Underlying essentially un-weathered fresh zone.

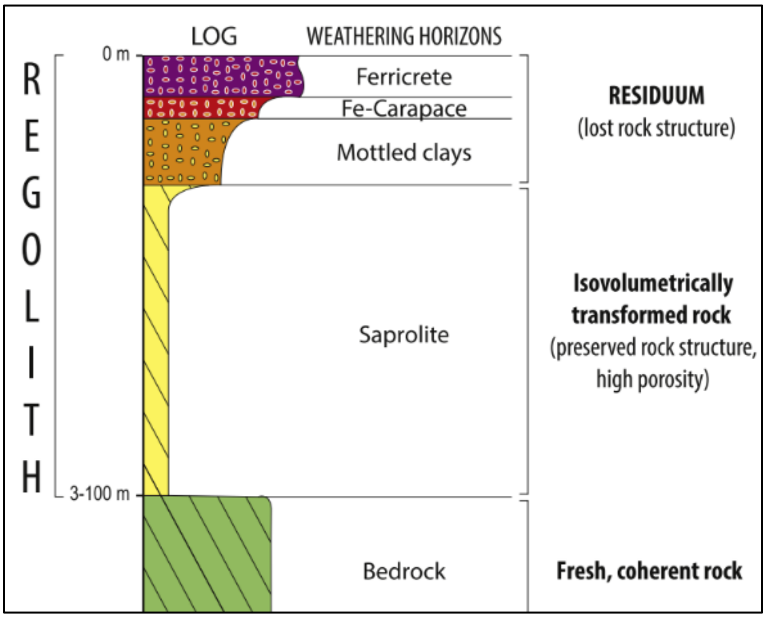

Figure 7.2: Typical Lateritic Weathering Profile (Chardon, Grimauld, Beauvaise, & Bamba, 2018)

In gold deposits, profile collapse and lateral dispersion of gold by humic or chloride complexes in groundwater may form extensive sheets of mineralisation in the ferricrete or mottled zones above or adjacent to bedrock deposits. In the saprolite zone, some lateral dispersion is possible above the redox front.

The complete laterite profile is preserved at NEB, which is under a ridge capped with resistant ferricrete. At BC, recent erosion has incised the currently active river valley, and mottled zone and saprolite are largely exposed at the surface of the artisanal workings, with a thin veneer of transported soil and alluvium elsewhere; a few small patches of remnant ferricrete have been identified. Examples of the weathering profile types are presented in Figure 7.3 to Figure 7.7.

At Fouwagbe, the saprolite zone is much thicker than at Bankan, reaching more than 150m below the natural surface.

Figure 7.3: BNED0087 Laterite Zone

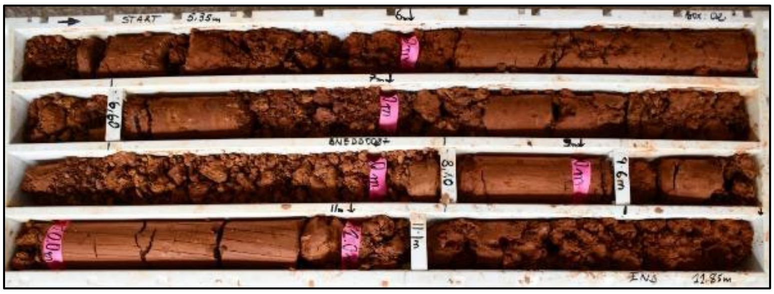

Figure 7.4: BNEDD0087 Mottled Zone

Figure 7.5: BNEDD0087 Saprolite Zone

Figure 7.6: BNEDD0087 Contact Between Saprolite Zone (upper) and Saprock Zone (Lower)

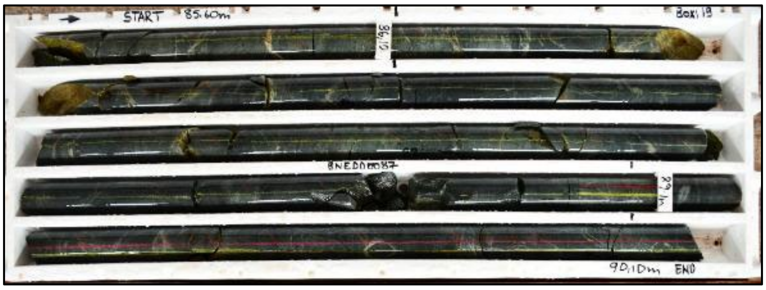

Figure 7.7: BNEDD0087 Fresh Zone

To test for the potential effect of supergene processes on the grade distribution, in 2021 the insitu mineralised composites at NEB were classified by weathering using the base of transported digital terrain model. The saprolite composites were compared statistically with the fresh composites in in the 30 m immediately below the base of weathering. The comparison in Table 7.1 shows that the means and medians of the composites above and below the base of weathering are virtually identical and visual inspection of the Leapfrog grade shells also suggests that there is no significant lateral dispersion above the base of weathering.

Table 7.1: NEB Saprolite versus Fresh Mineralised Composites, Au g/t Statistic

	Saprolite	Fresh
Count	415	559
Minimum	0.027	0.0025
Maximum	26.39	67.78
Mean	1.11	1.10
Median	0.57	0.54
Standard Deviation	1.87	3.23
CV	1.68	2.95

Diamond core through the weathered profile (i.e. core from surface) has been logged according to the rock mass description from Barton (Barton, 1978) which assigns a semi-quantitative code that can be related to a range of unconfined compressive strengths (UCS) of rocks as per Table 7.2.

Table 7.2: Field Rock Strength Codes, Empirical Tests and UCS Strengths (Barton, 1978)

Code	Description	Field Identification	Range of UCS
S1	Very soft clay	Easily penetrated several inches by fist	<0.025
S2	Soft clay	Easily penetrated several inches by thumb	0.25-0.05
S3	Firm clay	Can be penetrated several inches by thumb with moderate effort	0.05-0.10
S4	Stiff clay	Readily indented by thumb, but penetrated only with great effort	0.10-0.25
S5	Very stiff clay	Readily indented by thumbnail	0.25-0.50
S6	Hard Clay	Indented with difficulty by thumbnail	>0.50
R0	Extremely weak rock	Indented by thumbnail	0.25-1
R1	Very weak rock	Crumbles under firm blows with point of geological hammer; can be peeled by a pocket knife	1-5
R2	Weak rock	Can be peeled by a pocket knife with difficulty; shallow indentations made by firm blow of geological hammer	5-25
R3	Medium strong rock	Cannot be scraped or peeled with a pocket knife; specimen can be fractured with single firm blow of geological hammer	25-50
R4	Strong rock	Specimen requires more than one blow of geological hammed to fracture it	50-100
R5	Very strong rock	Specimen requires many blows of geological hammed to fracture it	100-250
R6	Extremely strong rock	Specimen can only be chipped with geological hammer	>250

The relative proportions of these logged codes by weathering zone, shown in Figure 7.8, suggest that the ferricrete and mottled zones are largely clay, with a small proportion of rocky materials; this is especially the case in the main ridge at NEB. By contrast, the saprolite and saprock are much more dominated by rocky materials, with the fresh zone composed of strong rock (R4) or harder.

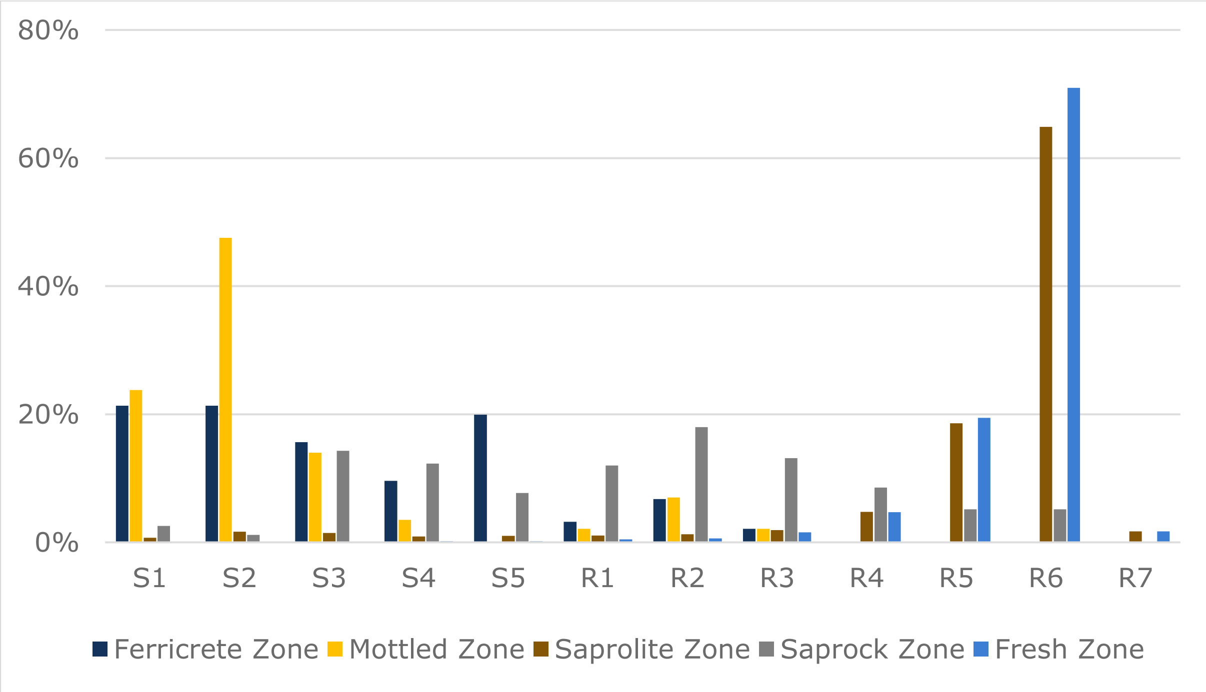

Figure 7.8: Proportions of Rock Strength Codes by Logged Weathering

7.2.1 Alteration and Mineralisation

Vein and alteration paragenesis record a history of early ductile through to late brittle events, mostly at greenschist facies conditions. Three vein types have been recognised:

- A-type veins formed relatively early, at a high angle to the foliation and are interpreted as related to foliation boudinage. Once formed, they underwent significant shearing, folding and dislocation. They are composed of quartz and minor feldspar, with chlorite and sericite. Sulphides are a minor component.
- L-type veins developed as shear veins along the foliation, and as deformed A-type veins, and are laminated quartz-chlorite veins.
- T-type veins formed during tensile failure, occur at a variety of orientations, with a dominant set at high angles to the foliation. Composed of mainly quartz and chlorite, they range from early quartz and chlorite infill to chloritic microfractures. Open space filling (i.e. dog tooth textures) are suggestive of epithermal textures. They are typically associated with pyrite and chalcopyrite.

Three alteration styles have been recognised:

- "Red rock" alteration is widespread in the granitic protolith, comprising K-feldspar and hematitic alteration. There is a clear association of potassic alteration linked to alteration fluids that are fed by T-type fractures. Hairline fractures can generate wide alteration haloes.
- "Green rock" alteration is pervasive sericite alteration, which can be texturally destructive of pre-existing foliation or more passively replace the foliation.

- "Black rock" alteration forms the bulk of the mineralised system. This is an extensive overprint by chlorite, silica, K-spar and sulphide mineralisation (pyrite, chalcopyrite and locally magnetite).

Much of the alteration occurred late in the deformation history. Pyrite pressure shadows indicate an overlap in timing of potentially mineralising fluids with the shearing

Sulphide mineralisation largely comprises pyrite with minor chalcopyrite. In the altered felsic igneous rocks, the sulphide mineralisation is generally associated with the later stage veining, with minor amounts disseminated through the rock texture. In NEB, higher grade ore is characterised by higher pyrite and covellite, and arsenopyrite and sphalerite while low grade ore lacks covellite, galena, sphalerite, and bismuth species (Table 7.3). Other sulphides that have been noted include tennantite-tetrahedrite, hessite, gersdorfitte, bornite and cobaltite. The bismuth phases are generally found as inclusions within pyrite, but also as local clusters within the gangue material nearby to pyrite or veining structures.

Fewer samples have been analysed from BC with no diagnostic sulphide species having been identified, although the assemblage is similar to that at NEB (Table 7.4).

Table 7.3: NEB Sulphide Species Distribution by Grade

Grade Range	Au	Py	Cv	Po	Cp	As	Gn	Sp	Bi species
>10g/t	Trace	Accessory	Accessory	Trace	Trace	Trace	Trace	Trace	Trace
2-10g/t	Trace	Trace	Trace		Trace		Trace		Trace
0.3-2g/t	Trace	Trace			Trace				
0.1-0.3g/t		Trace			Trace				

Table 7.4: BC Sulphide Species Distribution by Grade

Grade Range	Au	Py	Cv	Po	Cp	As	Gn	Sp	Bi species
>10g/t	Trace	Trace			Trace		Trace		
2-10g/t	Trace	Trace			Trace		Trace		
0.2-2g/t	Trace	Trace		Trace	Trace		Trace		

7.2.2 Gold Deportment

Free gold is commonly found in mineralised samples, predominantly associated with sulphides. The most common grain sizes noted are in the 5 µm to 30 µm range, as shown in Figure 7.9, and grains have varying silver content from near pure gold to electrum; the typical Ag content from scanning electron microscope (SEM) scans is 10%. Examples of the types of gold grains are shown in Figure 7.10, showing gold inclusions in sulphides with paler gold having higher silver content, and Figure 7.11, showing gold locked in quartz (left) and on silicate grain boundaries (right).

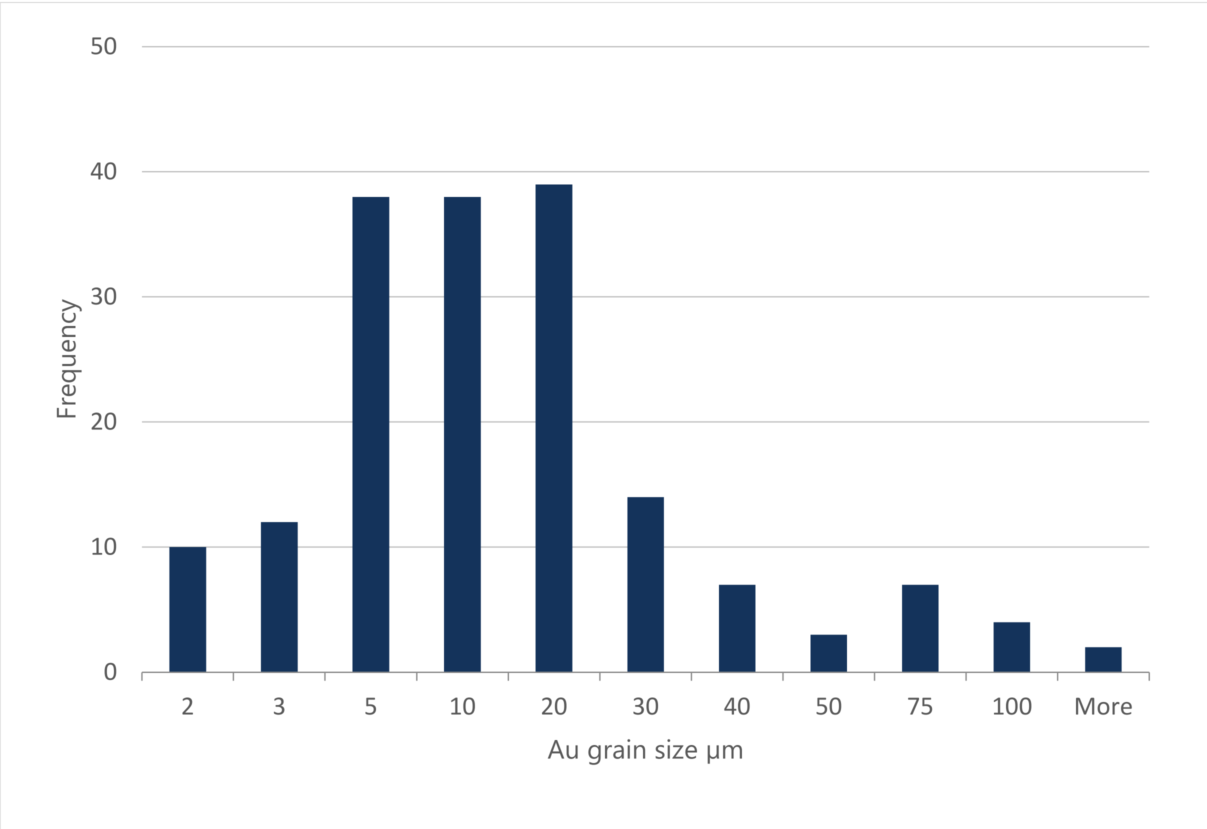

Figure 7.9: Gold Grain Size Distribution from 174 Individual Grains

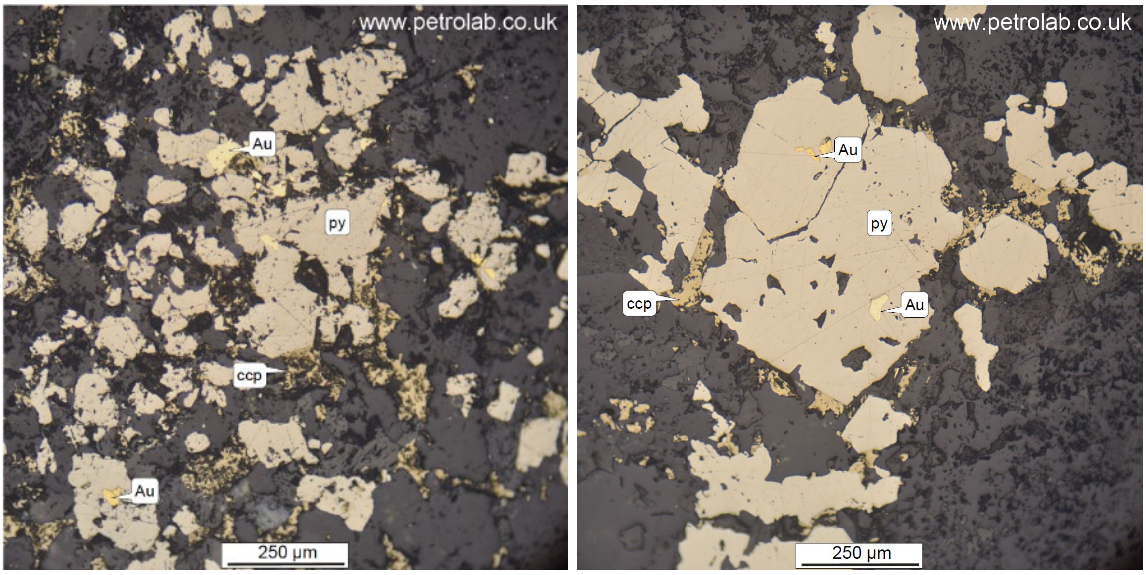

Figure 7.10: Left: BNEDD0088 325.6m; Right: BNERD0073 30m

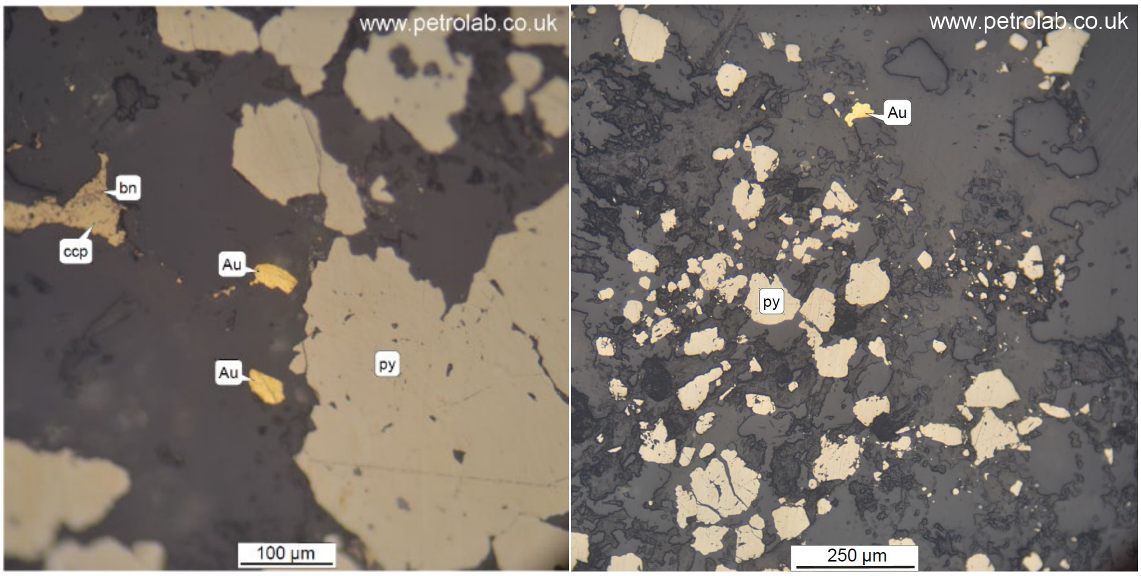

Figure 7.11: Left: BNEDD0106B 637.02m; Right: BNERD0107 553.42m

7.2.3 Structure

The mineralisation at NEB is structurally controlled by a network of dextral shear zones. The largest and most persistent is the main shear zone (STMZ), a structure dipping approximately 40° to the west, and which has been intersected over a strike length of at least 800 m and 1,150 m down dip. The STMZ it is open at depth and along strike to the south and typically consists of a zone of shearing, strong mylonite fabric and sericite alteration, often with significant quartz veining; it is at or just above the hanging wall contact of the main tonalite intrusion (Figure 7.13).

The STMZ is typically a single mylonite zone with associated alteration ranging from 4 m to 7 m thick, but may be up to 36 m or comprise up to four separate mylonite zones, which is likely to represent a zone of anastomosing structure. In the footwall, a very well developed second order shear 3 m to 5 m thick (STSZ01) has a very similar structure and alteration to the STMZ, and forms a step over or jog

(Micklethwaite, Sheldon, & Baker, 2010) from the STMZ to a more weakly developed structure; and hence is a locus for dilation and fluid flow associated with mineralisation.

The STSz01 nearly outcrops, whereas the STMZ terminates below the surface above its intersection with STSZ01. This fault duplex is interpreted to represent a soft-linked overlapping shear system, where a component of strain is accommodated by rotation or folding between the main bounding shear segments, as well as at the termination of the segments. This relationship can also be seen at a smaller scale, for example in the close-spaced grade control drilling pattern. Interpretations of the higher and lower grade parts of the mineralisation show that at a scale an order of magnitude smaller than the deposit, the same overlapping shear segments can be interpreted.

Structural measurements from the diamond core also show this relationship. Foliations are strongly clustered in an orientation parallel to the Main Shear (-42° to 271°), whereas the veins have a much greater scatter, but the maximum density is at an orientation of -36° to 269° i.e. parallel to the interpreted step over structures. Bedding planes and lithological contacts also show the general 40° westwards dip parallel to the foliations.

Limited petrography has identified shear fabrics, rotated augen of relatively undeformed rock, and pressure shadows on both the rotated augen and the sulphides. These relationships are likely to be reproduced on other scales from microscale to mesoscale (e.g. outcrop) to macroscale (e.g. deposit scale controls). Examples of the microscale structures are presented in Figure 7.12.

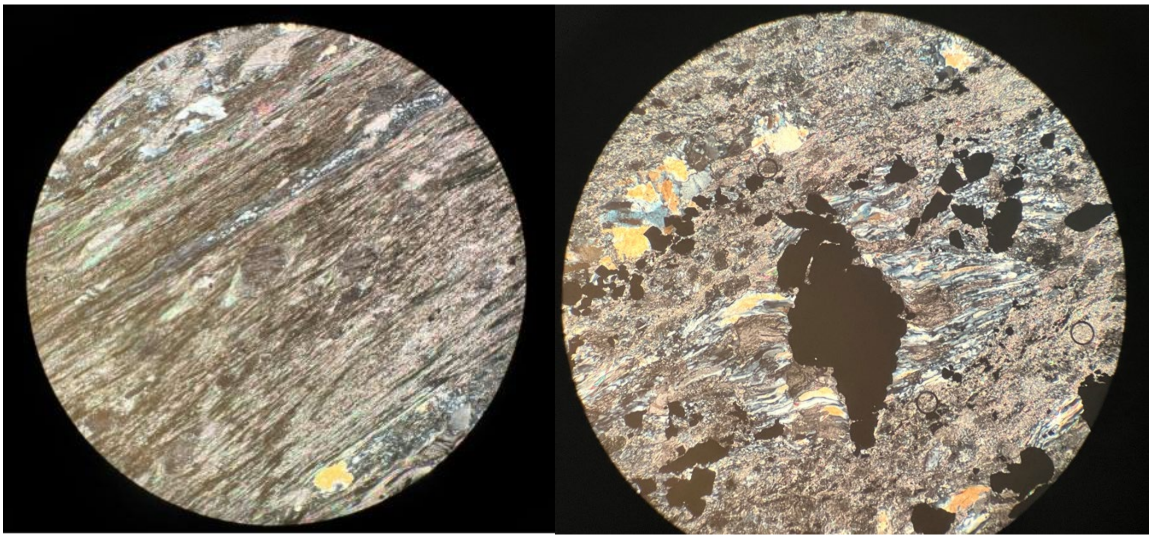

Figure 7.12: Left: BNERD0098 386.72m, Rotated Augen; Right: BNERD074 318.65, Pressure Shadows around Pyrite

Below the STSZ01 shear, four other parallel structures have been interpreted, with similar relationships to the STMZ; these however are less well constrained by drilling and hence have a greater degree of uncertainty in their location and extent.

Late stage faulting has affected the deposit, however in general there are no clearly defined fault offsets that are interpretable from the drilling data. One exception is a major ENE-WSW, steeply dipping fault that is interpreted from geophysical data and appears to sinistrally offset the Gbenbeden area from the main NEB deposit, informally named “Norm’s Fault”. Fault measurements and shear planes have the same ENE-WSW strike but in general have much shallower dips.

A set of plans showing the geology and structural interpretations are shown in Figure 7.14 to Figure 7.17 with NEB, GBE and 800W shears shown in light green, Norms Fault shown in light blue, tonalite shown as pink and metasediment shown as brown.

The structural controls for BC are much less well known. Since the publishing of the previous resource model, a relogging exercise has clarified the lithology, especially of the intrusion where some intervals had been previously mis-logged as extrusive or skarn lithologies. From the drillhole logging, two shears have been interpreted, a major one dipping moderately to the southwest and a second order structure dipping moderately to the northeast; these appear to constrain both the small tonalite intrusion and the mineralisation that is localised in the carapace of the intrusion. Foliations generally dip parallel to the major shear, whereas the veins have several preferred orientations, as well as a greater scatter than the veins at NEB. Bedding planes and contacts broadly dip parallel to the foliations and shears.

Figure 7.13: BNEDD0086 Main Shear 320.0-340.1m, 20.1m @ 0.48g/t

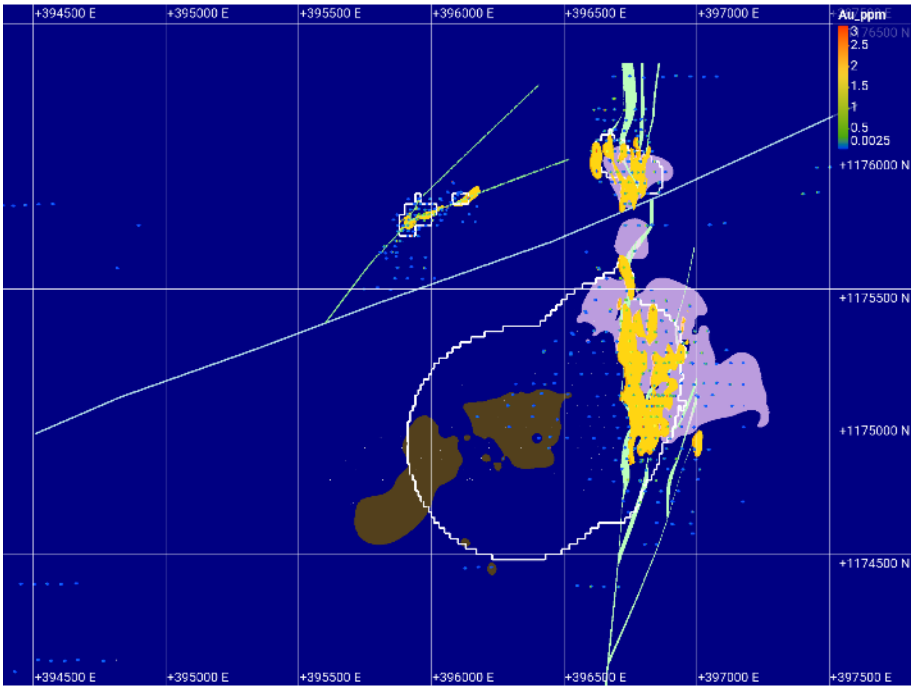

Figure 7.14: 10350mRL Geology Interpretation

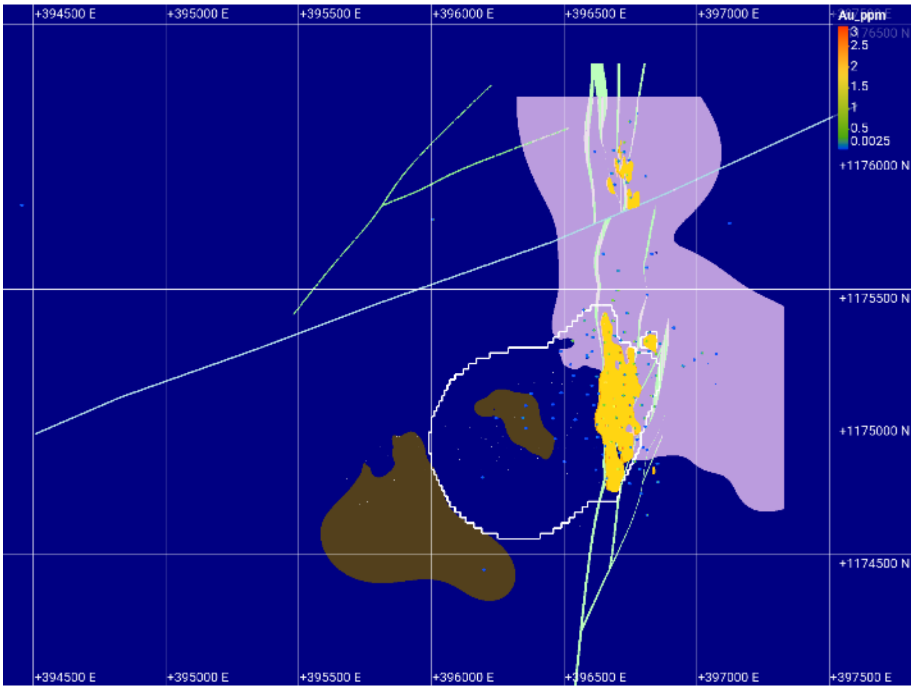

Figure 7.15: 10250mRL Geology Interpretation

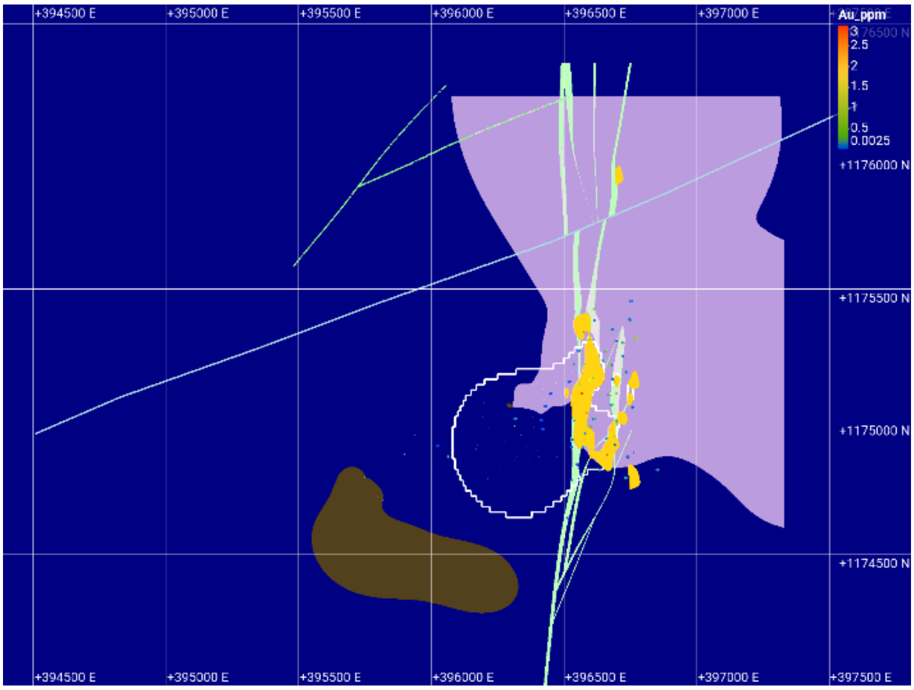

Figure 7.16: 10150mRL Geology Interpretation

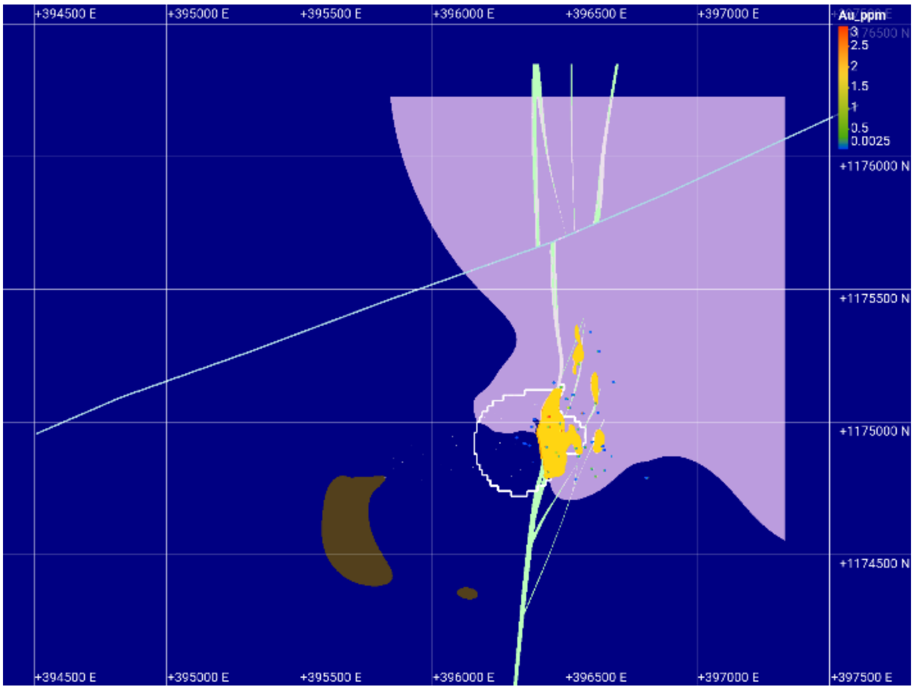

Figure 7.17: 10050mRL Geology Interpretation

8 DEPOSIT TYPES

The Siguiri Basin of upper Guinea and southwest Mali (Figure 8.1) is a Paleoproterozoic volcano-sedimentary basin, and part of the Birimian Supergroup which hosts most of the gold deposits of West Africa. Known major gold deposits in the Siguiri Basin include Siguiri (Anglogold Ashanti), Lefa (Nordgold), Kiniéro (Robex), Kouroussa (Nioko Resources), and Bankan.

The Siguiri Basin is interpreted to have been a marine platform, with a lower turbiditic sequence of sandstones with subordinate black siltstone and an upper sequence of limestones and acidic volcanics. The rocks have been metamorphosed to greenschist facies. Late orogenic felsic intrusive plutons and intrusions are found throughout the basin.

The gold deposits are orogenic lode deposits, temporally and spatially related to structures formed during the Eburnean Orogeny between 2200 Ma and 2088 Ma. The Eburnean orogeny was initiated with closure of the basin, amalgamation of the Paleoproterozoic island arcs, and their accretion back to the continental margin of Archean rocks. Compressional tectonics took place for about 25 to 30 million years, with widespread crustal thickening along orogen-parallel, commonly NE-trending, first-order thrust fault systems. Gold deposits mainly formed late during the Eburnean deformation.

Primary gold mineralisation in the Siguiri Basin is structurally controlled (Lahondère, et al., 1999), two mineralised structure orientations are found:

- E-W structures represented by veins attributed to tensional sinistral fractures oriented NW-SE and to the conjugated faults of NE-SW direction.
- NNE-SSW structures which are developed within a dextral normal fault.

These structures may have controlled the distribution of magmatic fluids. Magma was responsible for the formation of the granitic and dioritic intrusions and their related volcanic equivalents and constitutes the source of gold mineralisation (Lahondère, et al., 1999).

The gold-bearing brittle-ductile quartz veins, stockworks, breccias, and disseminated orebodies are located adjacent to major faults, typically in areas of second-order shears, large dilational jogs, regional fold systems, and rheological contrast. Mineralogy, alteration, structural geology, stable isotope geochemistry, and pressure-temperature conditions of gold deposition are typical of those observed in most orogenic gold provinces, such as lower greenstone to amphibolite facies metamorphism, moderate sulphide contents, sericite, chlorite and carbonate alteration and strong local shear and rheology controls on gold deposition.

Prolonged tertiary weathering has led to the formation of extensive lateritic duricrusts, and deep saprolite profiles; vertical remobilisation of the gold during the lateritic weathering is common and primary gold deposits may be overlain by lateritic or supergene gold deposits.

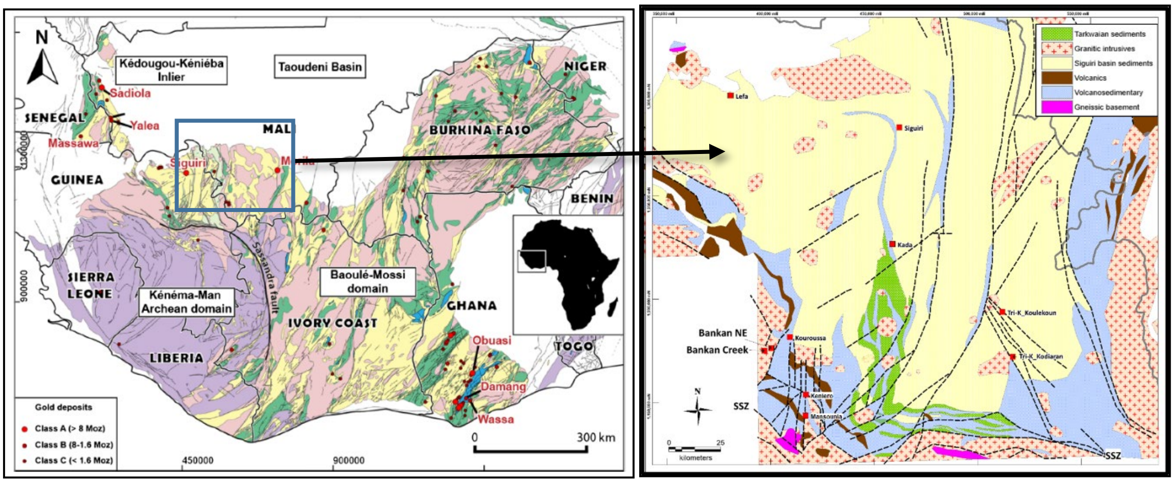

Figure 8.1: Location and Geology of the Siguiri Basin (Lebrun, Thébaud, Miller, Roberts, & Evans, 2017)

PDI's project generation strategy was to develop a broad scale structural architecture from geological and geophysical data, examining this in relation to mineral deposit distributions and host rock geology and identifying targets. Generated targets were ranked according to a series of criteria, such as fault complexity, deep-seated structure and proximity to known gold mineralisation.

Published public domain data was used to identify a north-south trending gravity gradient at the western limit of the Siguiri Basin coinciding with a possibly basin-margin related mafic volcanic suite, with the known artisanal mining at BC. First pass exploration comprised soil and auger sampling on a broad grid over identified targets to identify geochemical Au anomalies

The NEB deposit is characterised by a major jog and the lateral ENE-WSW dextral disruption of the belt basin margin and interaction of three primary structural domains as ENE-WSW (northern), WNW-ESE (southern) and N-S (cross-cutting). NEB sits at the triple junction of these three components, where the latest N-S corridor cross-cuts the older ENE-WSW and WNW-ESE domains. The BC deposit sits on the northern margin of the WNW-ESE domain.

9 EXPLORATION

Following the NEB discovery, PDI completed a series of early-stage exploration programs, including broad spaced auger drilling and a helicopter-borne magnetic and radiometric survey. The aeromagnetics identified a major 35km-long north-northwest structural corridor with the potential to host multiple "NEB style" discoveries. Structural targets identified using the aeromagnetics have been progressively followed up with power auger and aircore (AC) drilling. The strategy to date has been to undertake wide-spaced auger drilling covering the structural targets, typically 320 m by 80 m spacing, followed by closer spaced infill where encouraging gold results have been obtained (generally plus 0.25g/t composite values in saprolite to depths of around 20 m). AC drilling has then followed up the encouraging auger results, typically with pairs of scissor holes to help assess the orientation of the gold mineralisation.

Due to the deep weathering, transported cover and lack of outcropping rock, the most effective exploration methods have proved to be geophysical and geochemical vectoring, followed up by drill sampling.

9.1 Geophysics

PDI completed a comprehensive petrophysics and ground geophysics program at NEB. The petrophysics study of the NEB drill core was designed to calibrate the detailed ground geophysical orientation program. The analysis was completed by Terra Petrophysics in Perth, Australia and consisted of laboratory analysis of 31 drill core samples, selected from the geological model, to characterise magnetic susceptibility and remanence, bulk density and porosity, chargeability, Gaussian resistivity, inductive conductivity and P-wave velocity.

The ground geophysical techniques selected were gradient array induced polarisation ("GAIP") and pole dipole induced polarisation ("P-DIP"), magnetics and gravity. The objectives were to determine the geophysical signature of the NEB deposit, investigate its local geological setting, delineate possible extensions and identify near-resource target areas.

P-DIP methods proved the most effective in geophysically finger-printing the NEB deposit, with elevated chargeability (attributed to sulphide alteration) and elevated resistivity (attributed to silica alteration). Often the two responses overlap or showed marked contrast across the deposit (Figure 9.1).

The highest resistivity and chargeability D-PIP values in the central part of the survey coincided with the near-surface expression of the orebody of auger drilling contours of greater than 0.2 g/t. NEB was mapped for over 500 m in the near surface (using 2D GAIP) and deeper into the fresh rock to about 400 m using 3D P-DIP which delineated a west dipping, moderately plunging chargeability solid.

3D P-DIP modelling also contributed to extrapolation of the geological interpretation of NEB and its 3D lithological setting. The main controlling mafic-tonalite contact is characterised by a coincident density-magnetic-resistivity contrast adjacent to the chargeability shoot. Combined GAIP and P-DIP modelling provides a spatial and genetic context to the NEB gold architecture facilitating 3D drill targeting.

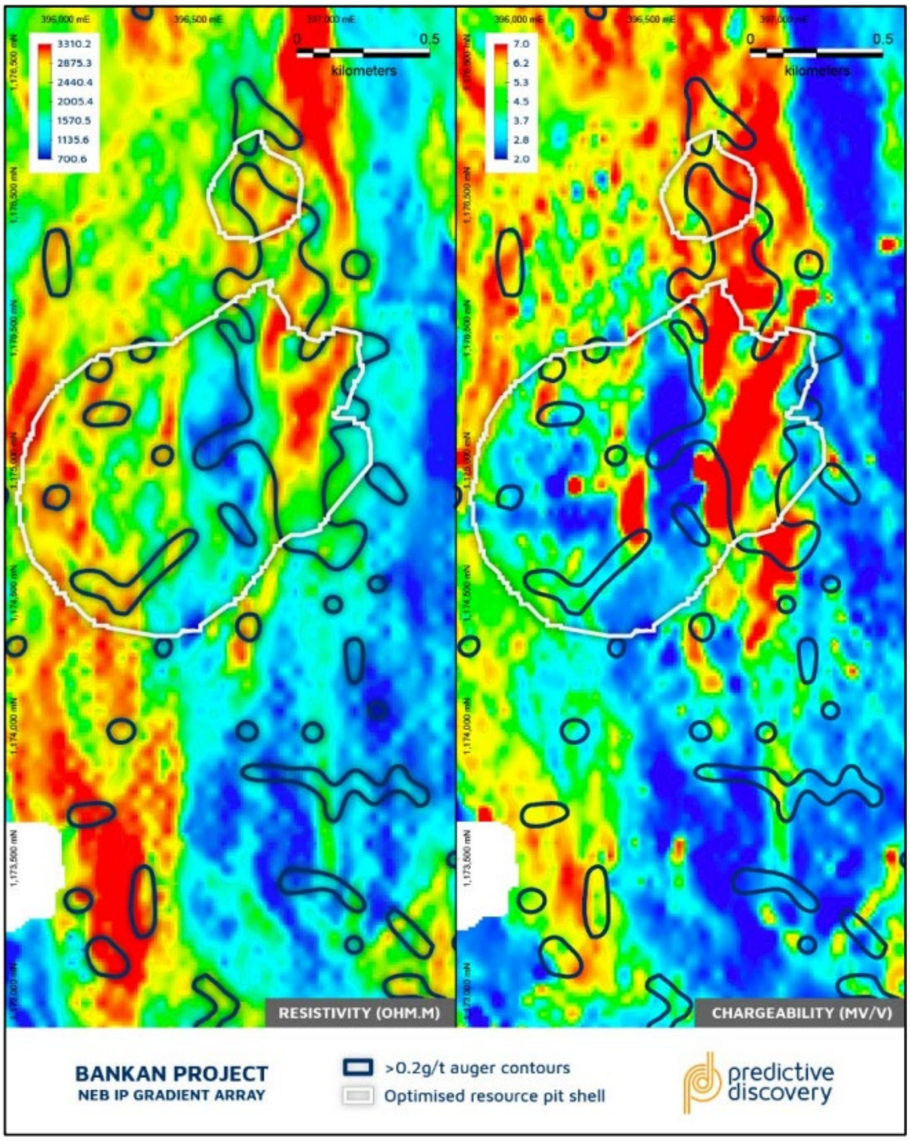

Figure 9.1: IP Gradient Array Images for NEB (resistivity left, chargeability right) Overlain with the NEV Optimised Resource Pit Shell and the >0.2 g/t Auger Anomaly Contours (PDI 2021)

9.2 Regional Exploration Targets

In the immediate vicinity of NEB and BC, numerous targets have been identified for follow up drilling (Figure 9.2), especially South Bankan and Southeast Bankan, which are probably along strike continuations of the NEB main deposit, as well as northern extensions of GBE at NEB North. Limited RC drilling has been completed but there are numerous highly encouraging aircore intersections. There may be an impact on the mine design if resource development drilling is successful.

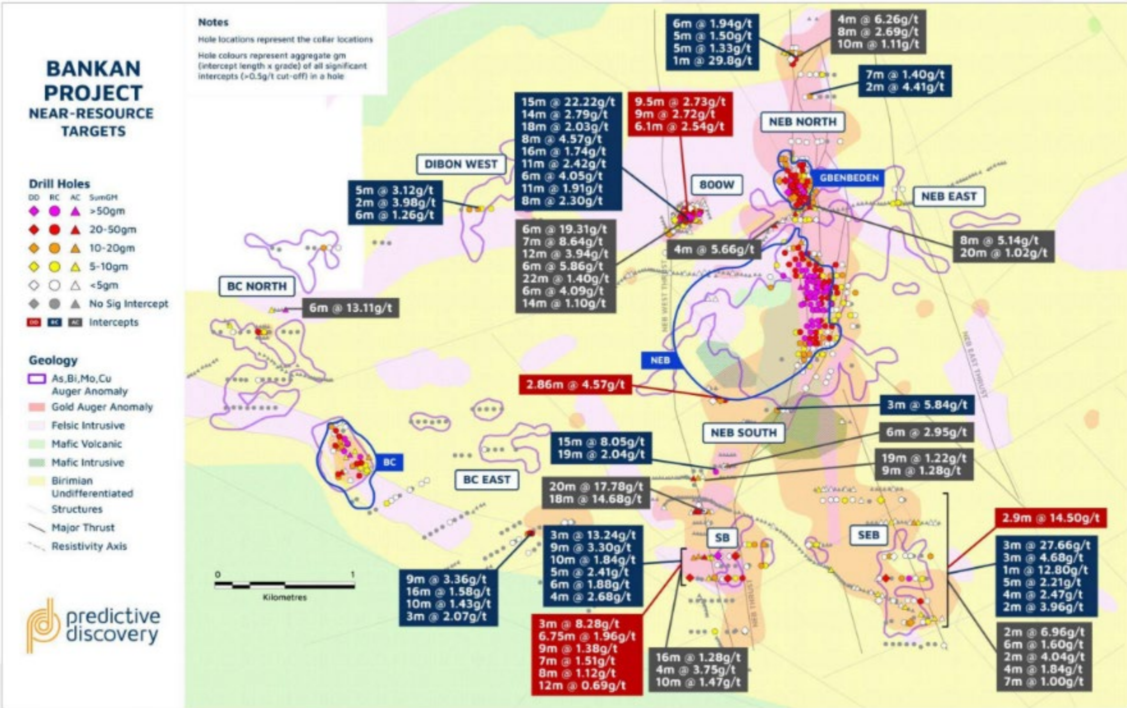

Figure 9.2: Near Bankan Targets and Drilling Results (PDI 2024)

In the Argo area (Figure 9.3) mineral resources have been identified at Fouwagbe and Sounsoun. In addition, significant results have been received from:

- Sanifolon South (12m @ 6.29g/t, 2m @ 41.71g/t, 7m @ 3.28g/t and 10m @ 1.09g/t) from AC drilling in the same corridor as Fouwagbe.
- Sinkoumba (8m @ 3.65g/t, 5m @ 5.55g/t, 4m @ 6.87g/t, 2m @ 10.90g/t and 5m @ 3.70g/t) also in the same corridor as Fouwagbe.
- Tindini (14m @ 1.97g/t and 4.95m @ 3.00g/t); and Somo (12m @ 4.82g/t).

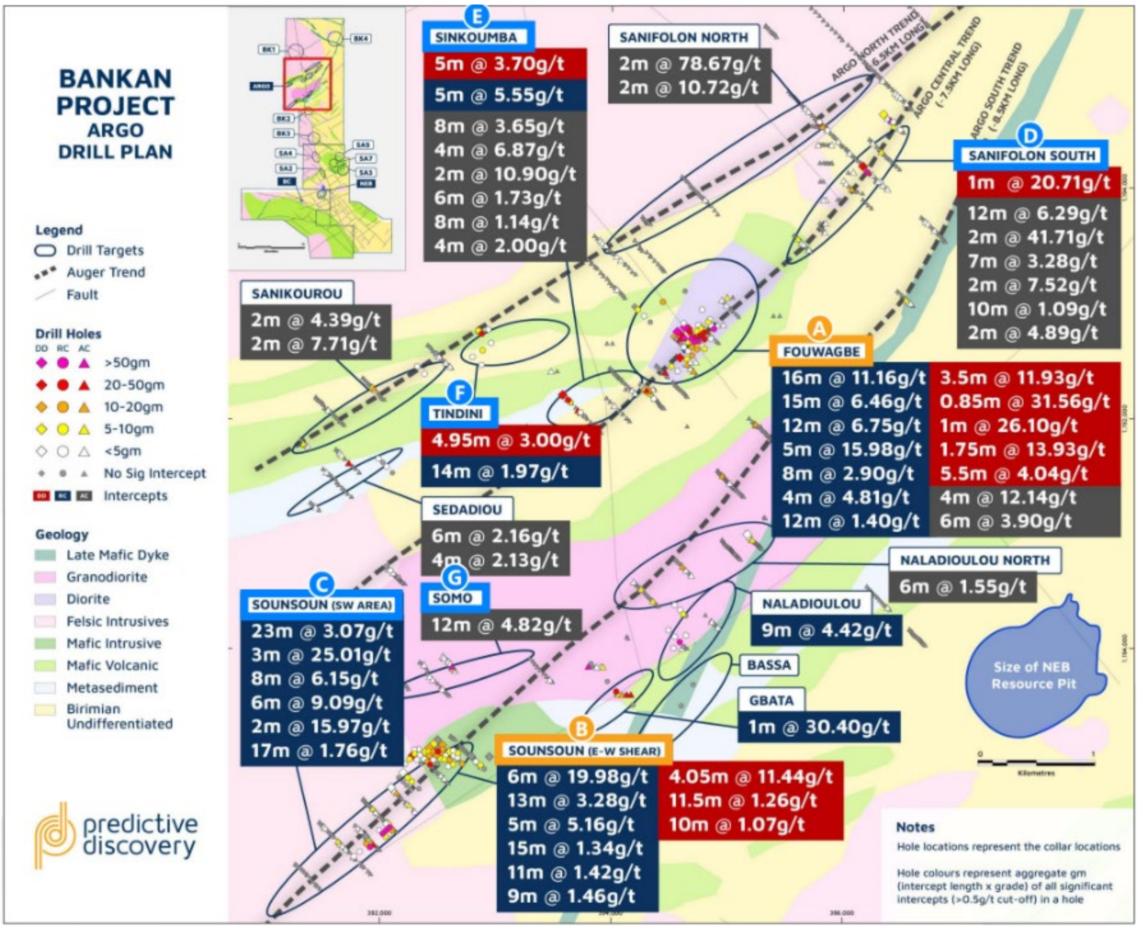

Figure 9.3: Argo Targets and Drilling Results (PDI 2024)

At Bokoro, extensive auger drilling and a first pass AC drilling program was completed to test structural and geophysical targets. The priority targets (Figure 9.4) are:

- BK7, a 850m x 150m NNE trending auger anomaly, potentially up to 2.5km long subject to completing auger grid. Initial AC drilling results of 10m @ 3.33g/t and 8m @ 1.05g/t appear to define two mineralised zones.
- BK2, a strong auger anomaly with initial AC drilling results of 2m @ 7.41g/t and 2m @ 2.20g/t.
- BK8, initial AC drilling have identified a potential 1.7km NNE trend. Best intercepts of 2m @ 2.99g/t, 2m @ 1.95g/t, 6m @ 0.51g/t and 2m @ 1.41g/t.

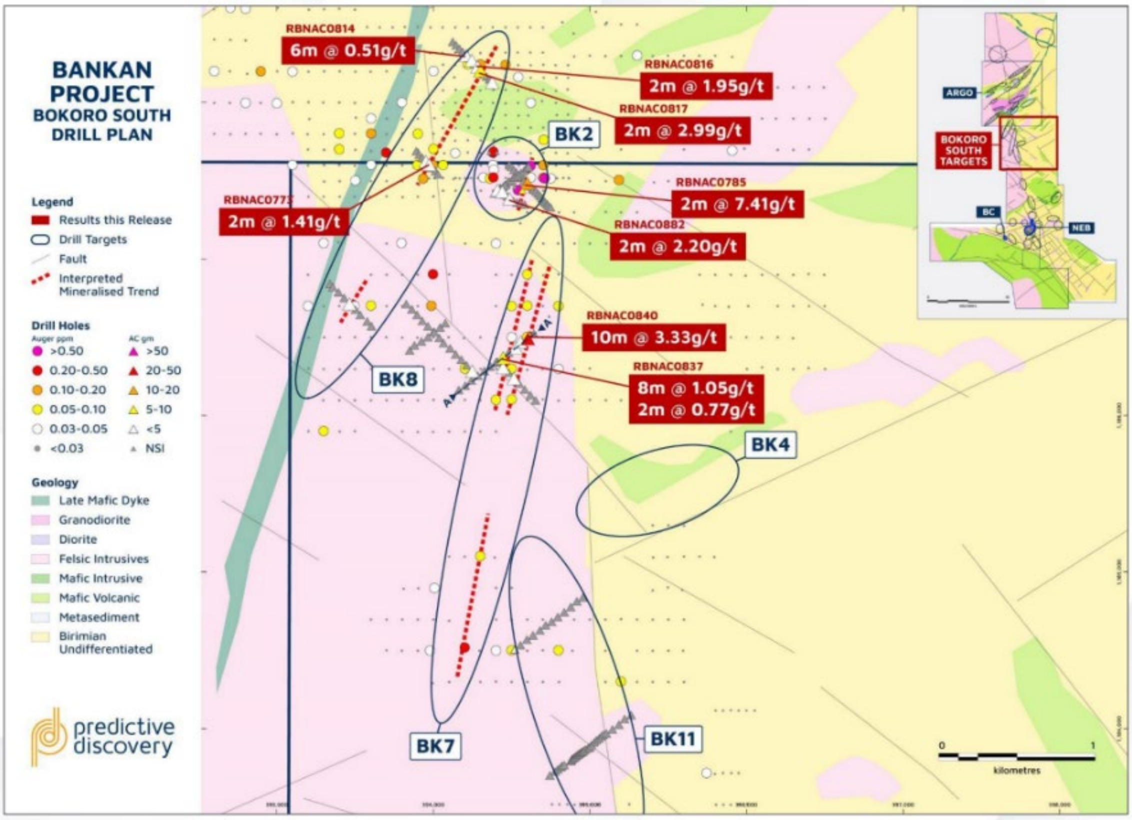

Figure 9.4: Bokoro Targets and Drilling Results (PDI 2024)

9.3 Conclusion

The Qualified Person considers that PDI has a systematic approach to identifying and testing exploration targets, and that the tenements are sufficiently prospective for further discoveries of gold resources. The success to date of the exploration and resource development programs van be expected to be replicated with sufficient future works of the types already completed.

10 DRILLING

10.1 Drilling Summary

PDI has used several different drilling contractors at the Project, drilling AC, RC, reverse circulation grade control (RCGC) and diamond core (DDH) holes, drilled between 2020 and 2025 (Table 10.1 and Figure 10.1). Of this drilling, a total of 1,052 AC, RC and DD holes for 157,171.49 m have been drilled (Table 10.2 and Table 10.3) and incorporated into the geological modelling with all data being used. For the resource estimate, however, only the DDH and RC drilling results were used. Some of the deeper diamond holes have utilised an RC pre-collar in expected hanging wall waste followed by DDH thereafter to reduce costs. Drillhole spacing is variable, with typically 40 m by 40 m in the upper parts of the deposits, and spacings as much as 100 m at the lower fringes.

Drilling results after the cutoff dates for the mineral resource estimates have been reviewed but not yet modelled. This includes infill drilling at both Northeast Bankan and Bankan Creek. The results of the additional data are in line with the resource models and are not expected to significantly change the mineral resource estimates or classification.

Table 10.1: Bankan Project Total Drillhole Summary by Year to 31 July 2025

Year		Aircore	Auger	Diamond	Reverse Circulation	RC Diamond Tail	RC Grade Control	Trench
2020	Holes	47	1451	16	19			8
	Metres	2,231	25712	3553.6	1698			448
2021	Holes	285	1,952	24	81	42		8
	Metres	12,233	40,529	5,196.65	8,755	17,798.54		298
2022	Holes	305	3,856	92	105	17	168	
	Metres	14,903	92,071	34,481.68	15,199	7,911.8	14,038	
2023	Holes	144	1,582	105	306	144		
	Metres	7,100	29,624	37,528.62	29,448	7,100		
2024	Holes	1523	1,955	75	283	1		
	Metres	71,215	28,156	15,723.8	27,236	354.5		
2025	Holes	479	946	23	37			
	Metres	21,235	16,177	3,939	4,218			
Type used in resource		No	No	Yes	Yes	Yes	No	No

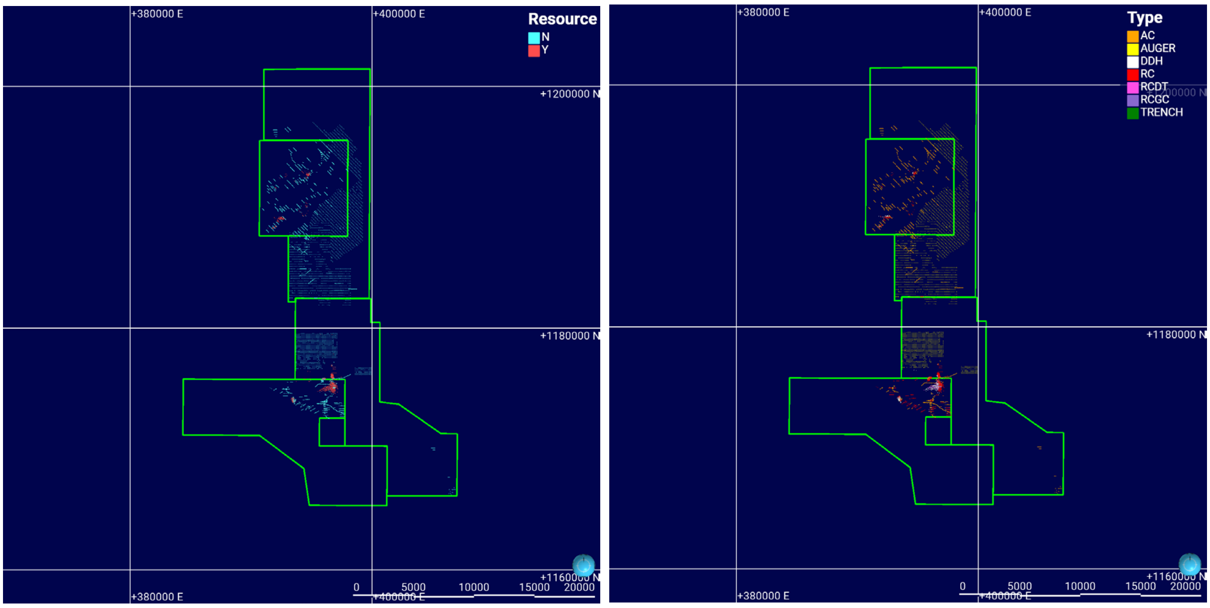

Figure 10.1: Bankan Project Drillhole Resource Plan; Resource and Non-Resource (L) and by Type (R)

Table 10.2: Bankan Project Diamond Drillhole Summary, Resource Areas Only

Hole From	Hole To	Area	Type	Holes	DDH metres
BCEDD0001	BCEDD0002	BC	DDH	2	365.3
BCKDD0001	BCKDD0054	BC	DDH	60	10,475.3
BCNDD0001		NEB	DDH		200
BNEDD0059	BNEDD0122A	NEB	DDH	14	6,139.45
BNEDD0123		GBD	DDH		211.9
BNEDD0124	BNEDD0125B	NEB	DDH	4	1,284.8
BNEDD0126		GBD	DDH		13.2
BNEDD0127	BNEDD0232	NEB	DDH	115	49,176.9
BNEDD0233	BNEDD0235A	GBD	DDH	4	764.7
BNEDD0236		NEB	DDH		860.3
BNEDD0237	BNEDD0238	800W	DDH	2	290.3
BNEDD0239	BNEDD0241B	NEB	DDH	6	2,927
BNEDD0242		800W	DDH		150
KKODD002		NEB			180.2
KKODD006	KKODD006A	BC		2	404.0
KKODD009	KKODD019	NEB		11	2,390.2
KKODD020		BC	DDH		250

Hole From	Hole To	Area	Type	Holes	DDH metres
RBNDD0002	RBNDD0006	FWG	DDH	3	719.5
RBNDD0007	RBNDD0032	SSN	DDH	23	5,052
RBNDD0033	RBNDD0038	FWG	DDH	6	1,525.5
RBNDD0039	RBNDD0044	SSN	DDH	6	1,538.5

Table 10.3: Bankan Project RC Drillhole Summary, Resource Areas Only

Hole From	Hole To	Area	Type	Holes	RC metres	DDH metres	Total metres
BCERC0001	BCERC0090	BC	RC	95	7,815		7,815
BCKRC0001	BCKRC0036	BC	RC	35	3,084		3,084
BCNRC0001	BCNRC0031	BC	RC	31	2,426		2,426
BCNRC0032	BCNRC0038	BC	RC	7	560		560
BCNRC0039	BCNRC0041	BC	RC	3	360		360
BNERC0004	BNERC0018	NEB	RC	10	1,769		1,769
BNERC0022	BNERC0024	GBD	RC	3	398		398
BNERC0025	BNERC0337	NEB	RC	140	17,747		17,747
BNERC0338	BNERC0352	GBD	RC	15	2,105		2,105
BNERC0353	BNERC0354	NEB	RC	2	286		286
BNERC0355	BNERC0358	800W	RC	6	750		750
BNERC0359	BNERC0380	GBD	RC	22	2,459		2,459
BNERC0381	BNERC0382	NEB	RC	2	146		146
BNERC0389		GBD	RC	1	100		100
BNERC0390	BNERC0405A	800W	RC	17	1,325		1,325
BNERC0418	BNERC0439	NEB	RC	25	1,920		1,920
BNERC0440	BNERC0442	800W	RC	3	200		200
BNERC0478	BNERC0480	800W	RC	3	220		220
BNERC0481	BNERC0485	NEB	RC	5	392		392
BNERC0518		NEB	RC	1	100		100
BNERC0545	BNERC0568	800W	RC	24	1,805		1,805
BNERC0569	BNERC0582	GBD	RC	15	1,691		1,691
BNERC0583	BNERC0587	NEB	RC	5	600		600
BNERC0588	BNERC0624	800W	RC	37	2,465		2,465
BNERC0625	BNERC0636	NEB	RC	12	1,010		1,010
BNERC0637	BNERC0639	800W	RC	3	210		210

Hole From	Hole To	Area	Type	Holes	RC metres	DDH metres	Total metres
BNERC0640	BNERC0641	NEB	RC	2	160		160
KKORC030		NEB	RC	1	87		87
KKORC050	KKORC054	BC	RC	5	401		401
KKORC059	KKORC061	NEB	RC	3	272		272
KKORC062		GBD	RC		90		90
KKORC063		NEB	RC		100		100
KKORC064	KKORC072	GBD	RC	6	586		586
KKORC073	KKORC074	NEB	RC	2	162		162
RBNRC0012	RBNRC0019	FWG	RC	9	1,163		1,163
RBNRC0020	RBNRC0024	SSN	RC	5	690		690
RBNRC0041	RBNRC0043	FWG	RC	3	530		530
RBNRC0050	RBNRC0051	FWG	RC	2	310		310
RBNRC0056	RBNRC0060	SSN	RC	5	708		708
RBNRC0069	RBNRC0080	SSN	RC	12	1,433		1,433
RBNRC0082	RBNRC0090A	SSN	RC	10	1,178		1,178
RBNRC0092	RBNRC0111	FWG	RC	20	2,519		2,519
RBNRC0112	RBNRC0134	SSN	RC	24	3,479		3,479
RBNRC0135	RBNRC0142	FWG	RC	8	912		912
RBNRC0143	RBNRC0157A	SSN	RC	20	2,544		2,544
BNERC0085	BNERC0276	NEB	RC GC	168	14,038		14,038
BNERD0001	BNERD0020	NEB	RC DT	20	2,400.71	3,502.46	5,903.17
BNERD0021		GBD	RC DT		100	126.21	226.21
BNERD0045	BNERD0097	NEB	RC DT	15	1,291.75	4,684.35	5,976.1

10.2 Auger Drilling

Following the NEB discovery, PDI completed a series of early-stage exploration programs, including broad spaced auger drilling and a helicopter-borne magnetic and radiometric survey. The aeromagnetics identified a major 35km-long north-northwest structural corridor with the potential to host multiple "NEB style" discoveries. Structural targets identified using the aeromagnetics have been progressively followed up with power auger and aircore (AC) drilling. The strategy to date has been to undertake wide-spaced auger drilling covering the structural targets, typically 320 m by 80 m spacing, followed by closer spaced infill where encouraging gold results have been obtained (generally plus 0.25g/t composite values in saprolite to depths of around 20 m). AC drilling has then followed up the encouraging auger results, typically with pairs of scissor holes to help assess the orientation of the gold mineralisation.

The auger drill is mounted on a light four-wheel drive vehicle (Figure 10.2) and has a variable penetration depending on the hardness of the weathered materials with holes typically terminated in the saprolite. Cuttings are collected at the collar in plastic bucket at 1 m downhole intervals and dumped in rows on the ground. Grab samples are taken from the piles and assayed at the third-party laboratory. These samples are useful for producing geochemical anomalies, however due to the open hole and non-representative sampling, are not used for resource estimation.

Figure 10.2: Auger Drill Rig (PDI 2024)

Th effectiveness of the auger sampling is demonstrated by Figure 10.3, where the contoured 0.1 g/t and 0.25 g/t values cover the footprint of the deposits.

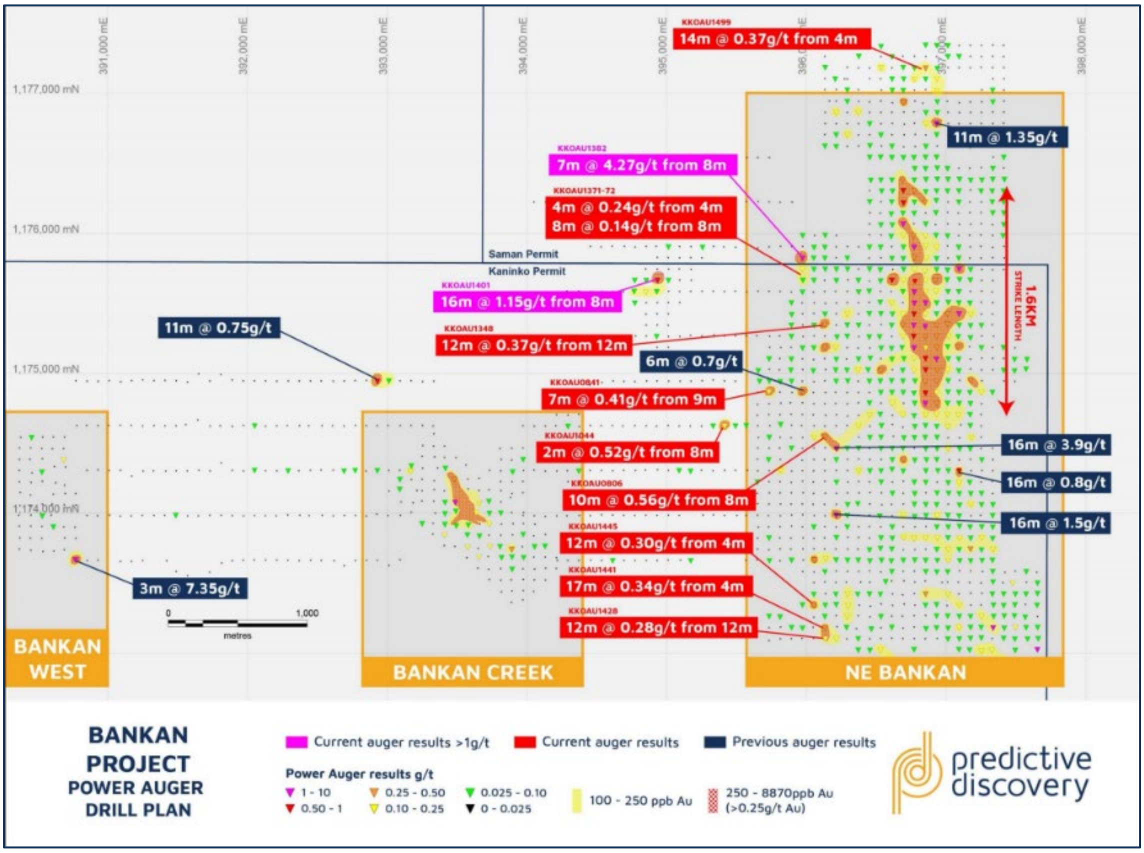

Figure 10.3: Auger Drill Result Contours around NEB and BC Deposits (PDI 2021)

10.3 Surveying

Collar surveying is by contracted surveyors using differential global positioning system (DGPS) enabled survey devices. Centimetric accuracy is achieved in the 3D positioning of drill collars and topographic features.

Holes are downhole surveyed with gyroscopic tools, utilising the Champ Gyro or the Reflex EZ Shot equipment, depending on the contractor.

10.4 Logging

All drill samples were logged systematically for lithology, weathering, alteration, veining, structure and minor minerals, with minor minerals estimated quantitively. The availability of qualitative and quantitative logging has appropriately informed the geological modelling, including weathering and oxidation, water table level and rock type. Photographs are taken of each core tray and chip tray of sieved and washed RC chips.

A WELLFORCE core orientation device was employed on all drilled DDH core, enabling orientated structural measurements to be taken.

Core is systematically logged for geotechnical properties. For each core run, the run length and recovered length are recorded. The rock quality designation (RQD) interval is also measured, this being the total length of core in each run recovered as pieces greater than 10 cm long; from these measurements a recovery percentage and RQD metric are calculated. Also recorded are rock strength and defect descriptions.

10.5 Conclusion

The Qualified Person considers that the drilling, surveying and logging methodologies are appropriately designed and implemented, and are capable of producing a dataset fit for purpose of identifying gold mineralisation and estimating Mineral Resources suitable for reporting classified as Measured, Indicated or Inferred as appropriate.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sample Dispatch

Samples from the Project are packed into polyweave bags, up to a maximum weight of 20 kg per bag. Samples are sent by truck to the assay laboratory. The assay laboratories utilised for the Project include:

- SGS, Bamako, Mali.
- SGS, Ouagadougou, Burkina Faso.
- Bureau Veritas, Conakry, Guinea.
- MSA Laboratories, Yamoussoukro, Côte d'Ivoire.

11.2 Sample Preparation and Assaying

Each of the four different laboratories used for the various programs have used slightly different, but comparable sample preparation and assaying techniques with differing detection limits (Table 11.1).

The majority have been assayed at SGS in Bamako, with their procedure including:

- Air drying of samples.
- Crushing to a P_{75} of 2 mm in a jaw crusher.
- Splitting by riffle splitter to provide a 1.5 kg sub-sample.
- Pulverising to a P_{85} 75 µm in a ring mill.
- Sub-sample of 200 g of pulp taken for assay.
- Barren flushing of equipment used before each job.
- Repeat analysis is performed when an analytical batch fails to meet SGS's data acceptance criteria or when requested by a client.
- Repeats by SGS due to internal QC failures are completed automatically and at no extra cost to the client.

Table 11.1: Analytical method summary

Laboratory	Method	Digest	Finish	Detection Limit	DL Unit
Bureau Veritas	FA550	Fire Assay	Gravimetric	0.9	g/t
Bureau Veritas	FA451	Fire Assay	AAS	5	ppb
Bureau Veritas	BL003	Cyanide Leach	ICP-MS	0.1	ppb
MSA	FAS-221	Fire Assay	AAS	0.01	g/t
MSA	FAS-121	Fire Assay	AAS	0.01	g/t
SGS	FAG505	Fire Assay	Gravimetric	0.5	g/t

Laboratory	Method	Digest	Finish	Detection Limit	DL Unit
SGS	FAA505	Fire Assay	AAS	0.01	g/t
SGS	FAA515	Fire Assay	AAS	5	ppb
SGS	ARE145	Aqua Regia	AAS	2	ppb

11.3 Data Management

All primary geological and sample data is physically captured in the field on standard logging templates. The data is then digitally entered into standard company Excel templates with data validation applied by the geologist who logs the hole. All data is reviewed and validated by the Senior Project Geologist before being emailed to PDI's Data Manager for uploading into a DataShed relational database. All Project data is contained in the Bankan Gold Project Dataset.

When analytical results are received from the laboratories, the raw assay files are directly merged into DataShed, which automatically matches and populates correct assay fields including QAQC records. Once completed, PDI exports the Project data to a number of formats including Microsoft Access, CSV, and Micromine, which are sent to the Project team as required.

11.4 QAQC Protocol

PDI have implemented a quality assurance/quality control (QAQC) program for the drilling and sampling at the Bankan project. QAQC comprises the use of:

- Certified reference materials (CRMs).
- Field duplicates, for both percussion (AC and RC) and diamond core drilling.
- Blanks.
- Laboratory duplicates.
- Umpire laboratory assaying.

On site, QAQC samples are inserted into the sample stream according to a pre-determined pattern, with slightly different patterns are used for percussion and core drilling as per Table 11.2.

Table 11.2: Bankan Project QAQC Sample Numbering Plan

QAQC type	Core Drillholes	Percussion Drillholes
Certified Reference Materials	Sample numbers *15 and *60	Sample numbers *05 and *55
Field Duplicates	Sample numbers *30 and *75	Sample numbers *20 and *70
Inserted Blanks	Sample numbers *45 and *90	Sample numbers *35 and *80

11.5 Certified Reference Materials

CRMs were inserted by PDI into the regular sample stream. These CRMs were sourced from Rocklabs Ltd, Oreas Pty Ltd, and Geostats Pty Ltd with their details are shown in Table 11.3. Control charts show that the assayed means of all the CRMs are very similar to their certified means, with no discernible trends and generally low variability.

Table 11.3: Bankan Project QAQC Sample Numbering Plan

CRM	Certified Au ppm	Number Analysed	Assayed Au ppm	Assayed CV
G307-1	3.37	19	3.42	0.11
G307-3	0.23	180	0.21	0.01
G307-4	1.40	58	1.41	0.08
G307-7	7.87	119	8.08	0.3
G313-4	2.00	339	2.03	0.06
G314-2	0.99	21	0.95	0.07
G316-1	0.31	333	0.32	0.02
G317-2	12.97	25	12.89	0.42
G320-1	78.81	117	80.27	3.53
G320-9	1.99	238	2.03	0.05
G397-6	3.95	342	3.87	0.2
G398-4	0.66	25	0.68	0.05
G910-2	0.90	18	0.91	0.09
G911-10	1.30	357	1.32	0.03
G913-2	2.40	28	2.38	0.2
G914-6	3.21	25	3.16	0.13
G919-5	11.30	131	11.67	0.31
G919-7	4.96	97	5.19	0.13
G919-8	0.57	101	0.65	0.02
OREAS-218	0.53	25	0.52	0.02
OREAS-221	1.06	22	1.04	0.05
OXD127	0.46	32	0.47	0.02
OXD151	0.43	62	0.44	0.02
OXD157	0.40	29	0.40	0.02
OXE150	0.66	21	0.67	0.02
OXE156	0.66	36	0.66	0.02
OXF162	0.83	57	0.83	0.02
OXF165	0.86	152	0.86	0.03
OXG140	1.02	29	1.04	0.03
OXH139	1.31	26	1.32	0.08
OXH149	1.28	9	1.28	0.02
OXH163	1.31	153	1.32	0.03
OXI138	1.86	52	1.85	0.06

CRM	Certified Au ppm	Number Analysed	Assayed Au ppm	Assayed CV
OXJ161	2.50	7	2.52	0.03
OXK160	3.67	36	3.69	0.08
SF85	0.85	172	0.85	0.02
SG84	1.03	155	1.02	0.03
SH82	1.33	101	1.32	0.03
SH98	1.40	232	1.39	0.09
SJ111	2.81	235	2.83	0.05
SJ80	2.66	46	2.64	0.1
SK94	3.90	130	3.88	0.1
SL76	5.96	15	5.90	0.1

11.6 Field Duplicates

Field duplicates were taken from both the DDH core (as quarter core duplicates) and from RC drilling (as second splits through the rig splitter). The statistics are shown in Table 11.4 to Table 11.7, with scatterplots from Figure 11.1 to Figure 11.4.

These show that the mean grades and variabilities of the paired sample sets are similar, however there is a moderate scatter in the RC duplicate pairs and considerable scatter in the DDH duplicate pairs. This latter suggests that the mineralisation is likely to be highly variable at a short scale, and this variability needs to be taken into account when planning future sampling programs.

Table 11.4: BNERC Holes Field Duplicate Statistics

	Original	Duplicate
Count	575	575
Minimum	0.04	0.03
Maximum	18.2	22.6
Mean	0.77	0.77
Median	0.4	0.4
Standard Deviation	1.40	1.49
CV	1.83	1.94

Table 11.5:KKORC Holes Field Duplicate Statistics

	Original	Duplicate
Count	139	139
Minimum	0	0
Maximum	3.48	3.00
Mean	0.36	0.33
Median	0.12	0.14
Standard Deviation	0.59	0.51
CV	1.64	1.53

Table 11.6:BNEDD Holes Field Duplicate Statistics

	Original	Duplicate
Count	440	440
Minimum	0	0
Maximum	35.1	31.9
Mean	1.14	1.17
Median	0.33	0.38
Standard Deviation	2.69	2.76
CV	2.36	2.37

Table 11.7:BCKDD Holes Field Duplicate Statistics

	Original	Duplicate
Count	100	100
Minimum	0.07	0.04
Maximum	17.3	13.4
Mean	0.98	0.97
Median	0.295	0.31
Standard Deviation	2.26	2.2
CV	2.32	2.29

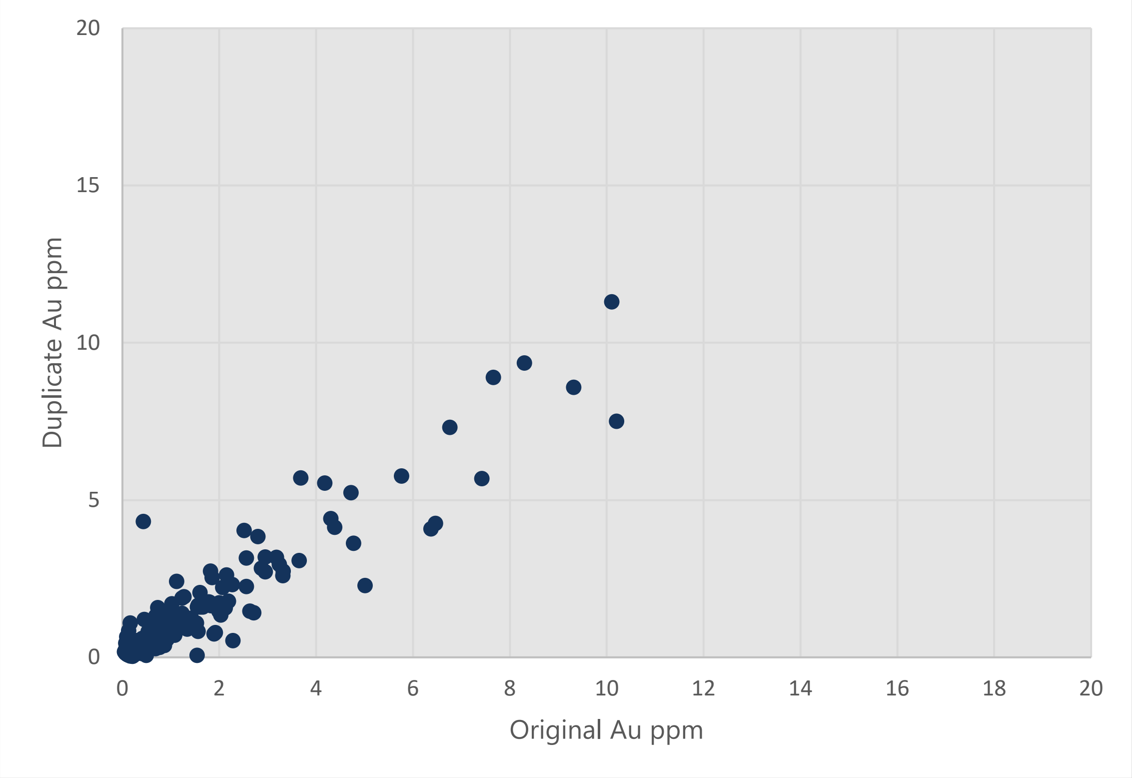

Figure 11.1: BNERC Holes Field Duplicates Scatterplot

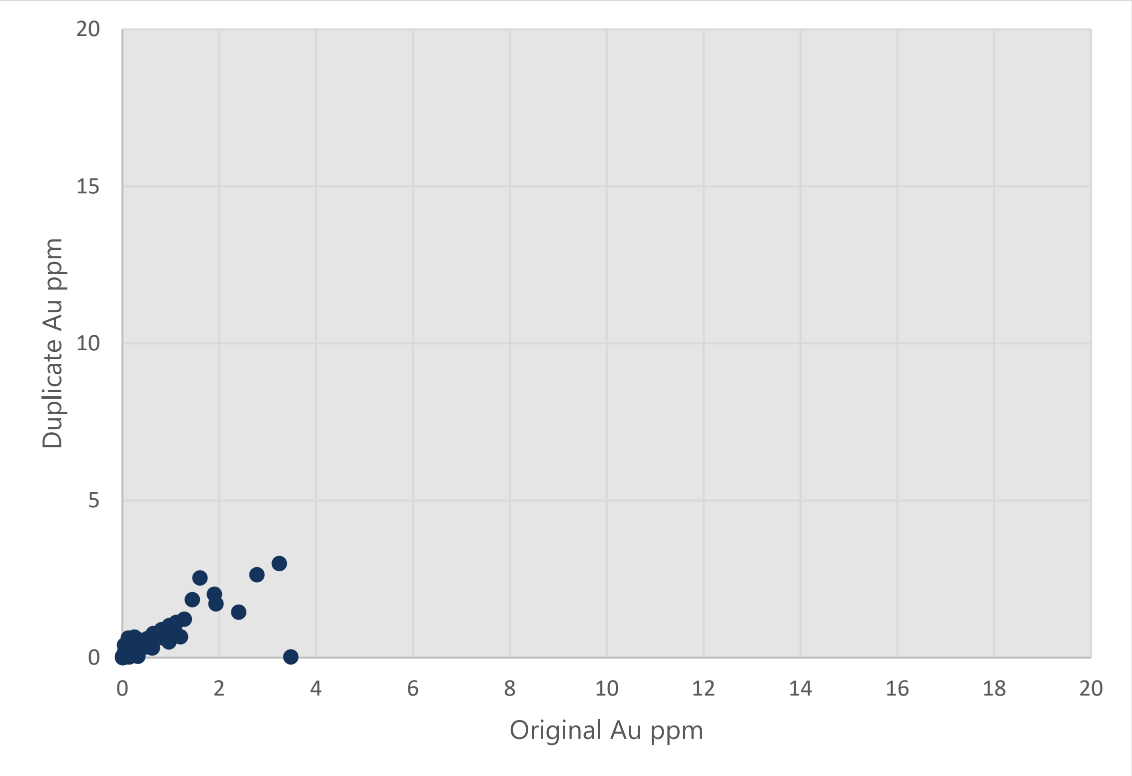

Figure 11.2: KKORC Holes Field Duplicates Scatterplot

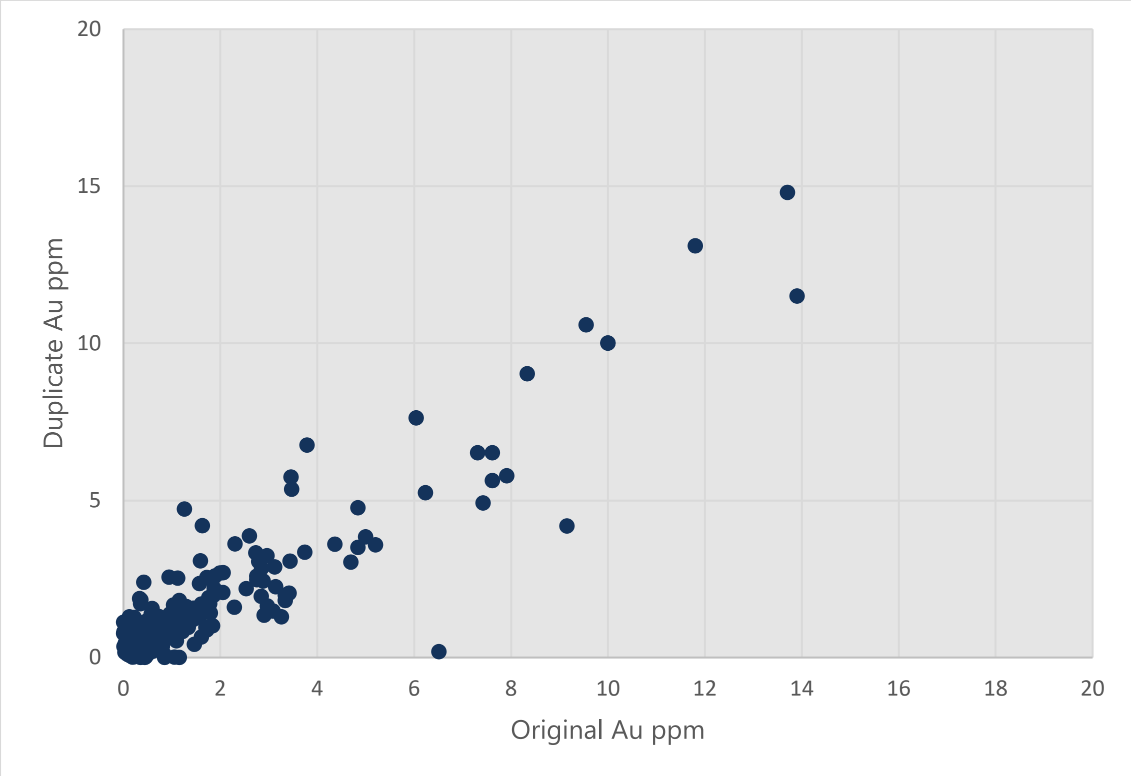

Figure 11.3: BNEDD Holes Field Duplicates Scatterplot

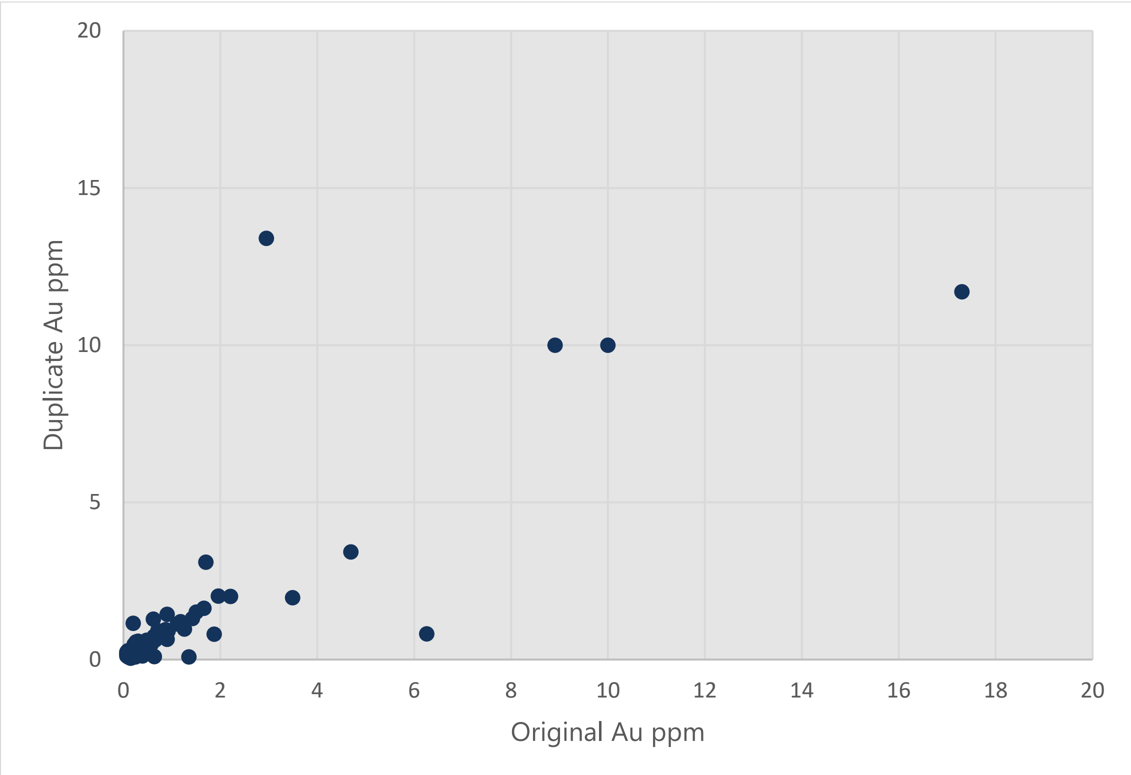

Figure 11.4: BCKDD Holes Field Duplicates Scatterplot

11.7 Blanks

Blank samples, consisting of non-mineralised gravel, are inserted into the sample stream to test for cross-contamination in the laboratory. A total of 3,769 blanks were assayed during the various sampling programme with 3,000 (80%) returning results that were below detection limit, 139 (4%) were between the detection limit and 0.01 g/t, 302 (8%) were 0.01 g/t, 234 (6.2%) were between 0.01 g/t and 0.02g/t, and 94 (2.5%) were between 0.02 g/t and 0.04 g/t. These results suggest a low level of contamination in the laboratory. Note that the blank has not been certified as being below detection limit.

11.8 Laboratory QAQC

In addition to the QAQC samples inserted by PDI, the laboratories also implemented their own internal QAQC programmes, the results of which were reported to PDI. The SGS Bamako QAQC programme comprised the following in each 84 samples prepared and assayed:

- 1 reagent blank.
- 1 preparation blank (prep process blank).
- 1 weighed replicate.
- 2 preparation duplicates (re-splits).
- 3 CRMs.

In addition, repeat assays of the analyte solutions are performed.

The comparative statistics (Table 11.8 and Table 11.9) show a very high correspondence between the original and duplicates and repeats.

Table 11.8: Laboratory Original and Duplicate Au g/t Statistics

	Original	Duplicate
Count	112	112
Minimum	0	0
Maximum	2.23	2.24
Mean	0.15	0.15
Median	0.06	0.06
Standard Deviation	0.29	0.29
CV	1.91	1.90

Table 11.9: Laboratory Assay Repeat Au g/t Statistics

	Original	Duplicate
Count	200	200
Minimum	0	0
Maximum	194.88	195.02
Mean	2.51	2.51
Median	0.09	0.09
Standard Deviation	17.82	17.79
CV	7.09	7.09

11.9 Umpire Laboratory Assaying

Sample pulps from SGS Bamako were selected for umpire laboratory assaying at Bureau Veritas Bamako. Samples were selected from the following holes: BNERD0001, BNERD0008, BNERD0012A, BNEDD0085, BCKDD0016 and KKORC006.

Bureau Veritas assayed pulps by fire assay/AAS finish with a 50 g charge, with analysis limits of 5 to 10,000 ppb Au and any samples that were over range were re-assayed by fire assay with a gravimetric finish.

On receipt of the results, there were 17 sample pairs in two groups of consecutive numbers where there appeared to be a number mislabelling issue, and these results were disregarded for the following analysis.

The resultant dataset contains 461 sample pairs, with the statistics of the original and umpire assays presented in Table 11.10. From this data, it is apparent that the mean of the original is higher than the umpire, although the medians are similar. This is likely due to a small population where the original grades were not reproduced by the umpire assay. This may be due to mislabelling and is being investigated. Quantile-quantile' (QQ') plots (Figure 11.5 and Figure 11.6) show that the distributions of the original and umpire datasets are similar.

Table 11.10: Original and umpire Au g/t statistics

Statistic	Original	Duplicate
Count	461	461
Minimum	0.0025	0.005
Maximum	66	44.4
Mean	1.65	1.47
Median	0.57	0.55
Standard Deviation	4.95	4.11
CV	3.00	2.79

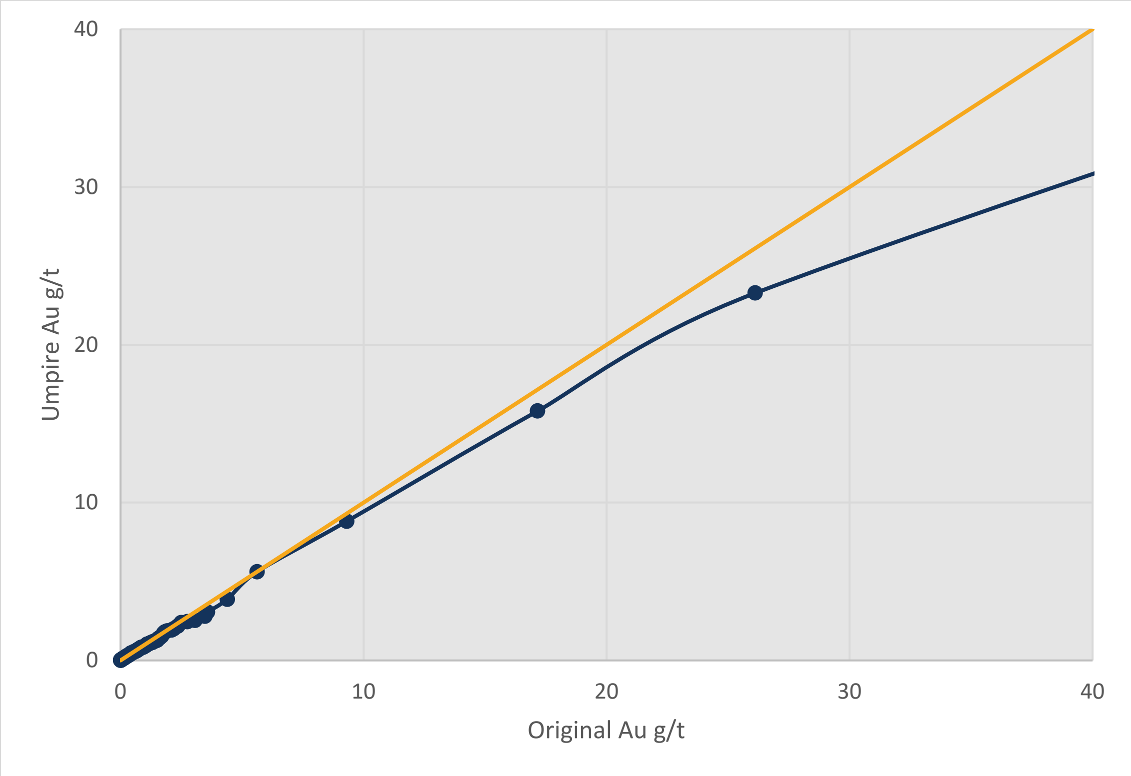

Figure 11.5: Original and Umpire Laboratory QQ′ Plot

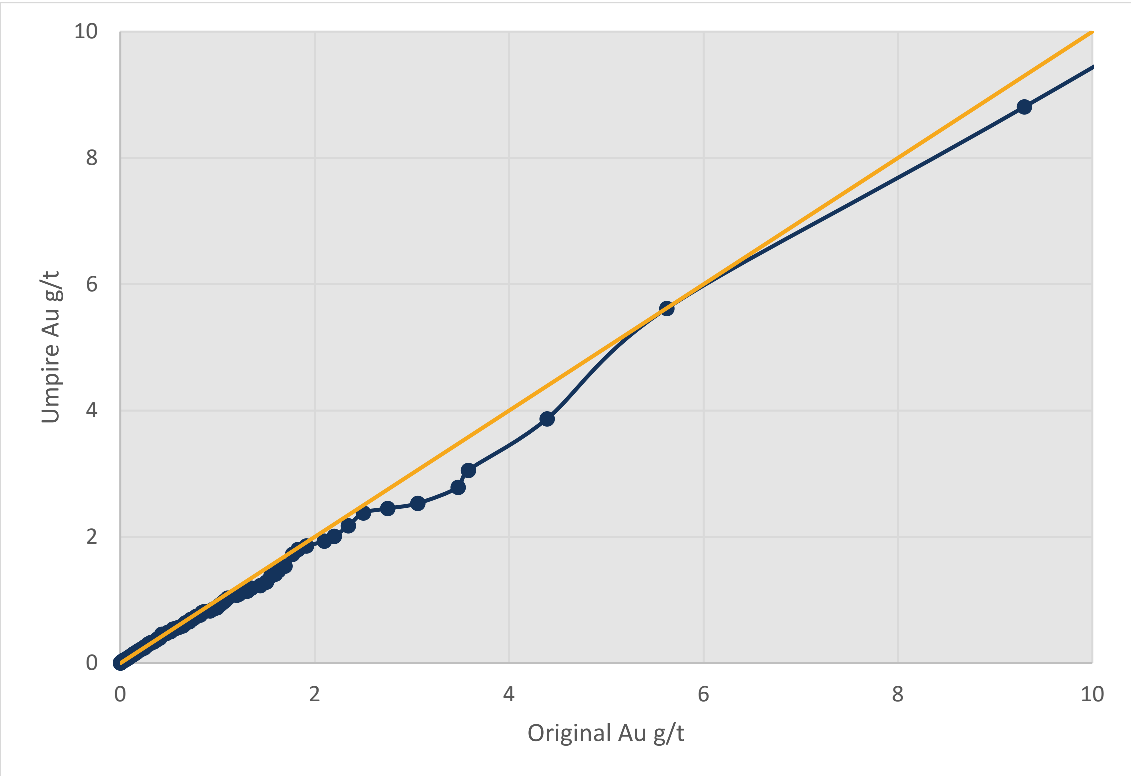

Figure 11.6: Original and Umpire Laboratory QQ′ Plot <10 g/t

11.10 Sampling Precision Comparison

To measure the relative precision of the different duplicate sample steps, a ranked average relative difference (ARD) plot was constructed, for all pairs of data where the mean grade was greater than 0.1 g/t Au. The plot (Figure 11.7) shows the distribution of the ARD of sample pairs with the expectation that duplicate sample pairs at an earlier stage of the sample preparation process (i.e. coarser particle size) should have poorer precision than duplicate sample pairs at a later stage (e.g. pulp duplicates). The plot shows that the precision of the sampling is reasonable, with the poorest precision in the coarse duplicate pairs, suggesting that there is a moderate to high fundamental nugget factor in the mineralisation.

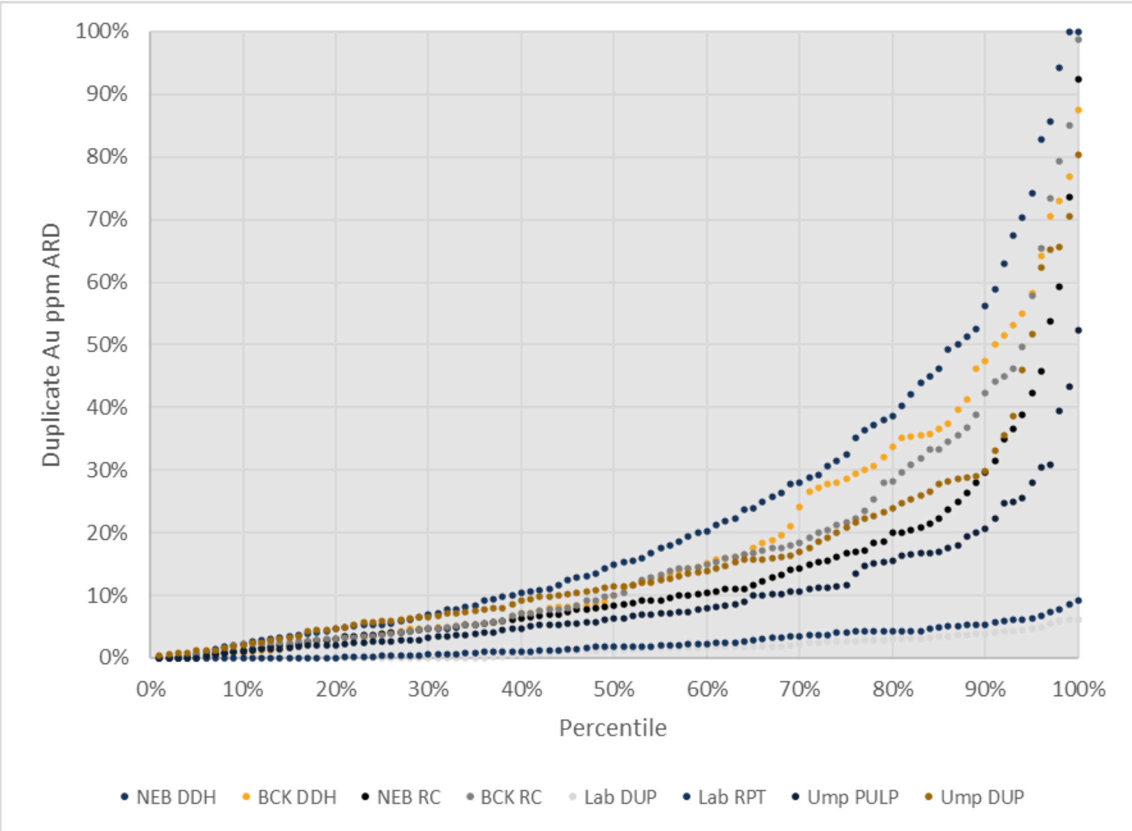

Figure 11.7: Ranked ARD Plot of Duplicate Sample Pairs

11.11 Drillhole Direction Analysis

Early drilling at NEB was orientated to the west, however after analysis of the results, it was realised that the mineralisation dips to the west, and these holes were down-dip. Subsequent drilling was orientated to the east being at a high angle to the dip of the mineralisation.

The composite dataset in the area with both drilling directions was analysed into east and west dipping datasets and split into saprolite and fresh subsets using the interpreted base of saprolite DTM. The composite statistics are presented in Table 11.11 and Table 11.12. These show:

- Mean and median of the west dipping holes are higher than east dipping in the saprolite.
- In the saprolite, the composites in the west dipping holes are more variable.

- West dipping holes in the saprolite have a larger population greater 2 g/t Au.
- Mean and median of the west dipping holes are lower than east dipping in the fresh sub-set.
- In the fresh, the composites in the west dipping holes are less variable.

Given that the data in the west dipping holes is redundant, with the areas drilled west dipping having been completely drilled by east dipping holes, the west dipping data was filtered from the composite dataset before further processing.

Table 11.11: Saprolite Composites Au g/t Statistics by Hole Direction

Statistic	East Dipping	West Dipping
Count	1,067	1,107
Minimum	0.0025	0.0025
Maximum	26.39	72.09
Mean	0.57	0.80
Median	0.18	0.22
Standard Deviation	1.37	3.28
CV	2.43	4.07

Table 11.12: Fresh Composites Au g/t Statistics by Hole Direction

Statistic	East Dipping	West Dipping
Count	492	496
Minimum	0.0025	0.0025
Maximum	67.78	26.06
Mean	0.71	0.58
Median	0.27	0.2
Standard Deviation	3.24	1.67
CV	4.58	2.89

11.12 Sample Security

Samples are bagged in polyweave sacks, sealed and then driven directly to the assay laboratory; the current laboratory used is SGS in Bamako, Mali which requires crossing an international border. The laboratory returns a subsample of each prepared pulp, with approximately 200g in a labelled paper packet and the packets collated into labelled cardboard storage boxes. The cardboard boxes are stored onsite in locked freight containers.

Drill core and RC retention bags are stored onsite with a 24-hour security presence. Core is stored in a combination of undercover storage racks and palletised stacks of core trays covered with tarpaulins. The core is well organised and can be easily retrieved. The core trays are labelled and have depth intervals clearly marked with labelled core blocks. In general, the core trays and blocks are in reasonable condition. RC retention bags are arranged by drillhole in the open air.

12 DATA VERIFICATION

PDI has been developing the resources since 2019. The Qualified Person has produced visited the site on four occasions, from the 10th to the 15th June 2022, from the 10th to the 21st November 2022, from the 11th to the 27th January 2023 and from 28th August 2024 to the 5th September 2024. During these visits, the following were inspected:

- General site layout and facilities.
- Surface area of Northeast Bankan, Bankan Creek, and Fouwagbe deposits.
- DDH, RC, AC and auger drilling. The inspection verified that the drilling and sampling procedures were appropriately designed and implemented, with the ability to produce representative samples.
- Drillhole setup. The inspection verified that the drillholes were accurately located at the planned locations.
- DDH core orientation and markup. The inspection verified that the core is correctly orientated, and that he markup of the core was accurate, consistent and legible enabling easy identification of relevant core intervals.
- DDH core logging and sampling. The inspection verified that the core logging procedures were appropriately defined, and that the procedures were being implemented correctly and that all relevant information was being recorded accurately. The sampling procedures were also appropriately designed and were being diligently implemented and accurately recorded.
- Density measurement procedure. The inspection verified that the procedure was appropriately designed and implemented, with the ability to produce accurate measurements of the selected core pieces, and that the core pieces were representative of the rock mass as a whole.
- Point Load Test (PLT) measurement procedure. The inspection verified that the PLT measurement device was appropriate and was being used correctly, with results accurately recorded.
- X-Ray Fluorescence measurement procedure. The inspection verified that the XRF device was appropriate and that the procedures produce representative values from the samples presented; and that the results were accurately recorded.
- AC, RC and auger logging and sampling. The inspection verified that the logging and sampling procedures were appropriately designed and implemented, with the ability to produce representative samples.
- Sample dispatch. The inspection verified that the chain of custody was maintained to the point of dispatch.
- DDH core and RC retention bag storage. The inspection verified that core and RC samples were being retained on site, and were appropriately labelled and arranged to be accessible in the future.

- Pulp storage. The inspection verified that returned samples were being retained on site, and were appropriately labelled and arranged to be accessible in the future. Note that some have been subsequently accidentally destroyed.
- Review of selected core intervals and comparison with assaying results. The comparison verified that elevated gold grades were associated with visible alteration, veining and sulphide mineralisation, and in some cases visible gold was observed.

Detailed technical discussions with PDI staff were also conducted.

The Qualified Person has checked a selection of the original assay certificates against the database and not identified any errors.

The Qualified Person has not selected, taken and independently assayed representative samples as verification of the processes in place and the comparison of selected core intervals with assay results provided sufficient confidence in the veracity of the data provided.

The drilling, sampling, assaying, quality assurance, sample security and data handling procedures at the Project are well designed and are well implemented; they are capable of producing a reliable dataset that is fit for purpose for mineral resource estimation.

Based on the assessment of the data, the Qualified Person considers the entire dataset to be acceptable for resource estimation, subject to the preceding comments regarding the analytical accuracy and precision.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Metallurgical testwork for the Project has effectively been completed in three programs:

- Program 1: Preliminary testwork managed by Mintrex (2021).
- Program 2: Interim testwork managed by Independent Metallurgical Operations (IMO) (2024).
- Program 3: Latest testwork managed by PDI/Dhamana (2024-2025).

The process definition testwork was largely completed in the first two of these programs, where the work was focused on defining the key process design criteria, which included:

- Grind size, selecting a grind size of (P_{80}) 75µm.
- Gravity gold recovery, which demonstrated that there is material levels of gravity recoverable gold.
- Carbon in leach (CIL) or carbon in pulp (CIP) which demonstrated that hybrid CIL is suitable due to minimal preg-robbing by the ore.
- Leaching residence time, demonstrating that industry typical residence time of 24 hours was suitable.
- Oxygen versus air for oxygen for the leaching, showing the use of air is comparable to oxygen in the cyanide leaching process, therefore the more cost-effective use of sparged air is justified.
- Leaching of the gravity tail with a range of cyanide concentration demonstrating that there is no loss of gold extractions when leaching commenced with an initial CN concentration of 500ppm then allowing it to decrease to 120ppm during the leach.
- Comminution testing to provide early estimates of ore hardness and grindability.

The results these programes were reported by Mintrex, Como Engineers (Como) and Independent Metallurgical Operations (IMO) in the following reports:

- Bankan Project, Scoping Testwork Report, Project Number 21017 PDI, Mintrex, August 2021. (Mintrex, 2021).
- 22033-PDI-MEM-001_0 – Additional Leach Results, Mintrex, July 2022, (Memorandum). (Ngo, 2022).
- Bankan Gold Project Metallurgical Testwork Review. Job No. 3903.01. October 2023. (Como Engineers, 2023).
- Bankan Gold Project, Ore Characterisation Testwork, IMO October 2024. (IMO, 2024).

The final program, completed in parallel with the DFS, used the earlier work that defined the general process flowsheet and conditions to complete a variability program and bulk testwork with the aims of better defining the range of process performance and design envelope and to define carbon loading and cyanide destruction performance and design parameters, focussing on the sizing, or requirements for, of the following aspects of the process:

- Pre-leach thickener and tailings thickener sizing and benefits across a range of feed blends.
- Tailings filtration sizing across a range of feed blends.
- Paste-fill feed requirements and the requirement for desliming, also providing representative feed for the testing of paste properties.

Materials handling testwork was also completed in this program across the range of lithologies present in the ore body.

The metallurgical testwork from this program was reported in the following:

- Metallurgy Testwork conducted upon samples from the Bankan Gold Project. Report No. A26173, ALS Metallurgy, 2025. (ALS Metallurgy, 2025).
- Gold Ore Flow Properties Report, 73065-1 Rev 0, Jenike & Johanson, 2025. (Jenike & Johanson, 2025).
- Bankan Paste Prefeasibility Study Testwork. Report No. 24054-RPT-0002. Rev A, Minefill Services, 2025. (Minefill Services, 2025).

It is the opinion of the Qualified Person that metallurgical test work programs described in Section 13 are of sufficient scope and quality to support the production planning, estimates of costs and the proposed process plant design described in Section 17.

13.2 Sample Selection

The samples used in the various metallurgical programs are presented in the tables below. Table 13.1 presents the 20 individual samples collected for Program 1, completed by Mintrex, as well as the two composites prepared from these samples to represent altered tonalite (Composite A) and saprolite (Composite B).

Table 13.1: Program 1 Samples (Mintrex)

Sample ID	Drill Hole ID	Interval (m)	Domain / Orebody	Lithology	Au Grade (g/t)
PDG-MT001	BNERD002	148.85-175	NEB	Altered tonalite	2.00
PDG-MT002	BNERD007	150-164	NEB	Mafic volcanics	1.89
PDG-MT003	BNERD007	178-196	NEB	Altered tonalite	5.47
PDG-MT004	KKODD013	33-60	NEB	Saprolite	0.83[1]
PDG-MT005	KKODD015	8-35	NEB	Saprolite	1.33
PDG-MT006	BNERC058	30-47	NEB	Saprolite	0.83[1]
PDG-MT007	KKODD020	82-112	BC	Skarn, marble, tonalite, vein	1.71[2]
PDG-MT008	KKODD020	112-143	BC	quartz	1.71[2]

Sample ID	Drill Hole ID	Interval (m)	Domain / Orebody	Lithology	Au Grade (g/t)
PDG-MT009	KKODD011	97-127	NEB	Altered tonalite	2.73
PDG-MT010	BNERD003	200-230	NEB	Altered tonalite	0.90
PDG-MT011	BNERD014	188-218	NEB	Mafic volcanics	2.04
Comp 1	BNERD0068	246-274	NEB	Altered Tonalite	3.20
Comp 2	BNERD0074	322.4-365	NEB	Altered Tonalite	2.87
Comp 3	BNEDD0087	273-1.03	NEB	Altered Tonalite	2.87
Comp 4	BNEDD0088	307-325	NEB	Altered Tonalite	21.72
Comp 5	BNERD0091	385-426	NEB	Altered Tonalite	4.25
Comp 6	BNERD0094	401-421	NEB	Altered Tonalite	1.34
Comp 7	BNERD0098	470.8-492	NEB	Altered Tonalite	1.59
Comp 8	BNERD0105	622-640	NEB	Altered Tonalite	3.89
Comp 9	BNERD0106B	661-681	NEB	Altered Tonalite	2.46
Comp A	-	-	NEB	Altered Tonalite	2.79
Comp B	-	-	NEB	Saprolite	1.00

Notes:

1. MT004 and MT006 were combined for head assay
2. MT007 and MT008 were combined for head assay

The samples, and subsequent composites prepared for use in Program 2, managed by IMO, are presented in Table 13.2.

Table 13.2: Program 2 Samples (IMO)

Comp.	Sample ID	Drill Hole ID	Interval (m)	Domain / Orebody	Lithology	Au Grade (g/t)
A	PDI23-MT002	BNEDD0176	162-172	NEB	Fresh veined, foliated K feldspar altered tonalite	2.13
	PDI23-MT003	BNEDD0176	172-174	NEB	Tonalite	2.09
	PDI23-MT010	BNEDD0191	556-562.3	NEB	Chlorite altered tonalite	2.06
	Composite A			NEB		2.37
C	PDI23-MT001	BNEDD0158	161-167	NEB	Sheared and chlorite altered mafic	1.51
	PDI23-MT004	BNEDD0149	132-140	NEB	Sheared and chlorite altered mafic	2.35
	PDI23-MT005	BNEDD0225	155-163	NEB	Sheared, chlorite altered and quartz veined mafic	2.33

Comp.	Sample ID	Drill Hole ID	Interval (m)	Domain / Orebody	Lithology	Au Grade (g/t)
	Composite C			NEB		2.16
F	PDI23-MT008	BNEDD0196	525.75-530	NEB	Sheared, foliated, heavily altered quartz veined	4.12
	PDI23-MT033	BNEGT02	548-551	NEB	Strongly foliated quartz veined metasediment	7.28
	PDI23-MT034	BNEGT02	557-560	NEB	Strongly altered tonalite	3.56
	Composite F			NEB		2.96
B[1]	PDI23-MT006	BNEDD0157	44 - 62	NEB	Saprolitic mafic	2.41
	PDI23-MT007	BNEDD0162	80 - 86	NEB	Saprolitic felsic	2.34
D[1]	PDI23-MT011	KKODD006A	72.1 - 82	BC	Chlorite altered, quartz veined tonalite	10
	PDI23-MT012	KKODD006A	82 - 90	BC	Chlorite altered, quartz veined tonalite	6.41
	PDI23-MT015	BCKDD0041	51 - 55	BC	Strongly altered and quartz veined tonalite	-
	PDI23-MT016	BCKDD0041	60 - 63	BC	Strongly altered and quartz veined tonalite	-
	PDI23-MT017	BCKDD0041	74 - 78	BC	Strongly altered and quartz veined tonalite	-
	PDI23-MT018	BCKDD0041	104 - 108	BC	Strongly altered and quartz veined tonalite	-
	PDI23-MT019	BCKDD0041	108 - 112	BC	Very strongly altered and foliated tonalite	-
	PDI23-MT020	BCKDD0041	112 - 116	BC	Strongly sheared and quartz veined tonalite	-
	PDI23-MT021	BCKDD0041	116 - 120	BC	Strongly sheared and quartz veined tonalite	-
	PDI23-MT022	BCKDD0041	120 - 124	BC	Strongly chlorite altered tonalite	-
	PDI23-MT023	BCKDD0041	124 - 128	BC	Serricite altered tonalite	-
	PDI23-MT024	BCKDD0041	128 - 132	BC	Strongly altered and quartz veined tonalite	-
	PDI23-MT025	BCKDD0041	132 - 136	BC	Silicified tonalite	-
	PDI23-MT026	BCKDD0041	136 - 140	BC	Sheared and k feldspar altered tonalite	-
	PDI23-MT027	BCKDD0041	140 - 144	BC	Sheared and k feldspar altered tonalite	-

Comp.	Sample ID	Drill Hole ID	Interval (m)	Domain / Orebody	Lithology	Au Grade (g/t)
	PDI23-MT028	BCKDD0041	144 - 148	BC	Sheared and k feldspar altered tonalite	-
	PDI23-MT029	BCKDD0041	148 - 150	BC	Strongly chlorite altered tonalite	-
E[1]	PDI23-MT030	BCKDD0037A	12 - 27	BC	Saprolitic tonalite	-
	PDI23-MT031	BCKDD0037A	27 - 38	BC	Saprolitic tonalite	-
	PDI23-MT032	BCKDD0037A	38 - 50	BC	Saprolitic tonalite	-
	PDI23-MT013	BCKDD0041	42.6 - 47	BC	Saprolitic sheared, quartz veined mafic	-

Notes:

1. The weathered NEB (Comp B) and BC samples (Comp D & Comp E) were not tested with these samples were made available for the later program conducted by ALS.

In Program 2, as shown in Table 13.2, not all intervals were assayed and three of the composites were not created as the program was trimmed to focus on the NEB fresh samples which make up the majority of the proposed mill feed.

A significant focus of the ALS program was to test the variability of performance across the deposits and identified lithologies, a large range of grades, lithologies and spatial sources were selected for testing. The samples collected for Program 3, along with those identified above as being carried over the Program 2, are shown in Table 13.3.

Table 13.3: Program 3 Samples (ALS)

Sample ID	Drill Hole ID	Interval (m)	Domain / Orebody	Lithology
Comp #01	BNEDD0232	124.1 - 166.9	NEB (Open Pit)	Mafic
Comp #02	BNEDD0232	168.4 - 190.7	NEB (Open Pit)	Mafic
Comp #03	BNEDD0232	190.9 - 218	NEB (Open Pit)	Tonalite
Comp #04	BNEDD0204	195.62 - 233.5	NEB (Open Pit)	Mafic
Comp #05	BNEDD0229	54.5 - 65.52	NEB (Open Pit)	Shear
Comp #06	BNEDD0229	66.5 - 75.5	NEB (Open Pit)	Saprock
Comp #07	BNEDD0229	75.63 - 126.9	NEB (Open Pit)	Mafic
Comp #08	BNEDD0229	128.1 - 143	NEB (Open Pit)	Tonalite/Int
Comp #09	BNEDD0147	41.1 - 48.6	NEB (Open Pit)	Shear
Comp #10	BNEDD0147	53.1 - 72.6	NEB (Open Pit)	Saprolite
Comp #11	BNEDD0147	74.1 - 87.6	NEB (Open Pit)	Saprolite
Comp #12	BNEDD0147	89.1 - 97.75	NEB (Open Pit)	Shear

Sample ID	Drill Hole ID	Interval (m)	Domain / Orebody	Lithology
Comp #13	BNEDD0147	98.32 - 148	NEB (Open Pit)	Tonalite
Comp #14	BNEDD0162	39 - 73	NEB (Open Pit)	Saprolite
Comp #15	BNEDD0162	73.33 - 79	NEB (Open Pit)	Shear
Comp #16	BNEDD0162	79 - 92.35	NEB (Open Pit)	Saprolite
Comp #17	BNEDD0162	96 - 105.3	NEB (Open Pit)	Mafic
Comp #18	BNEDD0227	355.9 - 364.85	NEB (Open Pit)	Shear
Comp #19	BNEDD0120	423.3 - 442.26	NEB (Open Pit)	Shear
Comp #20	BNEDD0120	443.65 - 458.44	NEB (Open Pit)	Tonalite/Int
Comp #21	BNEDD0180	409 - 412.65	NEB (Open Pit)	Shear
Comp #22	BNEDD0188	141.08 - 149.88	NEB (Open Pit)	Shear
Comp #23	BNEDD0157	62.69 - 157.8	NEB (Open Pit)	Tonalite
Comp #24	BNEDD0196	512 - 525.62	NEB UG	Tonalite
Comp #25	BNEDD0196	530 - 543.38	NEB UG	Tonalite
Comp #26	BNEDD0221	396.55 - 409.76	NEB UG	Shear
Comp #27	BNEDD0221	410.4 - 429.36	NEB UG	Tonalite
Comp #28	BCKDD0043	76.49 - 88.19	BC	Scarn
Comp #29	BCKDD0043	93.1 - 102	BC	Scarn
Comp #30	BCKDD0030	233.94 - 240.1	BC	Quartz Vein
Comp #31	BCKDD0030	271.5 - 279	BC	Quartz Vein
Comp #32	BCKDD0031	240.23 - 263	BC	Tonalite

Key to being able to select the individual lithologies was identification of the lithologies and the transition between lithologies. This was possible due to the clear differences and visible transitions in the core, which was used in addition to the logging for sample selection. This is demonstrated in the following figures as follows:

- Figure 13.1 presents a photograph of core from 90 m to 100.71 m showing the transition from shear zone to the tonalite zone in hole BNEDD0147.
- Figure 13.2 presents a photograph of typical saprolite lithology from hole BNEDD0147. This sample is typically recovered wet and has low competency compared to the shear and tonalite sections.
- Figure 13.3 presents an obviously mafic section from BNEDD0204.

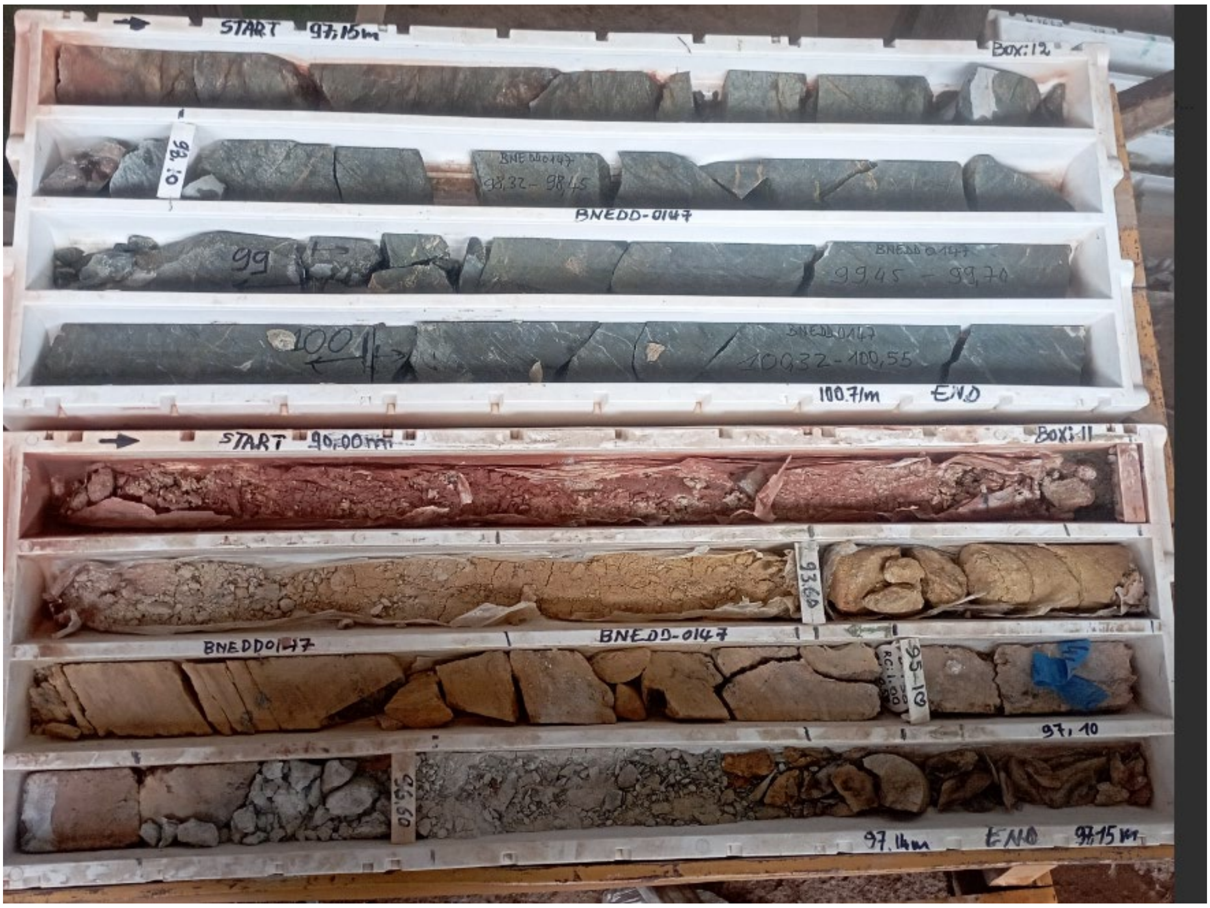

Figure 13.1: Shear and Tonalite Lithology Transition, BNEDD0147

Figure 13.2: Saprolite Lithology, BNEDD0147

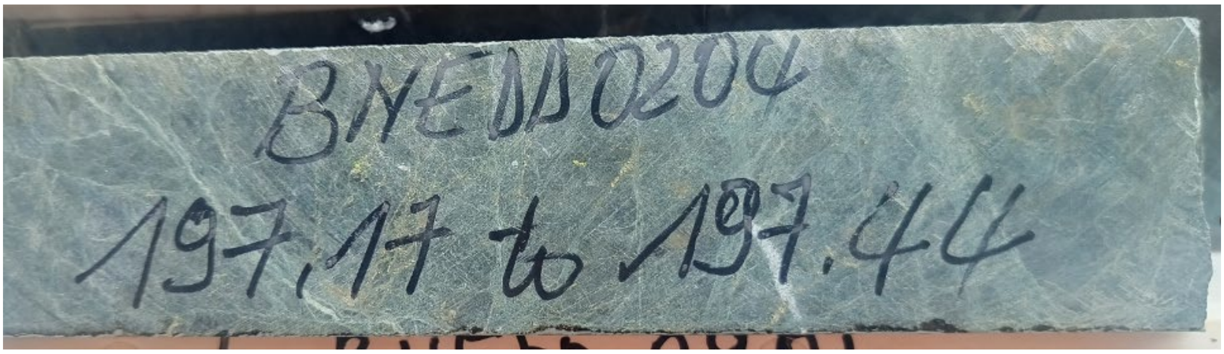

Figure 13.3: Mafic Lithology, BNEDD0204

13.3 Head Assays

The head assays of the variability samples used in Program 3 campaign are presented in Table 13.4 and Table 13.5.

Table 13.4: Program 3 Variability Sample Head Assays – Part 1

Sample ID	Ag	Al	As	Au (ave)	Ba	Be	Bi	C-tot	C-org	Ca	Cd	co	Cr	Cu	CNsCu	Fe	Hg	K
	ppm	%	ppm	g/t	ppm	ppm	ppm	%	%	%	ppm	ppm	ppm	ppm	ppm	%	ppm	%
Comp #01	0.6	8.96	90	0.64	200	<5	<10	0.36	<0.03	4.70	<5	40	200	136	102	8.88	0.2	0.80
Comp #03	0.9	7.72	20	1.30	700	<5	<10	0.15	0.06	2.10	<5	20	80	294	208	4.20	0.2	2.60
Comp #04	1.5	7.92	110	1.26	100	<5	<10	0.99	<0.03	4.40	<5	70	210	342	248	8.64	0.2	1.20
Comp #05	1.8	5.52	1190	1.40	400	<5	<10	0.15	0.12	<0.10	<5	15	110	258	34	5.66	0.2	2.40
Comp #06	0.9	9.04	340	0.69	300	<5	<10	0.27	0.03	2.10	<5	70	160	394	276	11.1	0.2	1.20
Comp #07	0.9	9.24	50	1.18	100	<5	<10	0.18	<0.03	4.10	<5	60	140	390	218	10.5	0.2	0.80
Comp #09	2.0	14.8	180	0.37	300	<5	10	0.15	0.12	0.10	<5	35	130	378	2	7.68	0.2	0.60
Comp #10	4.0	14.6	50	0.87	200	<5	<10	0.15	0.09	0.40	<5	55	200	288	62	11.7	0.2	1.00
Comp #12	1.2	8.28	740	0.47	700	<5	<10	0.54	0.12	0.20	<5	35	100	210	78	6.32	0.2	2.80
Comp #14	2.0	10.8	30	0.82	500	<5	30	<0.03	<0.03	0.10	<5	100	120	582	44	8.14	0.3	2.20
Comp #16	4.2	11.2	60	0.59	400	<5	<10	0.06	<0.03	0.20	<5	75	200	496	126	14.4	<0.1	1.00
Comp #17	0.9	9.6	20	1.04	200	<5	<10	0.15	0.12	3.00	<5	55	170	300	122	10.2	0.1	1.80
Comp #18	6.3	3.8	100	11.4	600	<5	30	1.47	<0.03	1.70	<5	40	220	4904	2198	6.26	0.3	1.80
Comp #19	4.5	3.20	910	3.99	300	<5	<10	2.37	<0.03	3.00	<5	50	80	1208	732	8.18	0.2	1.60
Comp #20	0.3	7.6	<10	2.09	500	<5	<10	1.50	<0.03	2.20	<5	15	70	348	234	3.54	0.2	2.20
Comp #25	2.7	6.76	130	4.94	1600	<5	<10	0.93	<0.03	1.70	<5	25	70	1030	610	4.26	0.1	4.20
Comp #26	2.7	5.48	150	4.96	500	<5	40	3.57	<0.03	5.30	<5	40	50	1690	1166	7.02	0.2	2.60
Comp #27	1.5	6.96	20	9.27	600	<5	10	1.20	<0.03	1.80	<5	115	60	1594	796	5.04	0.1	2.20
Comp #28	<0.3	5.6	40	1.15	400	<5	<10	1.86	<0.03	16.2	<5	15	60	58	34	5.86	<0.1	2.60
Comp #29	0.6	6.68	40	0.41	500	<5	30	1.14	<0.03	15.2	<5	25	160	130	100	4.52	<0.1	0.80
Comp #30	0.9	6.6	20	0.31	500	<5	20	1.11	<0.03	3.70	<5	15	80	112	100	2.94	<0.1	2.40
Comp #31	0.6	3.28	30	1.98	400	<5	20	0.63	<0.03	2.60	<5	10	90	72	54	1.70	<0.1	2.20
Comp #32	0.6	7.00	20	0.32	800	<5	<10	0.57	<0.03	3.20	<5	20	90	94	82	3.26	<0.1	3.60

Table 13.5: Program 3 Variability Sample Head Assays – Part 2

Sample ID	Li	Mg	Mn	Mo	Na	Ni	P	Pb	S-tot	S2-	Sb	Si	Sr	Te	Ti	V	Y	Zn
	ppm	%	ppm	ppm	%	ppm	ppm	ppm	%	%	ppm	%	ppm	ppm	%	ppm	ppm	ppm
Comp #01	20	3.16	1000	5	1.63	80	1000	35	1.00	0.88	1.5	24.5	320	0.8	0.64	210	<100	158
Comp #03	15	1.56	300	<5	2.76	30	1100	30	0.20	0.20	2.3	29.0	438	3.2	0.30	78	<100	50
Comp #04	20	3.20	900	5	1.91	80	1100	40	1.44	1.16	2.1	22.4	254	1.8	0.48	186	<100	108
Comp #05	5	0.12	600	60	0.17	25	600	15	0.06	0.04	13.3	28.8	40	2.8	0.32	150	<100	36
Comp #06	30	2.76	1400	10	0.07	90	1200	10	2.84	2.38	8.2	22.2	160	1.8	0.80	190	<100	92
Comp #07	20	2.60	1100	<5	1.35	80	1200	10	2.26	1.86	2.3	22.6	320	1.8	0.70	224	<100	76
Comp #09	10	0.08	400	<5	0.016	15	1200	15	<0.02	<0.02	4.3	22.9	68	0.2	1.48	316	<100	42
Comp #10	45	1.40	600	10	0.014	110	1100	<5	0.62	0.56	3.7	22.2	50	0.8	1.04	342	<100	104
Comp #12	10	0.60	600	45	0.10	45	700	15	0.68	0.60	9.9	29.1	40	0.6	0.5	126	<100	44
Comp #14	15	0.52	600	5	0.030	60	1100	<5	0.02	0.02	2.1	26.3	72	0.6	0.94	250	<100	62
Comp #16	20	0.16	2200	20	0.022	85	1600	15	0.32	0.32	4.5	21.5	14	0.8	0.80	262	<100	66
Comp #17	35	2.68	900	10	1.32	95	1200	15	0.78	0.68	2.1	22.1	440	1.8	0.80	210	<100	62
Comp #18	15	1.92	500	20	0.488	85	700	20	3.48	3.28	49.7	26.4	200	2.0	0.24	62	<100	36
Comp #19	<5	1.80	500	30	0.162	65	900	25	5.14	4.88	1.9	26.3	148	3.2	0.32	68	<100	48
Comp #20	15	1.52	400	<5	3.12	30	800	<5	0.18	0.18	1.2	27.5	320	0.8	0.32	72	<100	18
Comp #25	15	1.48	200	<5	1.39	30	900	10	0.54	0.52	2.3	24.9	282	3.8	0.28	66	<100	32
Comp #26	5	2.48	1000	20	0.280	30	1100	50	2.24	2.12	15.4	22.8	200	2.4	0.56	106	<100	34
Comp #27	10	1.28	300	15	3.05	35	700	20	2.70	2.66	0.8	27.3	280	12.6	0.28	56	<100	18
Comp #28	5	1.64	900	30	0.572	25	500	10	0.16	0.10	1.2	20.1	560	2.0	0.28	54	<100	40
Comp #29	5	2.80	900	5	1.41	40	700	30	0.42	0.34	2.7	20.9	1000	5.2	0.36	98	<100	50
Comp #30	20	1.56	500	20	2.24	20	800	25	0.54	0.46	0.5	28.5	440	7.8	0.36	74	<100	34
Comp #31	10	1.04	300	10	0.82	20	500	20	0.24	0.22	0.5	20.3	200	3.2	0.20	58	<100	20
Comp #32	15	1.64	400	<5	2.42	25	1200	30	0.42	0.36	0.5	26.4	390	1.8	0.38	94	<100	44

Figure 13.4 and Figure 13.5 present scatter plots of gold versus copper and sulphur versus copper respectively. The gold versus copper shows a significant trend of increasing copper grade with increasing gold grade. A similar trend exists between total sulphur and copper, although the correlation is not as strong.

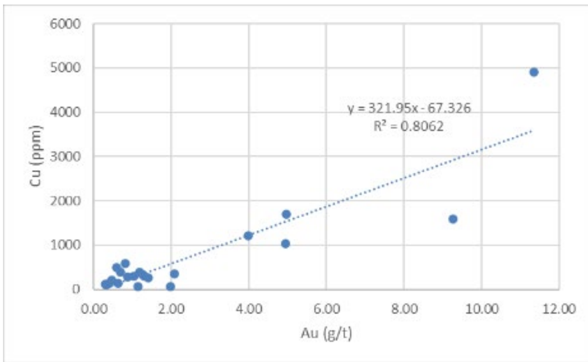

Figure 13.4: Head Grade Analysis, Au vs Cu

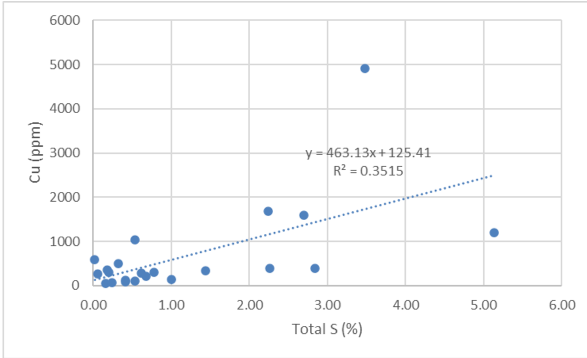

Figure 13.5: Head Grade Analysis, S vs Cu

Traditional deleterious element that present issues in gold processing (Hg, Te, Cd, As) are generally low and not considered to be high enough to present a significant issue. No sample measured greater than 5ppm Cd, and the range of grades are presented in Figure 13.6.

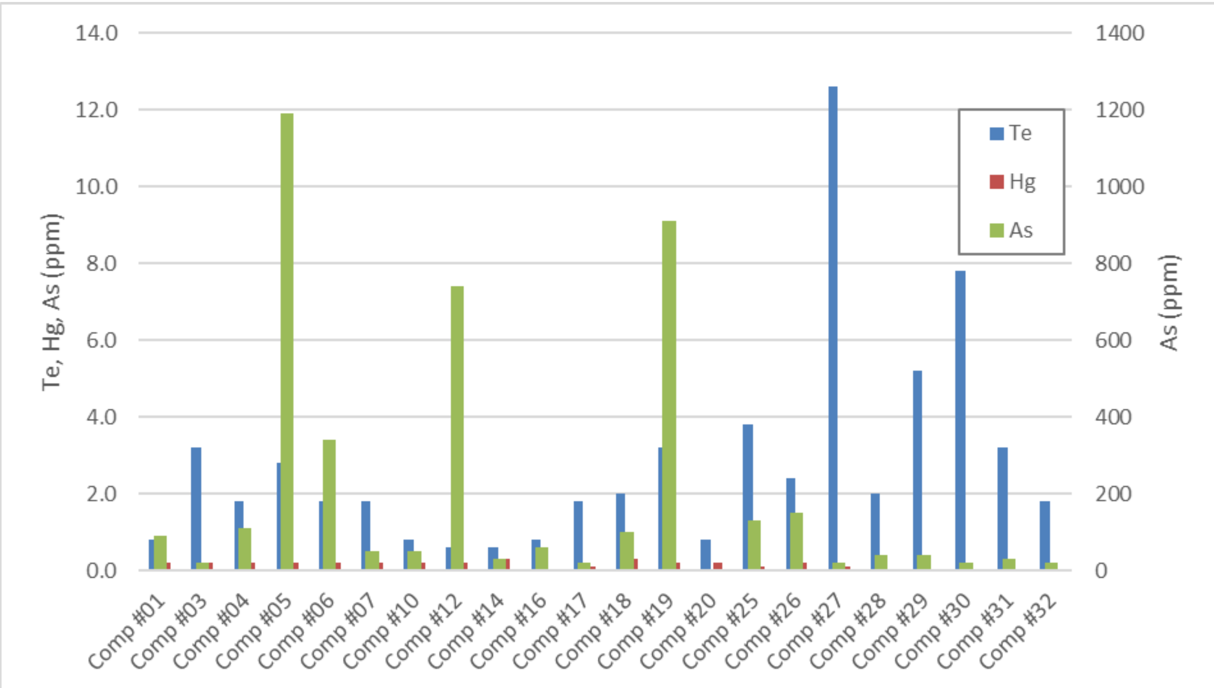

Figure 13.6: Deleterious Element, by Sample

13.4 Comminution Testwork

Samples from both Program 1 and Program 3 were selected for comminution testing using typical techniques, including:

- SMC.
- Bond rod mill work index.

- Bond Ball Mill Work Index.
- Bond abrasion index.
- Specific gravity.

As the full SMC and Bond testwork suite requires large samples only a limited number of samples have been tested with the standard comminution testing procedures. The Geopyora test requires less sample, and therefore more variability samples could be tested using this procedure to expand the dataset available to reduce the design risk.

Within the combined dataset are five samples that were tested by both the Geopyora test and the traditional SMC and Bond work index procedure (paired) so that a comparison could be made between the techniques.

A summary of the comminution data available for circuit design is provided in Table 13.6 and Table 13.7. Table 13.6 presents the results from full SMC and Bond work index testing and Table 13.7 presents the calculated SMC and BWi results as calculated by the Geopyora test.

Table 13.6: SmC, BWi, RWi and Ai Results – Full Test

Lithology	Composite	Axb	BWi[1] kWh/t	RWi kWh/t	SG	Ai, g
Altered Tonalite	PDG MT - 001	28.4	24.8	24.4	2.75	0.400
Mafic Volcanics	PDG MT - 002	30.9[3]	18.5	21.3	2.75	0.300
Altered Tonalite	PDG MT - 003	30.9	21.7	22.8	2.75	0.400
Saprolite[2]	PDG MT – 004	150	3	3	2.6	-
Saprolite[2]	PDG MT – 006	150	3	3	2.6	-
Skarn, Marble, Tonalite, Quartz	PDG MT - 007	35.7	21.1	22.3	2.76	0.345
Skarn, Marble, Tonalite, Quartz	PDG MT - 008	35.1	21.1	22.3	2.76	0.489
Altered Tonalite	PDG MT - 009	23.8	21.2	24.0	2.86	0.500
Altered Tonalite	PDG MT - 010	23.5	24.3	24.0	2.73	0.300
Mafic Volcanics	PDG MT - 011	22.7	25.4	26.5	2.72	0.300
Laterite	LAT_01	140.1	11.0	-	2.7	-
Laterite	LAT_02	171.1	10.0	-	2.5	-

Lithology	Composite	Axb	BWi[1] kWh/t	RWi kWh/t	SG	Ai, g
Paired with Geopyora						
Tonalite	COMP #03	30.5	20.4	-	2.73	0.3301
Mafic	COMP #04	25.3	25.5	-	2.89	0.1206
Shear	COMP #19	69.1	16.0	-	2.88	0.1391
Tonalite	COMP #25	44.0	20.1	-	2.76	0.1971
Tonalite	COMP #32	32.5	19.7	-	2.72	0.2437

Notes:

1. Closing-screen 106µm
2. Benchmarked ore characteristics
3. Assumed value from the same Hole BNERD007

Table 13.7: Geopyora Test Results

Lithology	Composite	Axb	BWi[1] kWh/t	SG
Paired with Standard Comminution Suite				
Tonalite	COMP #03	27.0	19.0	2.73
Mafic	COMP #04	20.9	22.1	2.86
Shear	COMP #19	38.9	16.1	2.85
Tonalite	COMP# 25	26.3	18.9	2.77
Tonalite	COMP #32	25.4	19.9	2.71
Non-Paired Geopyora				
Mafic	COMP #01	22.9	22.6	2.91
Shear	COMP #05	104.2	9.9	2.57
Saprock	COMP #06	81.6	11.5	2.58
Mafic	COMP# 07	21.0	23.0	2.96
Shear	COMP #12	121.8	10.0	2.63
Mafic	COMP #17	45.4	15.7	2.68
Shear	COMP #18	35.0	16.5	2.85
Tonalite/Int	COMP #20	23.4	20.7	2.76
Shear	COMP #26	37.3	15.8	2.85
Tonalite	COMP #27	27.8	19.3	2.76
Skarn	COMP #28	25.2	18.3	3.02
Skarn	COMP #29	17.9	22.0	3.04
Quartz Vein	COMP #30	24.8	20.1	2.72

Lithology	Composite	Axb	BWi[1] kWh/t	SG
Quartz Vein	COMP #31	33.1	16.8	2.68

Notes:

1. Closing-screen 106µm

In order to justify the use of Geopyora data, the results of the five paired tests were compared as presented in Figure 13.7.

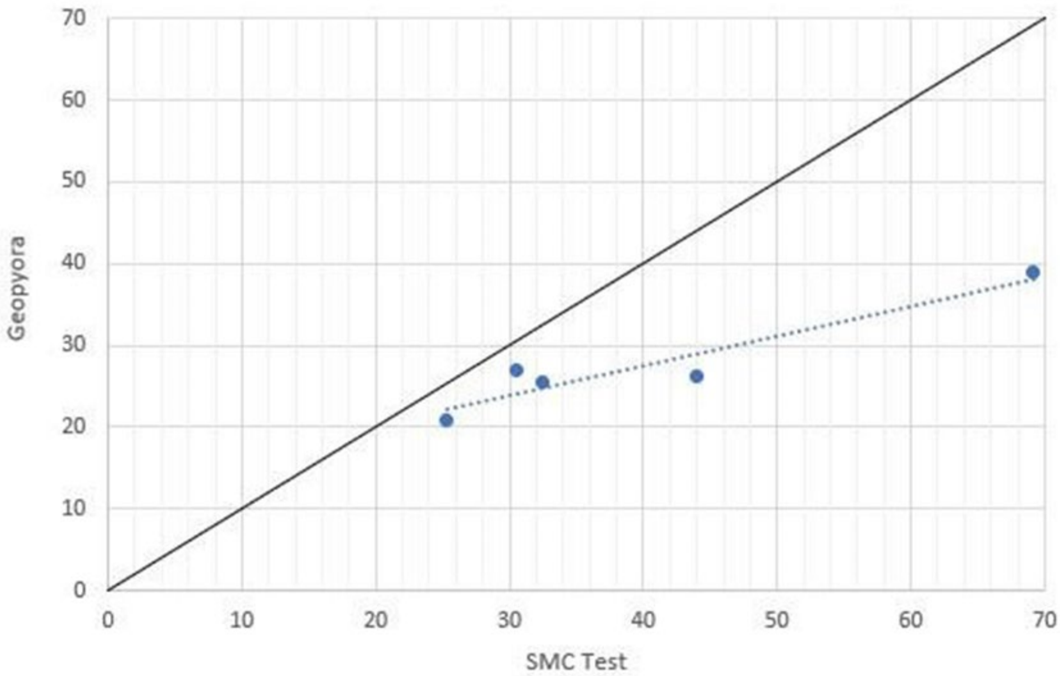

Figure 13.7: Axb Comparison – SMC vs. Geopyora

Orway Mineral Consultants (OMC) concluded (Orway Mineral Consultants, 2025) that the Axb results are comparable for samples of very high ore competency (with Axb between 25 and 35) as determined by the SMC, with the Geopyora test results appearing to show less variability. Therefore, the Geopyora method provides more conservatively estimates at higher ore competency for the same dataset compared to the SMC method. With this understanding, it was deemed appropriate to use the Geopyora test results in the dataset to define comminution characteristics

OMC reviewed the available comminution data (Orway Mineral Consultants, 2024) (Orway Mineral Consultants, 2025) and made the following conclusions:

- Fresh ore samples are extremely competent. In terms of Axb (with lower values indicating harder material) the 85th percentile value is representative of the top 5% of materials tested in the OMC database, and are also extremely hard in terms of grindability, with Bond ball work indices, also in the top 5%.
- The number of fresh samples tested for Axb an BWi, and included in determining the design point, is suitable for this level of study, however additional testwork on oxide lithology types

(primarily laterite and saprolite) would enable a better understanding power requirements to achieve a relatively fine target grind size, as well as expected variability.

- Average abrasion index is in the most abrasive 20% of the OMC database.
- Laterite lithology samples tested demonstrated very low competency, owing to the heavily weathered state with BWi in the lower 14th percentile of the OMC database.
- Saprolite lithology samples were very fine, with 65% already at product size (<75 µm), and too soft/fine for comminution testwork. In the absence of testwork data, ore characteristics have been benchmarked. The ore has been classified as having very low ore competency and grinding requirements. A BWi of 2 kWh/t, though not likely to be an accurate result, validates the benchmark value of 3 kWh/t nominated for design.

It is worth noting that the saprolite samples, whilst unsuitable for comminution testing, consistently presented as a "plasticine" consistency with high moisture content that would be expected to present materials handling issues because of the sticky nature of this ore (see further discussion in Section 13.7) and may warrant a different feed/crushing arrangement than the other, more competent ore types.

13.5 Leach Testwork

13.5.1 Grind Size Optimisation

During Program 1, grind size optimisation was conducted considering grind size products (P_{80}) of 150, 106, 75 and 53µm on composites of the altered tonalite and saprolite created for this testwork program. The results from this program, which did not include gravity recovery prior to leaching, are presented in Table 13.8.

Table 13.8: Program 1 (Mintrex) Grind Size Optimisation Leach Test Results

Composite[1]	Lithology	Grind Product P80 (µm)	Assay Head Grade (g/t)	Calc'd Head Grade (g/t)	Tail Grade 24 h (g/t)	Recovery Au 24 h (%)
Comp A	Altered Tonalite	53	2.79	2.87	0.15	94.6%
Comp A	Altered Tonalite	75	2.79	2.64	0.17	**93.6%**
Comp A	Altered Tonalite	106	2.79	2.42	0.25	89.8%
Comp A	Altered Tonalite	150	2.79	2.35	0.34	85.5%
Comp B	Saprolite	53	1.00	1.08	0.05	95.4%
Comp B	Saprolite	75	1.00	1.25	0.13	**89.9%**
Comp B	Saprolite	106	1.00	1.14	0.06	94.7%
Comp B	Saprolite	150	1.00	1.11	0.09	92.3%

Notes:

1. Composites as per Mintrex testwork program (Mintrex, 2021)

This program concluded that there was minimal value gained for a grind sizes (P_{80}) finer than 75µm. This conclusion was influenced by the rationale that the saprolite test at 75µm appeared anomalous, and the majority of the resource is represented by the altered tonalite composite.

However, Mintrex recommended that the optimal grind size be investigated further, and two more samples were tested, this time including gravity recovery of gold prior to leaching, in Program 2 (IMO, 2024). The two samples tested in Program 2 were Composite A and C as presented in Table 13.2, and the results are summarised in Table 13.9.

Table 13.9: Program 2 (IMO) Grind Size Optimisation Leach Test Results

Composite	Lithology	Grind Product P80 (µm)	Assay Head Grade (g/t)	Calc'd Head Grade (g/t)	Gravity Recovery (%)	Recovery Au 24 h (%)
Composite A	Tonalite	150	2.37	2.30	36.7	86.9
		106		1.97	39.2	87.3
		75		**2.06**	**52.2**	**94.7**
Composite C	Mafic	150	2.16	2.04	34.9	92.0
		106		1.84	39.2	90.2
		75		**1.89**	**48.0**	**93.2**

The results presented in Table 13.9 confirmed that a grind product P_{80} of 75 µm is an appropriate design target with the increase in extraction and gravity recovery of gold far outweighing the likely additional costs associated with the finer grind.

13.5.2 Gravity Gold Recovery

Program 1 (Mintrex, 2021) assessed the potential for gravity gold recovery using a laboratory scale Knelson concentrator, followed by amalgamation using mercury. The results of these tests are presented in Table 13.10.

Table 13.10: Program 1 (Mintrex) Gravity Gold Recovery

Deposit	Sample ID	Lithology	Grind Size	Gravity Recovery
NEB	PDG-MT-001 (A)	Altered Tonalite	NA	31.8%
NEB	PDG-MT-002 (B)	Altered Tonalite	NA	20.0%
NEB	PDG-MT-003 (C)	Altered Tonalite	NA	19.6%
NEB	PDG-MT-004/006 (D/F)	Altered Tonalite	NA	13.1%
NEB	PDG-MT-005 (E)	Altered Tonalite	NA	33.8%
BC	PDG-MT-007/008 (G/H)	Altered Tonalite	NA	32.8%
NEB	PDG-MT-009 (I)	Altered Tonalite	NA	37.0%
NEB	PDG-MT-010 (J)	Altered Tonalite	NA	26.5%
NEB	PDG-MT-011 (K)	Altered Tonalite	NA	28.1%

Deposit	Sample ID	Lithology	Grind Size	Gravity Recovery
NEB	Comp 1	Altered Tonalite	180/75	30.0%
NEB	Comp 2	Altered Tonalite	180	40.6%
NEB	Comp 3	Altered Tonalite	180	28.0%
NEB	Comp 4	Altered Tonalite	180/75	39.6%
NEB	Comp 5	Altered Tonalite	180	35.5%
NEB	Comp 6	Altered Tonalite	180/75	41.3%
NEB	Comp 7	Altered Tonalite	180/75	45.9%
NEB	Comp 8	Altered Tonalite	180/75	38.8%
NEB	Comp 9	Altered Tonalite	180	39.4%

The average gravity recovery of gold achieved was 32% across a range of 13% to 46%. Five samples were also subjected to paired comparison of leaching with and without gravity recovery to determine if the inclusion of gravity recovery improved overall recovery. The results of these paired tests are presented in Table 13.11.

Table 13.11: Comparison of Leach Extraction with and without Gravity Recovery

Sample	Au Head Assay (g/t)	Recovery Without Gravity Prior to Leaching (%)	Gravity Recovery (%)	Recovery With Gravity
Comp 1	3.29	93.1	30.0	93.4
Comp 4	22.3	93.9	39.6	96.5
Comp 6	1.43	93.0	41.3	95.6
Comp 7	1.52	89.8	45.9	96.1
Comp 8	3.60	90.8	39.8	94.8
Average		**91.7**	**39.1**	**95.0**

The results in Table 13.11 clearly demonstrate that the application of gravity recovery prior to leaching is likely to increase overall gold recovery by greater than 3%, on average, although it is noted only samples Comp 6 and Comp 7 are near to the expected design gold head grade. Importantly, it was noted that the kinetics of the cyanide leaching was consistently increased when gravity recovery was applied, likely because the coarser gold particles that take additional time to fully leach, were already recovered. Full detailed gravity and leach extraction plots can be found in the report from Como Engineers (Como Engineers, 2024).

Based on these results, it was concluded that a gravity circuit is justified and the latter variability program (Section 13.5.6) that defined metallurgical performance across the resources and lithologies included a gravity recovery stage.

13.5.3 Cyanide Concentration

The impact of cyanide concentrations was investigated using results from Program 1 and Program 2, reported separately by Como Engineers (Como Engineers, 2024) and IMO (IMO, 2024), with some anomalous results in both programs with regards to defining a trend for the impact of cyanide concentration on cyanide consumption in the leach.

These two programs tested cyanide concentrations (presented as initial concentration/maintained concentration) ranging from 1,500/750 ppm, 1,000/500 ppm, 500/300 ppm and 300/200 ppm using four separate composite samples.

Table 13.12: NaCN Consumption (kg/t) at Various NaCN Concentration Targets

Sample (Program)	NaCN Consumption at Concentration Target Initial / Maintained (ppm)				
	1,500/750	1,000/500	750/500	500/300	300/200
Comp A (Program 1)	2.05	2.47	-	0.56	-
Comp B (Program 1)	1.6	2.52	-	0.96	-
Composite A (Program 2)	-	-	0.34	0.22	0.33
Composite C (Program 2)	-	-	0.51	0.34	0.48
Average Consumption	**1.83 kg/t**	**2.50 kg/t**	**0.43 kg/t**	**0.52 kg/t**	**0.41 kg/t**

Neither program demonstrated a measurable reduction in gold extraction from the lower cyanide concentrations. However, no significant trend for consumption versus cyanide concentration was identified as there were anomalous results in Program 1 at the 1,000/500 ppm condition. For example, the median cyanide consumption from the variability tests carried out in Program 3, conducted under the 1,000/500 NaCN concentrations, was 0.40 kg/t (Section 13.6.2) compared to 2.50 kt/t for these conditions in Program 1.

Typically, lower cyanide concentration in operations result in lower cyanide consumptions, although this is difficult to quantify from the results to date. Therefore, it is reasonable to conclude that basing cyanide consumption on the variability program (Program 3) using CN concentrations of 1,000/500 ppm NaCN is conservative.

It is also worth noting that the bulk leach test conducted as part of Program 2 (IMO, 2024), used to generate bulk samples for initial carbon loading and cyanide destruction testwork, allowed the NaCN concentration in liquor decrease to 120 mg/L without a decrease in gold extraction, as shown in Figure 13.8.

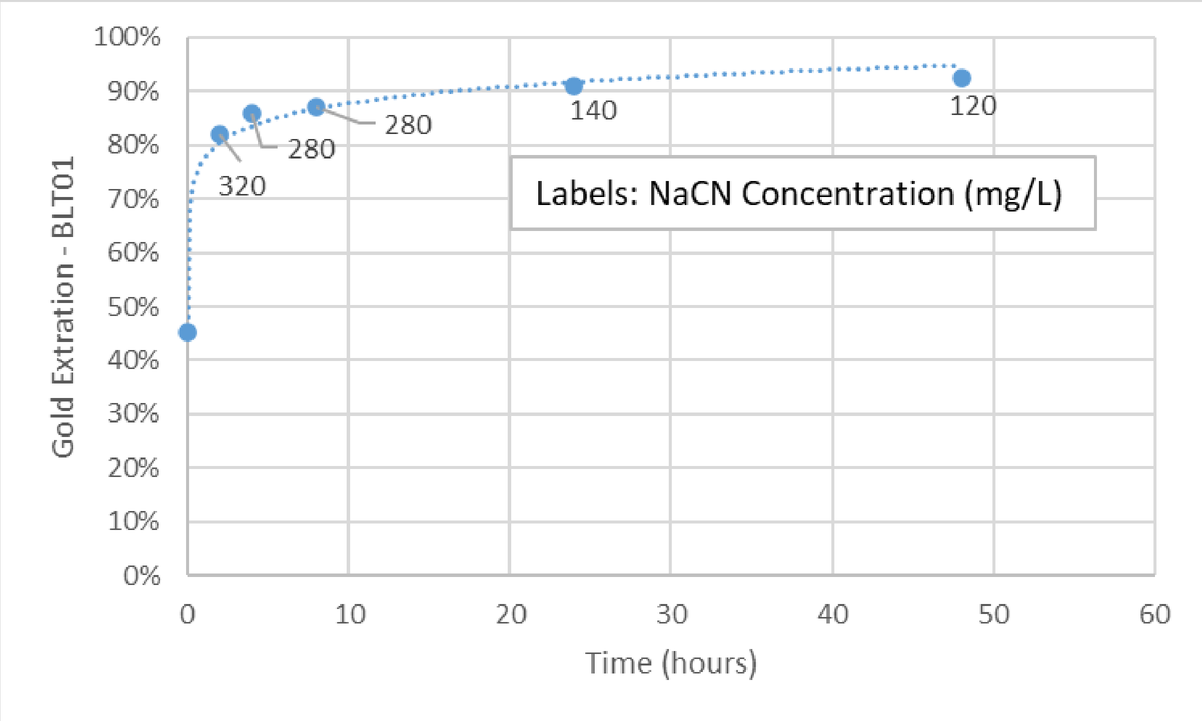

Figure 13.8: Program 2 Bulk Leach Gold Extraction with Decreasing NaCN Concentration

The results of this bulk test do suggest that there is no loss in the leachability of gold at lower concentrations and therefore lower concentrations should be targeted in operations. This conclusion is predicated on there being no other operational factor influencing the need to increase cyanide concentrations (e.g. potential soluble copper that would otherwise limit gold loading onto carbon, discussed in Section 13.12).

13.5.4 Oxygen Versus Air Addition

The impact of oxygen versus air addition has been investigated in Programs 1 and 2 and reported separately by Como Engineers (Como Engineers, 2024) and IMO (IMO, 2024). From both testwork programs, the use of oxygen or air had negligible impact on gold extraction, as demonstrated by comparing the residual gold grades from the relevant tests in Table 13.13. As such air sparging was selected for further testwork and the process design.

Table 13.13:Air vs Oxygen Sparging Residual Gold (g/t) Comparison

CN Conc. Target initial/maintained	Oxygen	Air
Comp A (Program 1)	0.24	0.24
Comp B (Program 1)	0.07	0.07
Composite A (Program 2)	0.17	0.15
Composite C (Program 2)	0.11	0.11

13.5.5 Tailings Diagnostic Leach Tests

Four selected leach residues from the bulk leach extractable gold tests (BLEG) in Program 1 (Como Engineers, 2024) were subjected to simple diagnostic leach analysis to ascertain the form of the remaining gold. The samples were selected to represent a range of lithologies with presented in Table 13.14.

Table 13.14: Program 1 Diagnostic Leach Results

Lithology	Residue Calc Head (g/t)	Free gold (%)	Gold locked in sulphides (%)	Gold locked in quartz (%)
Altered Tonalite	0.42	3.57	79.6	16.8
Mafic Volcanic	0.19	7.77	55.4	36.8
Skarn	0.21	7.03	53.5	39.5
Weathered Saprolite	0.09	33.1	44.3	22.6

The results in Table 13.14 demonstrates majority of the gold remaining in leach residue solids from the altered tonalite, mafic volcanic and skarn samples is locked in sulphides and silicates and that vast majority of the cyanide soluble gold was leached in the BLEG tests. This suggests that, to increase the gold extraction, the ore would either have to undergo ultrafine grinding or a chemical oxidation method, which would be unlikely to be economic given the low residue grades.

While the weathered saprolite composite indicated 33.1% of the gold remaining in leach residue was cyanide soluble, the residual gold grade was low at 0.09 g/t and therefore this is an inherent larger analytical error with the multi-step process and therefore the calculated results.

A more complete diagnostic leach was subsequently completed in Program 2 (IMO, 2024) on three more samples with the results are presented in Table 13.15.

Table 13.15: Program 2 Diagnostic Leach Results Summary

Sample	Composite A	Composite C	Composite F
Residue Au Grade (g/t)	0.14	0.12	0.15
Diagnostic Leach Stage	**Gold Distribution (%)**		
Free Cyanide Soluble Gold	4.53%	4.77%	15.8%
Intensive Cyanide Soluble Gold	0.0%	22.0%	6.13%
Weak Acid Soluble Gold	0.45%	0.48%	0.51%
Dolomite/Pyrrhotite/ Goethite Occluded Gold	13.3%	20.2%	19.7%
Other Refractory Gold	72.4%	31.0%	50.5%
Silicate Locked Gold	9.30%	21.6%	7.42%

The results in Table 13.15 are consistent with the earlier results from Program 1 in that the majority gold remaining in the residual after the bulk leach is not readily extractable with cyanide only at the design grind product P_{80} of 75µm and therefore advanced recovery techniques would be required, which would be unlikely to be economic at the low residue grades.

13.5.6 Gravity and Leach Extraction Variability

Program 3 was focused on the testing of various samples across a range of lithologies and grades in a variability program. All variability samples in Program 3 were metallurgically tested using the following test conditions:

- Primary grind (P_{80}) of 75 µm.
- Gravity recovery with an intensive cyanide leach on gravity concentrate with the residue returned to the bulk leach.
- 48-hour bottle roll leach of gravity tails and intensive leach residue at the following conditions:
  - 50% solids, except for some of the oxide samples which were too viscous, where 40% solids were used.
  - Target pH of 10.5.
  - Initial NaCN concentration of 1,000 ppm (0.1% (w/v)), maintained >500 ppm (0.05% (w/v)).
  - Preleach oxygen sparging targeting 25 ppm dissolved oxygen.
  - Residence time of 24 hours.

It is noted that the cyanide concentrations for the variability program were higher than recommended for the optimised cyanide concentrations (Section 13.5.3). This higher concentration was selected to avoid potential anomalous results from in advertent low cyanide concentrations in the event that one of the samples exhibited high cyanide consumption.

A summary of the variability program results is presented in Table 13.16.

These results have been used as the basis for estimating geometallurgical relationships for reagent consumption (Section 13.6.2) and gold extraction (Section 13.6.3).

Table 13.16: Summary of Variability Test Results

Test #	Comp ID	Deposit	Lithology	P80 (µm)	Au Head Grade (g/t)		Au Extraction (%)			Au Tail Grade (g/t)	Reagents (kg/t)	
					Assay	Calc.	Grav	24-h	48-h		NaCN	Lime
BK21079	COMP #01	NEB (Open Pit)	Mafic	75	0.64	0.76	31.9	92.8	92.8	0.06	0.43	0.29
BK21020	COMP #03	NEB (Open Pit)	Tonalite	75	1.30	1.77	55.8	97.2	97.5	0.05	0.31	0.16
BK21021	COMP #04	NEB (Open Pit)	Mafic	75	1.26	1.09	29.6	92.8	93.6	0.07	0.52	0.29
BK21080	COMP #05	NEB (Open Pit)	Shear	75	1.40	1.69	25.1	91.5	92.0	0.14	0.27	0.24
BK21081	COMP #06	NEB (Open Pit)	Saprock	75	0.69	0.55	36.8	91.9	92.7	0.04	0.74	2.33
BK21082	COMP #07	NEB (Open Pit)	Mafic	75	1.18	0.73	41.3	93.8	93.8	0.05	0.62	0.42
BK20979	COMP #09	NEB (Open Pit)	Shear	75	0.37	0.42	21.2	81.2	81.2	0.08	0.25	1.88
BK20980	COMP #10	NEB (Open Pit)	Saprolite	75	0.87	1.32	20.6	88.1	88.6	0.15	0.40	1.78
BK21083	COMP #12	NEB (Open Pit)	Shear	75	0.47	0.45	28.4	90.8	91.8	0.04	0.44	0.41
BK20981	COMP #14	NEB (Open Pit)	Saprolite	75	0.82	1.00	28.0	92.5	92.5	0.08	0.40	1.41
BK20982	COMP #16	NEB (Open Pit)	Saprolite	75	0.59	0.48	37.2	92.9	93.8	0.03	0.39	2.73
BK21084	COMP #17	NEB (Open Pit)	Mafic	75	1.04	1.06	29.3	89.6	90.1	0.11	0.48	0.55
BK21085	COMP #18	NEB (Open Pit)	Shear	75	11.35	9.80	54.4	97.6	97.2	0.27	0.98	0.24
BK21022	COMP #19	NEB (Open Pit)	Shear	75	3.99	3.41	31.6	91.9	91.9	0.28	0.62	0.22
BK21086	COMP #20	NEB (Open Pit)	Tonalite/Int	75	2.09	1.24	40.3	94.8	95.2	0.06	0.40	0.20
BK21023	COMP #25	NEB (UG)	Tonalite	75	4.9	3.95	46.7	96.5	96.7	0.13	0.57	0.15
BK21087	COMP #26	NEB (UG)	Shear	75	4.96	4.29	48.2	93.9	94.3	0.25	0.74	0.28
BK21088	COMP #27	NEB (UG)	Tonalite	75	9.27	7.29	57.6	95.2	95.8	0.31	0.65	0.21
BK21089	COMP #28	BC	Scarn	75	1.15	1.04	24.5	92.3	92.3	0.08	0.22	0.15
BK21090	COMP #29	BC	Scarn	75	0.41	0.45	44.8	92.5	94.5	0.03	0.29	0.18

Test #	Comp ID	Deposit	Lithology	P80 (µm)	Au Head Grade (g/t)		Au Extraction (%)			Au Tail Grade (g/t)	Reagents (kg/t)	
					Assay	Calc.	Grav	24-h	48-h		NaCN	Lime
BK21091	COMP #30	BC	Quartz Vein	75	0.31	0.30	28.3	81.6	81.6	0.06	0.31	0.15
BK21092	COMP #31	BC	Quartz Vein	75	1.98	1.57	32.6	90.2	90.8	0.15	0.24	0.13
BK21024	COMP #32	BC	Tonalite	75	0.32	0.28	36.9	89.5	89.5	0.03	0.29	0.15

Other elements in solution, that are relevant to design, include copper and iron in solution due to their impact cyanide destruction (Section 13.13). The variability dataset provides the best reference for the range of concentrations of these species measured after 24 hours of leaching, and histograms of copper and iron concentration in solution which are presented in Figure 13.9 and Figure 13.10.

The histogram of soluble copper shows a non-normal distribution, which is expected as the samples were selected to span the range of ore characteristics and assays from discrete samples, not match the distribution of characteristics and assays in the proposed LOM production schedule.

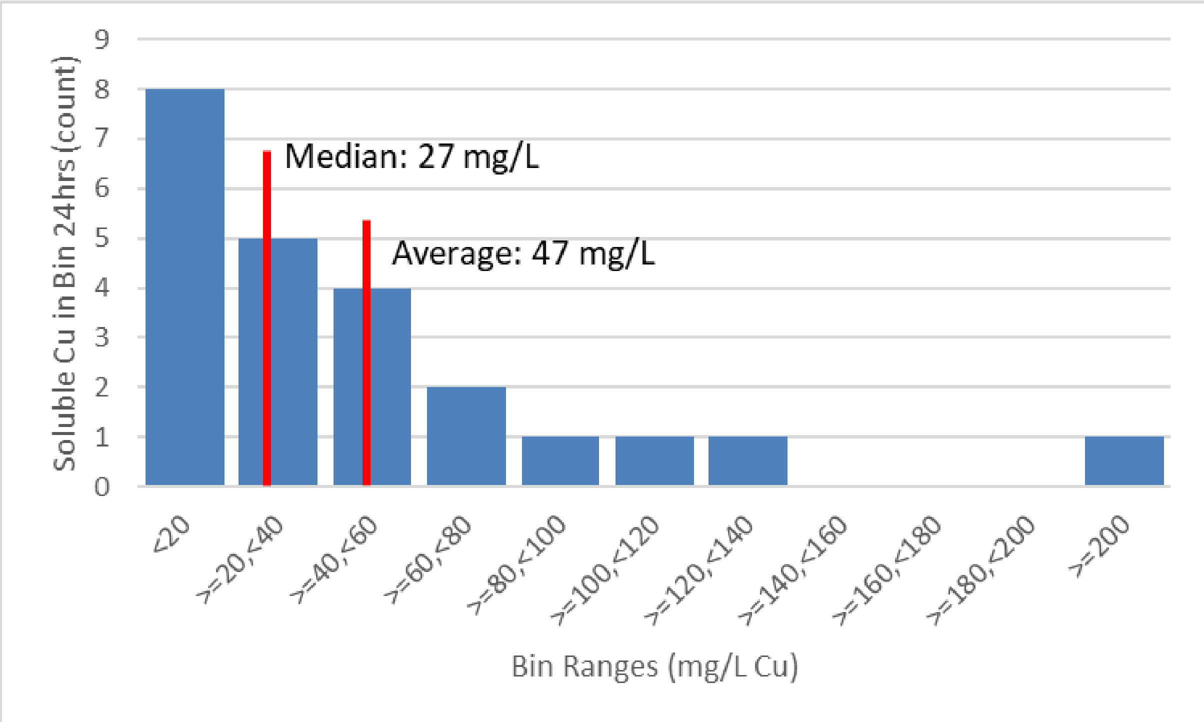

Figure 13.9: Copper in Solution Histogram

In these results for copper in solution, the average is biased by the outlier samples and therefore is not likely to be representative of the midpoint of the data set. As a result, the median dissolved copper grade, and therefore median cyanide consumption, given copper is a key cyanide consuming species, for the dataset is a more appropriate measure of the midpoint of the dataset than the average.

The median copper in solution is 27 mg/L, which has implications for estimates of cyanide destruction reagent consumption (Section 13.13).

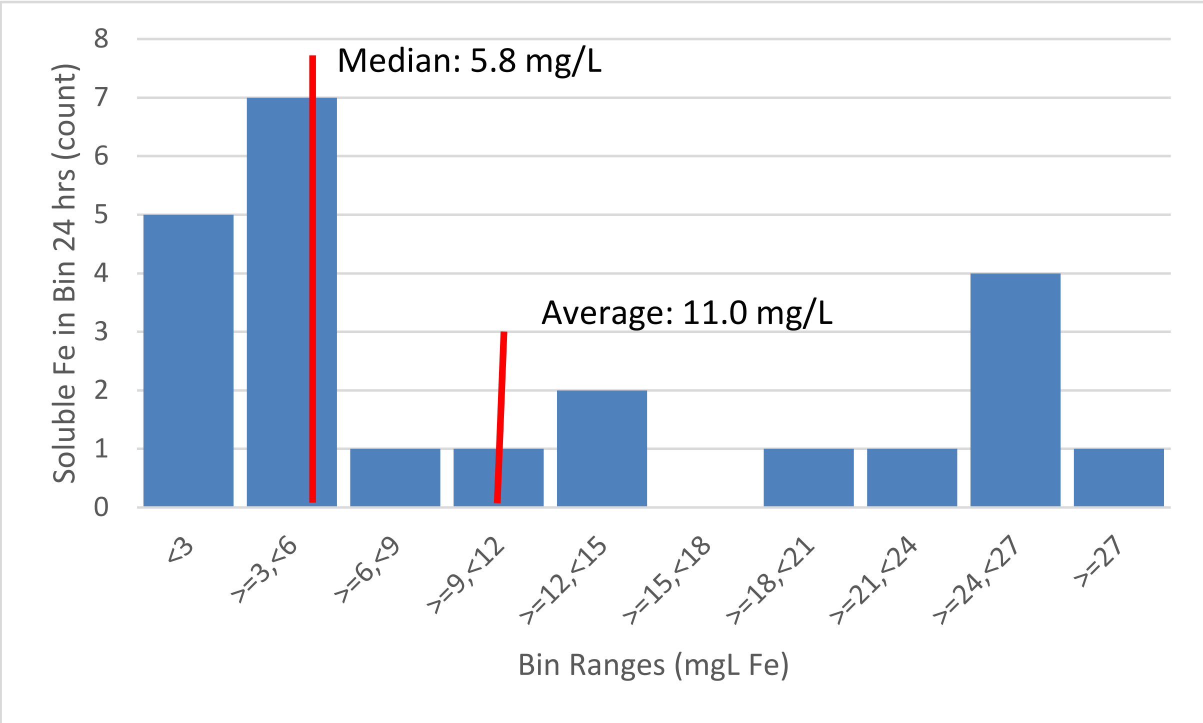

Figure 13.10: Iron in Solution Histogram

Whilst the distribution of iron in solution presented in the histogram in Figure 13.10 does not present a normal distribution, it is not as skewed as with the copper dataset, and the therefore the average of 11.0 mg/L Fe is an more appropriate midpoint than the median.

13.6 Geometallurgical Relationships

13.6.1 Power Demand

The modelled power demand for various blends was calculated by OMC (Orway Mineral Consultants, 2025) and has been used to estimate a comminution circuit power draw per lithology based on the median (P_{50}) comminution characteristics giving the following values:

Table 13.17: Lithology Median (P_{50}) Specific Comminution Power Draw

Lithology	Spec Energy (kWh/t)
Fresh	29.2
Laterite, Saprock, Shear Zone	13.1
Saprolite	5.6

It is worth noting that the installed milling power has been designed to cater for the specific energy of the 85th percentile of hardness results on the design blend of 70% fresh ore and 25% laterite ore with no saprolite giving a designed installed milling power of 20 MW (10 MW SAG and 10 MW ball mill).

However the operation will be required to contend with variable ore breakage characteristics as the blend varies. Therefore, when considering the average operating conditions (which impact operating costs), the average median hardness (50th percentile or P_{50}) is a relevant measure. Table 13.18 presents the expected milling power demand for a range of 50th percentile blends anticipated to be encountered over the life of mine.

Table 13.18: Lithology Blend Operating Grinding Power Estimates

Design Condition	BlendFresh : Lat : Sap	Spec Energy (kWh/t)	Total Energy (MW)	% Design Power
Design (P_{85})	75% Fresh 25% Laterite	29.6	16.7	100%
Design (P_{50})	75% Fresh 25% Laterite	25.1	14.1	84%
Hardest (P_{50})	100% Fresh	29.2	16.4	98%
Softest (P_{50})	32% Fresh 28% Laterite 40% Saprolite	15.2	8.5	51%

Table 13.1 demonstrates that the conservative comminution circuit design should allow full design tonnage whilst maintaining the design grind product on average, for all ore blends anticipated to be encountered across the life of mine.

To accommodate this variability in power demand related to the ore lithology, the variable power draw by lithology presented in Table 13.17 is used in the estimation of mill power draw based on the lithological feed blend, with other power use in the processing plant considered fixed.

13.6.2 Reagent Consumption

Two components of reagent consumption require estimation to develop an overall estimate of reagent consumption, being:

- Losses due to direct consumption during the leaching phase.
- Losses attributed to rejection or destruction of reagents that remain in the leach tailings liquor.

The reagent consumption throughout the leach phase has been estimated from the variability testing carried out in Program 3. Figure 13.11 and Figure 13.13 present the variability in cyanide and lime consumption respectively across the range of lithologies tested, with the red bars representing the weathered (predominately saprolite and laterite) lithologies.

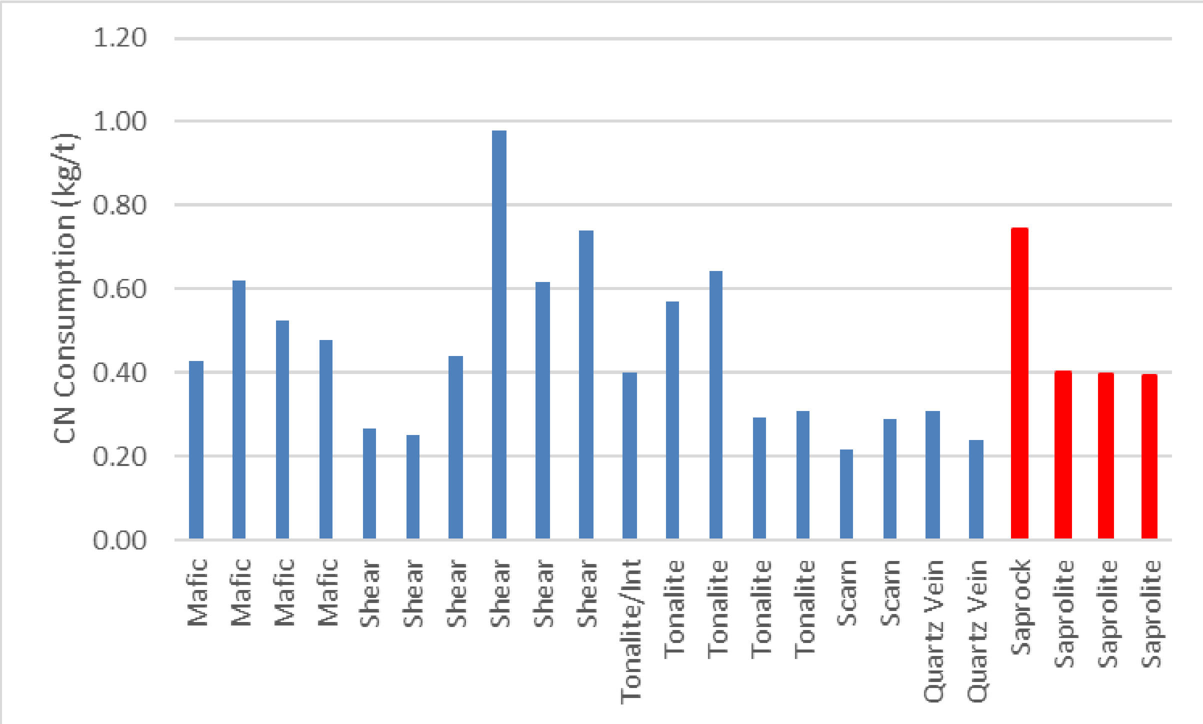

Figure 13.11: Cyanide Consumption by Lithology

Based on these results, lithology cannot be statistically identified as a driver for cyanide consumption. However, a review of the cyanide consumption and copper in solution data, presented in Figure 13.12, demonstrates that there is a strong relationship between these two variables.

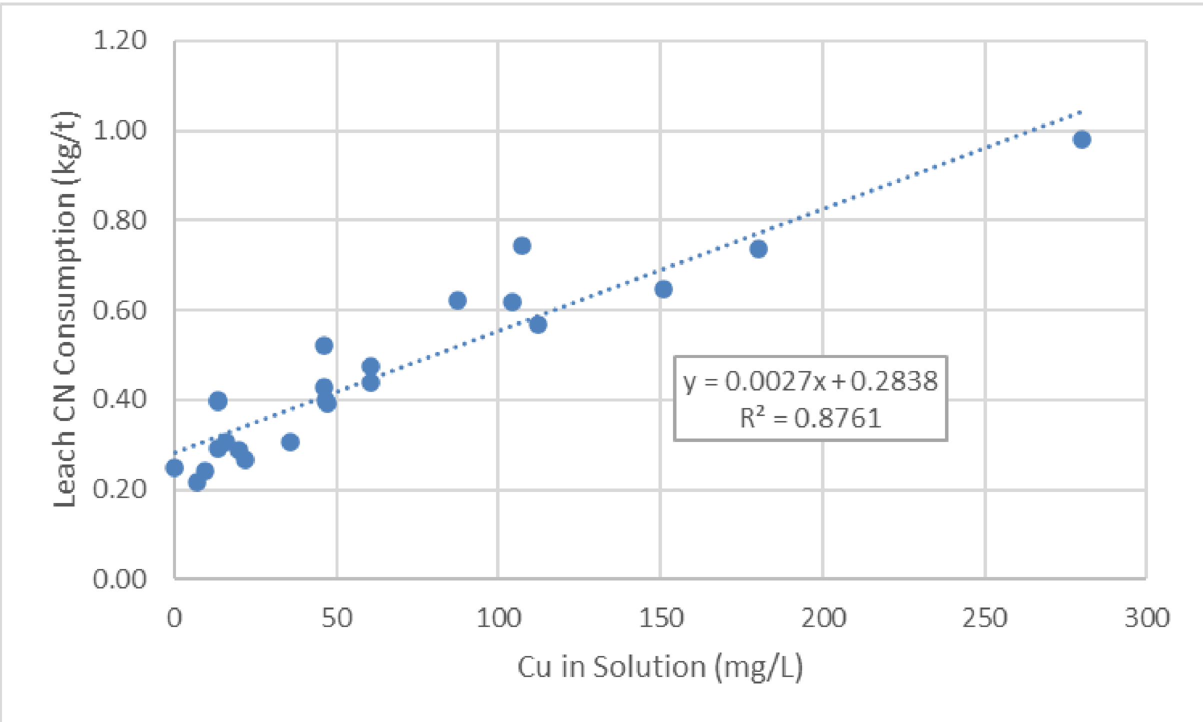

Figure 13.12: Cyanide Consumption versus Copper in Solution

Unfortunately, no method of calculation for the prediction of cyanide soluble copper has been identified as it is not correlated to copper head grade, or lithology. As such, the median of the cyanide consumption dataset from the variability testwork of 0.40 kg/t is deemed the appropriate measure of the midpoint of the available data, for the same reason that the median is selected to represent the midpoint (Section 13.5.6) for copper.

The remaining cyanide loss is determined by the mass balance where the residual cyanide concentration required for optimal leaching and carbon loading (selected as 150 ppm CN) is destroyed in detoxification and therefore lost to the process.

Adding this loss to the cyanide consumption gives a cyanide addition requirement of 0.60 kg/t at the design throughput of the facility.

The results for lime consumption across the variability dataset is presented in Figure 13.13. This shows that there is a significant difference between the lime consumption for fresh lithologies (mafic, shear, tonalite, skarn and quartz vein) compared to weathered lithologies (saprolite and saprock). The outlying result for the one sample of shear may be due to a higher degree of weathering of this sample compared to other samples, however it is not considered significant due to the relatively low proportion of shear in the orebody.

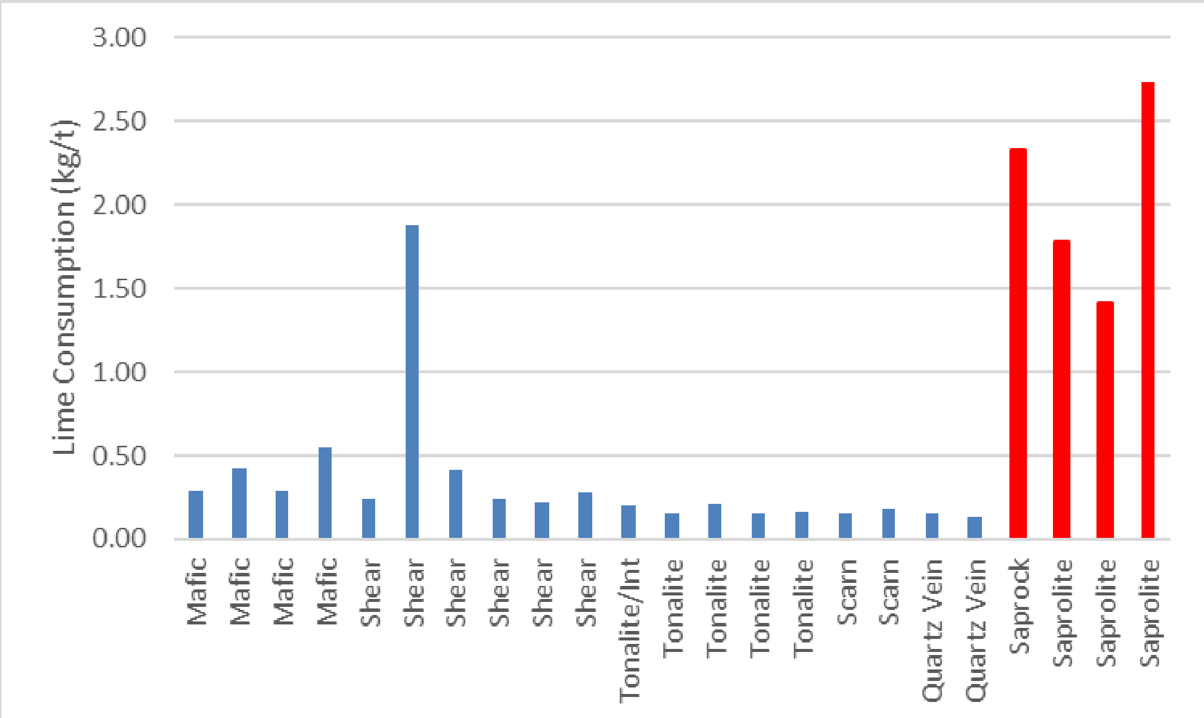

Figure 13.13: Lime Consumption by Lithology

Consequently, the best prediction of lime consumption was determined to be separate average values for the fresh ore lithologies and the weathered ore lithologies. No losses are required to be considered for lime as the testing directly measures the addition required to meet the target pulp pH for safe operation.

The predicted lime consumption determined for the modelling of costs is:

- Fresh ore, 0.33 kg/t.

- Weathered ore, 2.06 kg/t

13.6.3 Gold Extraction

A number of simple models were assessed for the prediction of gold extraction, and it was found that the most reliable model predicted the 24-hour gold extraction using a linear regression against the measured head Au grade (g/t). The 24-hour gold extraction versus head gold grade for all tests in the variability program are presented in Figure 13.14, classified by deposit

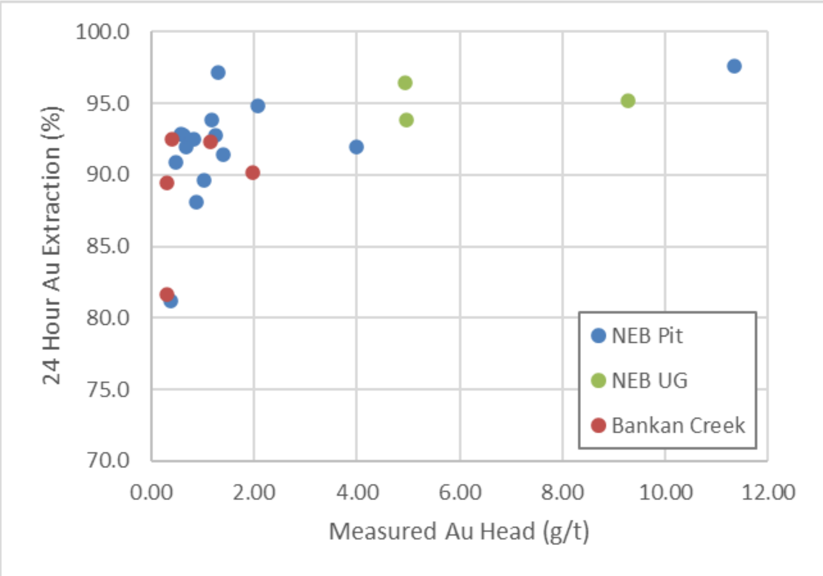

Figure 13.14: 24-Hour Gold Extraction versus Head Grade by Deposit

Figure 13.14 suggests there is no evidence to suggest that there is justification to separate the data by deposit to improve the accuracy of gold extraction estimation. A similar analysis by lithology (Figure 13.15) resulted in the same observation.

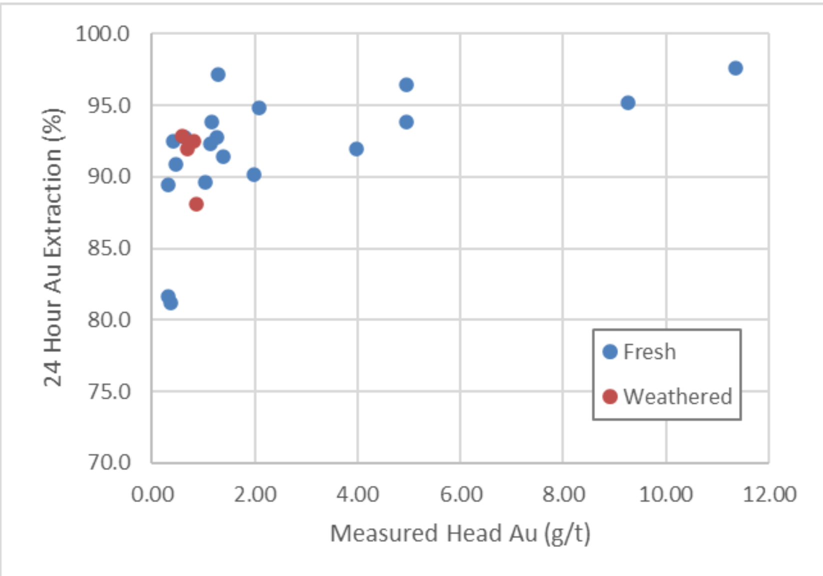

Figure 13.15: 24-Hour Gold Extraction versus Head Grade by Lithology

Within the dataset, there are two tests, Comp #09 and Comp #30, that present a lower-than-expected gold extraction compared to the remainder of the dataset. Both were very low-grade samples (with head grades of 0.42 g/t and 0.30 g/t respectively) that would significantly impact the estimated gold extraction relationship if included in the dataset and as such these results were excluded from the analysis.

Figure 13.16 presents the same data as Figure 13.14, except for outlying samples, with the linear best fit of the data included.

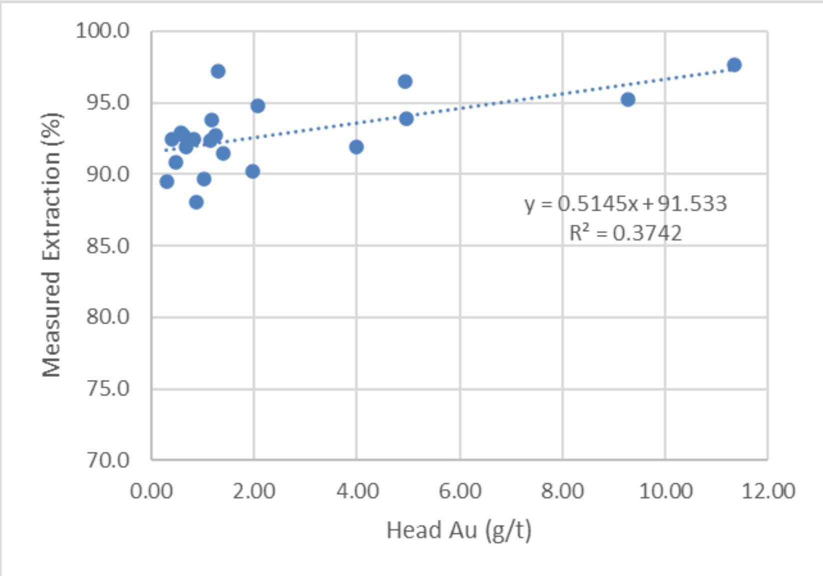

Figure 13.16: 24-Hour Extraction versus Head Grade

The relationship for the linear regression presented, which is used for the estimation of gold leach extraction across all deposits and lithologies tested is:

Au Recovery (%) = 0.5145 x [Au head, g/t] + 91.533

Using this relationship, for a 2 g/t Au head grade, this implies an average extraction of 92.6%.

Figure 13.17 compares the modelled gold extraction with the measured gold extraction. The standard error of 1.98 defines the accuracy of the model with 2 times the standard error relating to a 95% confidence Interval. This represents an improvement on simply using the average of all recovery data or a model based on the prediction of the leach residual which resulted in standard errors of 2.50 and 2.28 respectively.

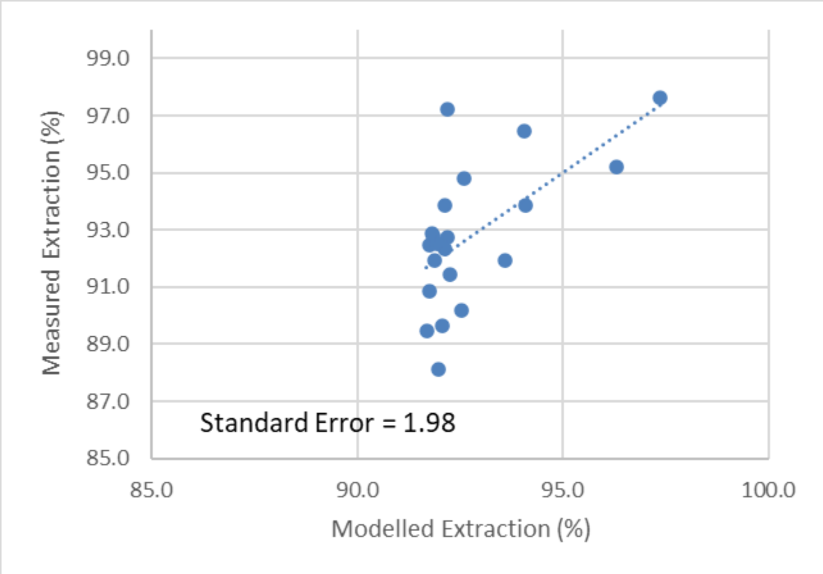

Figure 13.17: Modelled vs Measured Au Extraction

13.7 Materials Handling Testwork

Inspection of the ore samples as received suggested that saprolite was likely to be the most difficult component of the ore to physically handle because of the “sticky clay like” appearance. This was supported anecdotal evidence from operations where it was noted that a single truck load of “sticky” ore can block transfer points, regardless of the blend target for the period.

The materials handling program was designed to assess the materials handling risk of saprolite and determine whether laterite must be processed via the mineral sizer, or whether it was suitable to be crushed and stockpiled.

Three sample of different lithologies were prepared for materials handling testwork to understand any challenges that may arise in the design and operation of bins, chutes, hoppers and transfer points. The work was carried out by Jenike & Johanson, recognised specialists in this area. The targeted lithologies were:

- 100% fresh ore.
- 100% saprolite ore.
- 50% fresh ore and 50% laterite ore, which was chosen to define the impact of laterite ore on fresh ore handling characteristics and assess whether laterite should be processed via the jaw crusher and stockpile, or the mineral sizer.

The fresh and laterite samples were crushed to 100% passing 6.3mm and the saprolite sample, which is naturally fine, was tested as received without further crushing.

The samples were tested for material flow properties by Jenike & Johanson (Jenike & Johanson, 2025) and the interpreted summary of the results are:

- Saprolite is significantly more difficult to handle in chutes and reclaiming with feeders compared to the fresh and fresh/laterite blend sample.

- Saprolite at 25% moisture content is very cohesive, and effectively untestable and it is recommended that the design of crushing and ore handling component of the flowsheet be designed with minimal transfer points and steep angles to avoid blockage.

A jaw crusher to a stockpile with a reclaim feeder would be suitable for fresh and fresh/laterite blends. However, saprolitic materials handling properties are not suitable for this system and relatively small quantities has the potential to bridge or block transfer points.

With regards to materials handling at the beginning of the process, it is recommended that saprolite be reclaimed and crushed separately to ore with fresh and lateritic handling properties. This component of the circuit requires moisture control, minimised drop energies, chute angles of greater than 74 degrees from horizontal and low friction resistance liner materials (e.g. Matrox) to contend with difficult materials handling properties of saprolite ores.

13.8 Rheology

Rheology of a fresh ore sample (Composite A) was measured by during Program 2 (IMO, 2024) for a range of slurry densities from 50% to 60% w/w solids. The sample tested exhibited shear thinning characteristics, as shown in Figure 13.18.

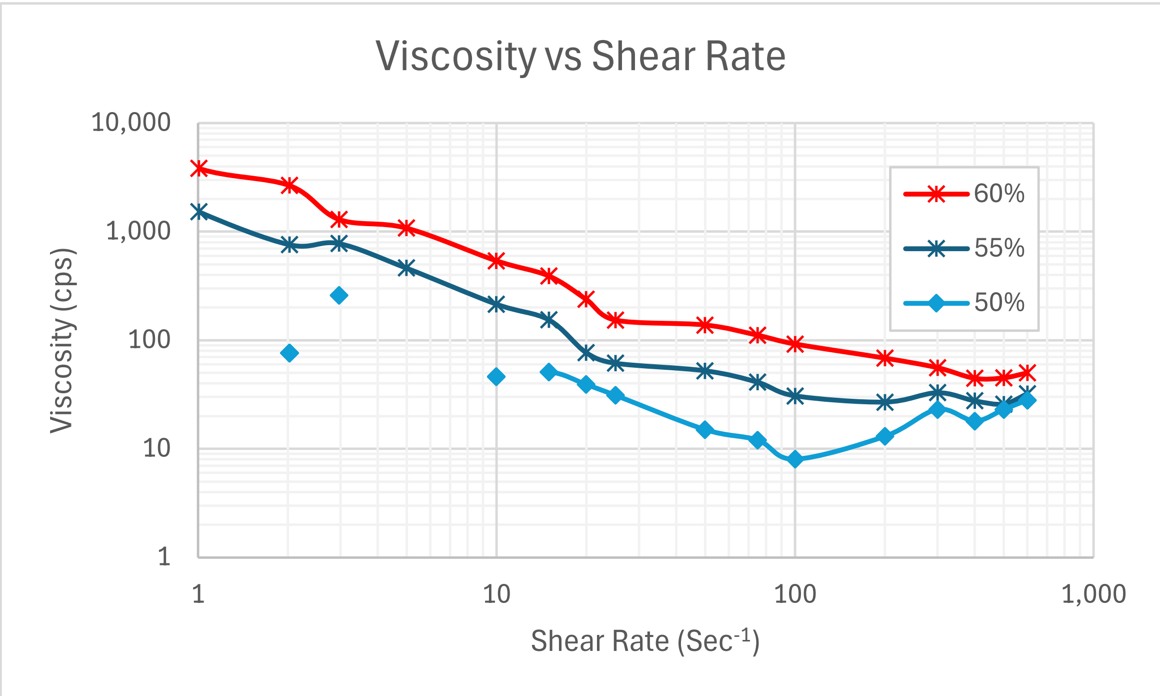

Figure 13.18: Viscosity versus Shear Rate for Fresh Ore

Subsequent testwork carried out in Program 3 (ALS Metallurgy, 2025) focused on impact of increasing proportions of saprolite (sample number M747 PKI23 MT006 44-66) with a fresh ore (tonalite/mafic Composite) as the saprolite was considered to be a driving factor for changing the rheology of the slurry.

The introduction of saprolite at only 25% in the blend appears to change the viscosity characteristics to shear thickening (Figure 13.19) where the viscosity increases as the shear rate (sec^{-1}) is increased, however the viscosity burden with this proportion of saprolite is not significantly worse than for fresh ore, therefore operating at similar densities within the leaching circuit should still be achievable.

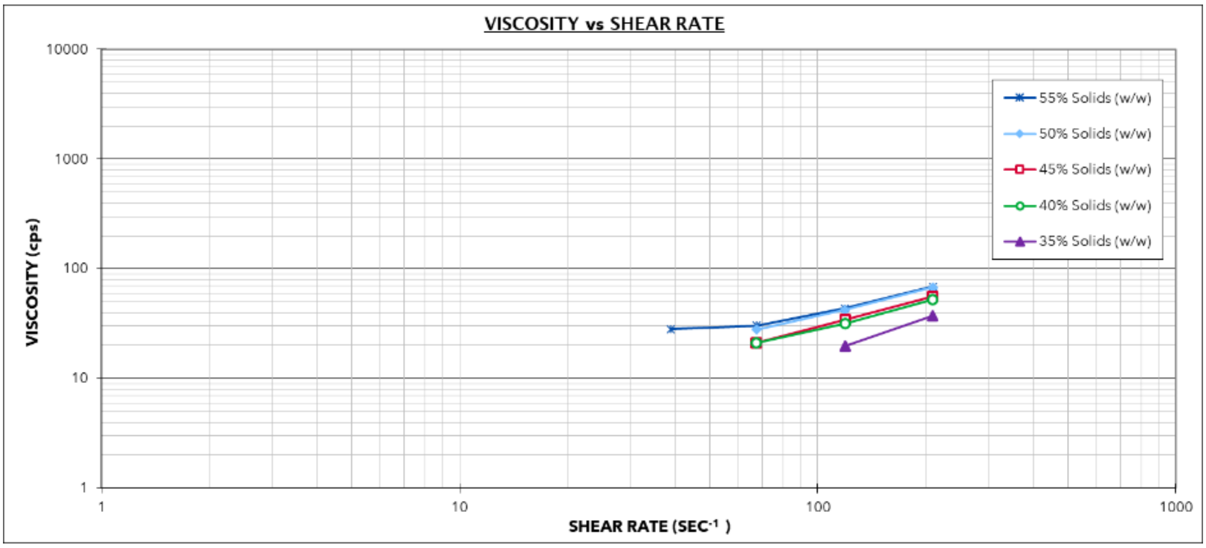

Figure 13.19: Viscosity versus Shear Rate for 25% Saprolite / 75% Fresh Ore

Figure 13.20, Figure 13.21 and Figure 13.22 present the viscosity curves as the proportion of saprolite in the blend increases to 50%, 75% and 100% respectively.

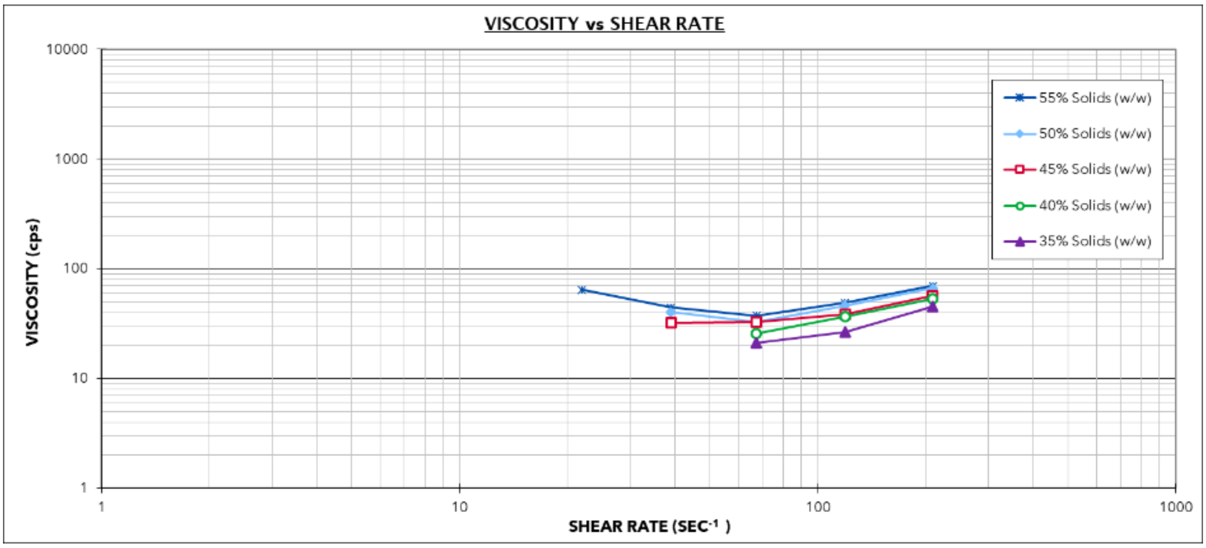

Figure 13.20: Viscosity versus Shear Rate for 50% Saprolite / 50% Fresh Ore

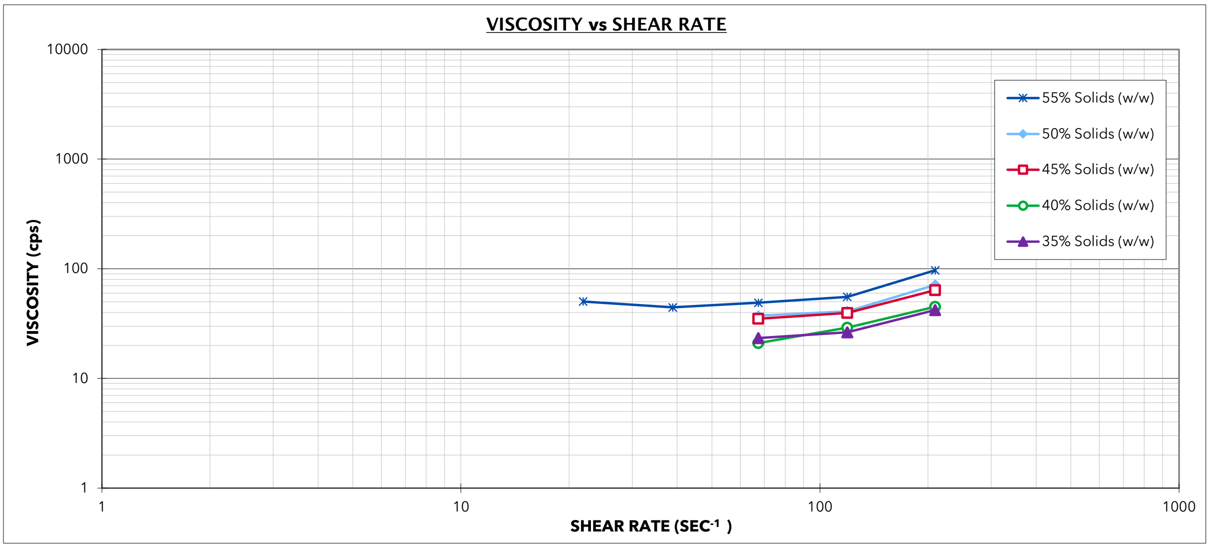

Figure 13.21: Viscosity versus Shear Rate for 75% Saprolite / 25% Fresh Ore

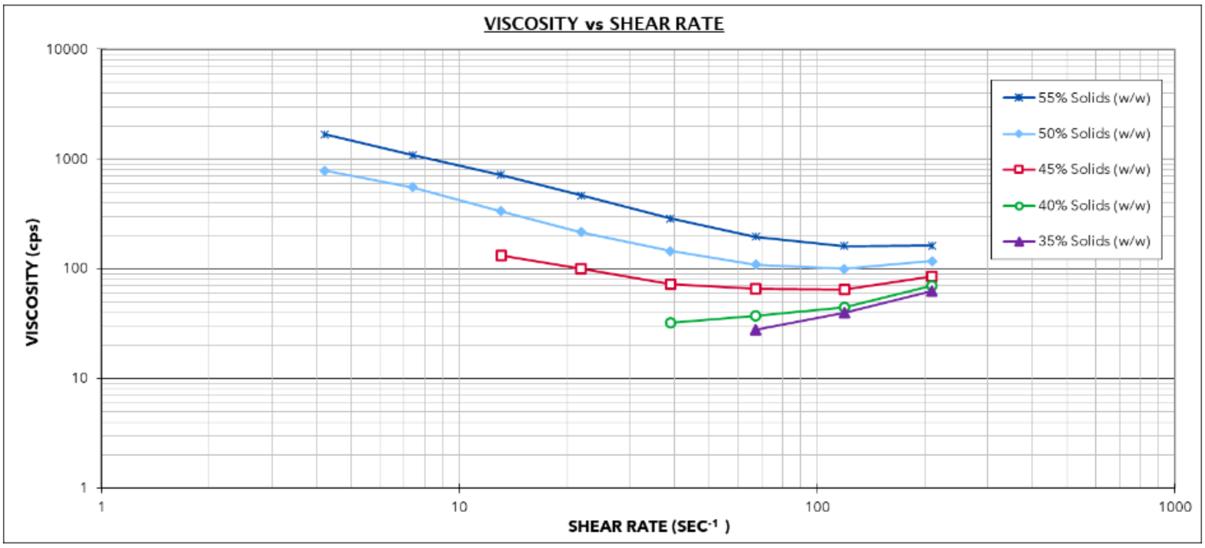

Figure 13.22: Viscosity versus Shear Rate 100% Saprolite

Notably, it is only the 100% saprolite sample that demonstrates a significant increase in viscosity (above 100 cps) that would pose difficulties with pumping, agitation and launder flow in the leach tanks. As a result, it can be concluded that blends with less than 75% saprolite feasibly operate at densities of approximately 50% solids in the leach without specific design to accommodate slurry rheology, provided it can be thickened to this density.

13.9 Thickening Testwork

The same saprolite and fresh (tonalite/mafic) ore blends used for rheology testwork (Section 13.8) were also used for dynamic thickening testwork that was conducted by Metso. The full report is available (Metso, 2024), and the summarised results, all carried out at thickener loadings ranging from 0.25 to 1.5 $t/(m^2/h)$, are:

- 100% saprolite blend can be thickened to underflow densities of 41-52% solids (w/w), with associated yield stress of 27-84 Pa.
- 75% saprolite / 25 % fresh blend can be thickened to underflow densities of 51-56% solids (w/w), with associated yield stress of 44-80 Pa.
- 50% saprolite / 50 % fresh blend can be thickened to underflow densities of 52-61% solids (w/w), with associated yield stress of 29-83 Pa.
- 25% saprolite / 75 % fresh blend can be thickened to underflow densities of 53-64% solids (w/w), with associated yield stress of 10-87 Pa.

As such, if the quantity of saprolite in the blend is limited to 75% (w/w), then it can be considered practical to target underflow densities of at least 51% solids in design and operation.

To achieve these results, 20-30 g/t of Magnaflocc 155 was used to achieve excellent overflow clarities without the need for additional coagulant and Metso recommended a design solids loading of 1t1 t/m^2h.

13.10 Filtration Testwork

The same saprolite and fresh (tonalite/mafic) ore blends used for rheology testwork (Section 13.8) and thickening testwork (Section 13.9) were also used for belt and pressure filtration testwork that was conducted by Metso. The full report is available (Metso, 2024), and the summarised results are:

- Pressure filtration technology of the 100% saprolite blend demonstrated dewatering with filtration capacity of 249 kg DS/m^2h to achieve a cake moisture content of 21.9% (w/w) with a cake thickness of 40 mm and filtrate clarity of 1,500 mg/l.
- Pressure filtration technology of the 75% saprolite / 25% fresh blend demonstrated dewatering with filtration capacity of 272 kg DS/m^2h to achieve a cake moisture content of 20.5% (w/w) with a cake thickness of 40 mm and filtrate clarity of 500 mg/l.
- Pressure filtration technology of the 50% saprolite / 50% fresh blend demonstrated dewatering with filtration capacity of 175 kg DS/m^2h to achieve a cake moisture content of 15.9% (w/w) with a cake thickness of 37 mm and filtrate clarity of 1,200 mg/l.
- Pressure filtration technology of the 25% saprolite / 75% fresh blend demonstrated dewatering with filtration capacity of 266 kg DS/m^2h to achieve a cake moisture content of 15.2% (w/w) with a cake thickness of 47 mm and filtrate clarity of 1,600 mg/l.

Noting that the filtration capacity (DS/m^2h) refers to the instantaneous rates of filtration and do not account for the overall cycle time that a specific industrial filter design will have to account for.

Generally, as the proportion of saprolite in the blend decreased, the filtration performance improved and at 25% saprolite in the blend, larger filtration chambers are practical, and the achievable moisture approaches 15% (w/w).

It is notable that even the 100% saprolite sample produced a competent cake, although it did appear damp and sticky, despite being easily discharged from the cloth, as per Figure 13.23.

(a) (b)

Figure 13.23: 100% Saprolite Sample in (a) the Chamber and (b) the Top View of the Cake

Figure 13.24 presents a filter cake from the 50% saprolite / 50% fresh sample which was stable, but dry and crumbly as it easily fell apart when disturbed. As the proportion of saprolite in the sample reduced, the filter product presented comparable physical characteristics.

Figure 13.24: 50% Saprolite Sample from 40mm Chamber @15.9% Moisture

A key trend observed was the decrease in filtration capacity to achieve a comparable moisture, as the proportion of saprolite increased in the blend. This is demonstrated in Table 13.24 which compares the product moisture versus filtration capacity for all samples with the 50mm chamber.

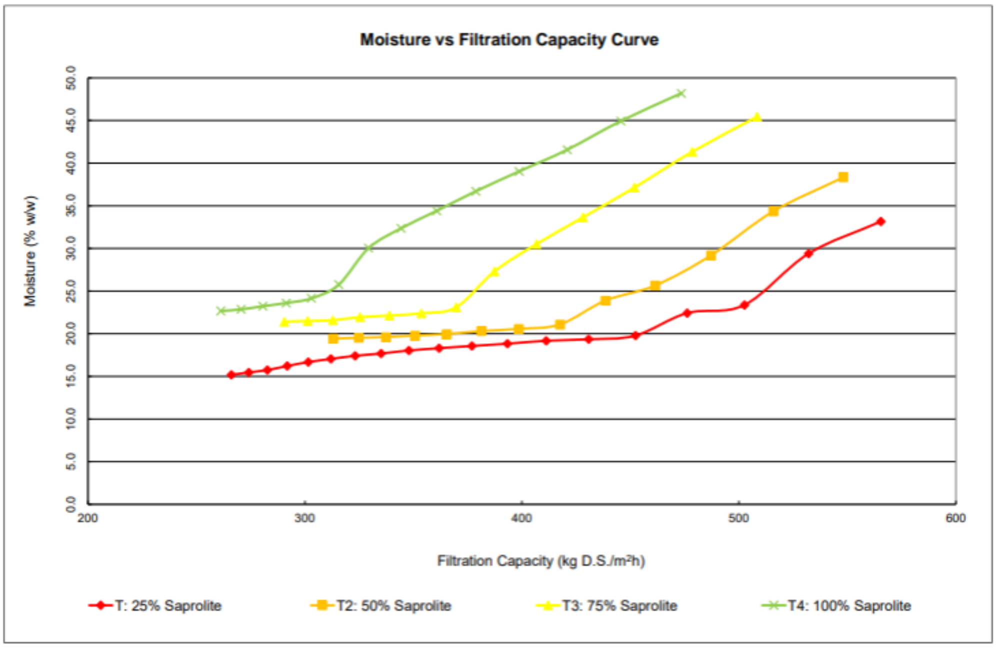

Figure 13.25: Moisture versus Filtration Capacity – 50mm Chamber

Operationally, it is likely that the proportion of saprolite in the blend will influence the capacity of pressure filtration achieving an acceptable product cake moisture at the design tonnage.

For the purpose of equipment sizing, vendors were instructed to design for a maximum of 50% saprolite in the filtration feed slurry and a minimum density of 42% solids.

Belt (vacuum) filtration tests, regardless of ore blend produced cakes that were wet, sticky and not practical for transportation or dry stacking tailings and therefore were not further considered/

13.11 Bulk Leach Tests

Three bulk leach tests were completed in Program 2 and Program 3 to prepare bulk samples for carbon loading and detoxification testwork. The makeup of the composites is detailed in the respective testwork reports (IMO, 2024) (ALS Metallurgy, 2025), however they can generally be described by their lithologies as:

- Program 2 Test BLT01, fresh lithology composite.
- Program 3 Test BK21126, tonalite/mafic composite.
- Program 3 Test BK21127, saprolite/saprock composite.

For all tests the common test conditions included:

- Grind size (P_{80}) of 75 µm.
- pH 10, maintained with lime.
- Perth tap water.
- Dissolved oxygen at 8-10 ppm, maintained with air.

The test in Program 2 was conducted for 48 hours and included a gravity recovery prior to leaching, which reduced the gold in solution for this test. Additionally, while it was intended to maintain the sodium cyanide in solution in this test at concentrations greater than 300 ppm, by the end of the test the sodium cyanide in solution had reduced to 120 ppm, and it was this slurry that was used for the Program 2 carbon loading testwork (Section 13.12).

For the two bulk leaches conducted as part of Program 3, the leach conditions were tailored to present an ideal residual slurry condition for the subsequent carbon loading tests. i.e. no gravity recovery was applied to increase the gold in solution and excess cyanide was intentionally maintained (targeting greater 500 ppm sodium cyanide) to inhibit the loading of any copper in solution.

13.12 Carbon Loading Testwork

Initial carbon loading testwork (triple contact, sequential) was completed as part of Program 2 based on the bulk leach test described above (IMO, 2024), however the results achieved, , as presented in Table 13.19, were well below expectations with only 426 ppm gold loaded on the carbon.

Table 13.19: Program 2 Triple Carbon Contact Testwork Results

Parameter	Units	BLT01
Au in solution	mg/L	1.12
Cu in solution	mg/L	21.7
CN concentration	mg/L	120
Assayed Carbon Loading	ppm	426
Carbon Weight	g	2.54
Empirical Rate Constant, k	hr^{-1}	124
Exponent for Time, n	unitless	0.42

Upon review, it was hypothesized that two factors limited the gold loading for this test:

- Low cyanide tenors in the solution phase when carbon contact was initiated with 120 ppm cyanide in final leach liquor.
- Significant copper in solution with 21.7 mg/L Cu, which is 20 times greater than gold in solution.

To address this, the additional tests conducted as part of Program 3 (ALS Metallurgy, 2025) were designed to present conditions that limit the competitive loading of soluble copper and would therefore be favourable for gold loading, as discussed in Section 13.11. Specifically, the cyanide concentration was elevated, and gravity recovery of gold was not applied prior to leaching to ensure that gold in solution were as high as practical.

The results of the triple carbon contact tests conducted in Program 3 are summarised in Table 13.20.

Table 13.20: Program 3 Triple Carbon Contact Test Results

Parameter	Units	Mafic/Tonalite Composite	Saprolite/Saprock Composite
Au in solution	mg/L	1.62	1.05
Cu in solution	mg/L	40.1	66.7
CN concentration	mg/L	680	850
Assayed Carbon Loading	ppm	**1,677**	**1,530**
Carbon Weight	g	7.58	5.49
Empirical Rate Constant, k	hr^{-1}	230	149
Exponent for Time, n	unitless	0.62	0.78

Table 13.20 demonstrates that the presence of copper in solution does not inhibit the loading of gold onto carbon, provided the cyanide concentration is maintained at levels that minimise the loading of gold. Note, that these are not the optimised cyanide concentrations for gold loading as the cyanide is likely to be well in excess which was deemed appropriate to demonstrate that soluble copper could be managed.

It is also notable that the copper levels tested were significantly greater than median of the variability dataset (27 mg/L Cu, Section 13.5.6).

Sub-samples of the same tailings slurry were also subjected to equilibrium carbon loading tests where the concentration of carbon in slurry was varied. The resultant equilibrium loading curves for the mafic/tonalite and saprolite/saprock samples are presented in Figure 13.26.

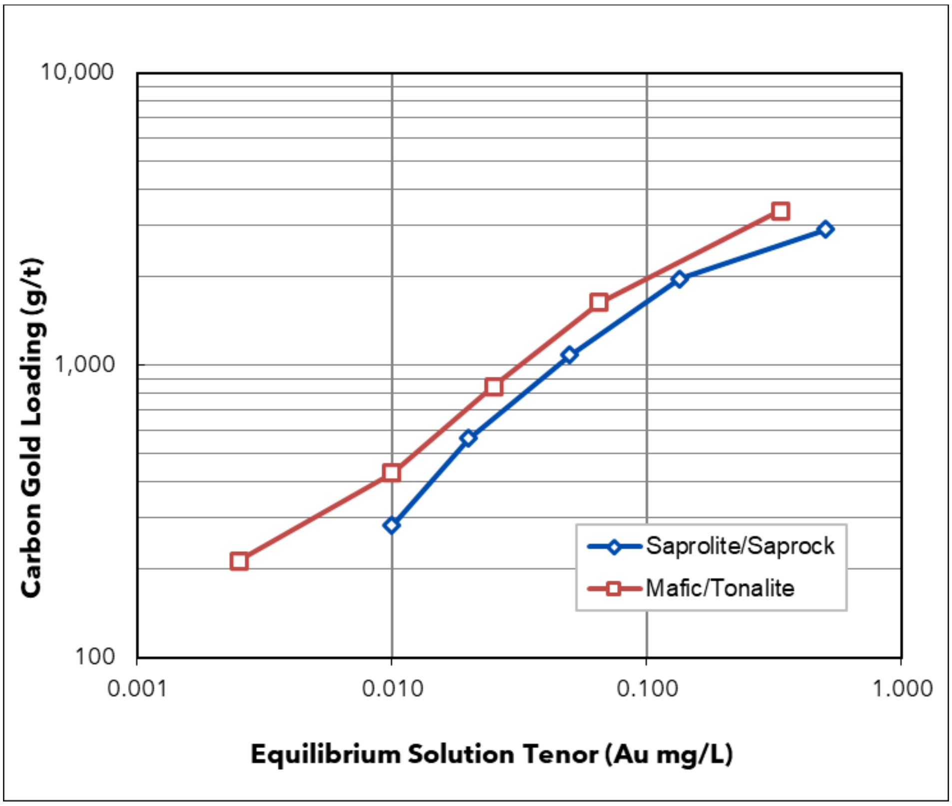

Figure 13.26: Equilibrium Gold Loading Curves

Figure 13.26 shows that the slope of the equilibrium loading curves are comparable for each sample, and only a small variation in the intercept for the data. Therefore, it can be concluded that there is negligible difference the equilibrium loading performance of the two samples, despite the different lithologies.

A line of best fit of the combined dataset can be used to define the carbon loading capacity using the following equation:

$$\log (X/M) = m.\log (C) + \log (K)$$

Where:

- X/M is mg of gold adsorbed per gram of carbon at equilibrium
- C is mgpl of gold remaining in solution
- m is constant (slope), determined to be 0.582.
- K is constant (intercept), with log (K) determined to be 3.729.

The equilibrium loading curves provide confidence that the carbon inventories in the plant can be managed to achieve the loading of gold required on carbon in operations.

13.13 Cyanide Destruction Testwork

Initial cyanide destruction testwork conducted and reported as part of Program 2 (IMO, 2024) using a sodium metabisulphite (SMBS) addition at a stoichiometric ratio of 150%. The results of the test were conflicting and did not meet expectations in that:

- Final free cyanide titration completed by IMO reported 0 ppm free cyanide (picric acid titration) after detoxification.
- Full cyanide speciation of the detoxification product reported by an external laboratory, showed CNO concentrations consistent with full detoxification, but also free and weak acid dissociable (WAD) cyanide which is not consistent with full detoxification.

Four additional detoxification tests were therefore completed as part of Program 3 (ALS Metallurgy, 2025) on the residual slurries from the bulk leach tests (Section 13.11) that were also used for carbon loading tests.

Initial tests targeted 5 g SO_2/g WAD CN which resulted in almost complete destruction of WAD cyanide, so the SMBS addition was reduced to optimise reagent consumption and still achieve the target product WAD cyanide concentration of below 50 ppm. The optimised tests targeted the following conditions:

- 3 g SO_2/g WAD CN, added as $Na_2S_2O_5$.
- 30 mg/L Cu excess, added as $CuSO_4.5H_2O$.
- pH 8.5.
- 2-hour residence time.

The cyanide speciation of the feed and product liquors from the optimised cyanide destruction test are presented in Table 13.21.

Table 13.21: Cyanide Speciation for Detox Feed and Products (Optimised)

Species	Unit	Fresh Ore		Weathered Ore	
		Feed	Product	Feed	Product
CN Free	mg/L	**250**	<5	**260**	<5
WAD CN	mg/L	340	**0.03**	410	**3.5**
CN total	mg/L	650	270	500	58
CNO	mg/L	8.9	510	14	640
SCN	mg/L	170	150	170	150
Ag CN	mg/L	<10	<10	<10	<10
Au CN	mg/L	<0.5	<0.5	<0.5	<0.5
Co CN	mg/L	<0.1	<0.1	<0.1	<0.1
Cr CN	mg/L	<0.5	<0.5	<0.5	<0.5
Cu CN	mg/L	**85**	1.5	**130**	2.9
Fe^{2+} CN	mg/L	110	75	22	16
Fe^{3+} CN	mg/L	<1.0	<1.0	<1.0	<1.0
Ni CN	mg/L	1.7	<0.2	3.2	<0.2
Free + Cu CN	mg/L	335		390	

Table 13.21 shows that optimised conditions were successful in exceeding the cyanide destruction target product 50 mg/L WAD cyanide and effectively reduced WAD cyanide by greater than 99%.

Using the definitions of WAD cyanide presented in Figure 13.27, the combination of free and copper cyanide complex comprises of greater than 95% of WAD cyanide. This is relevant because the operational SMBS demand is driven by the concentration of WAD cyanide in solution at the time, and the variability program and reagent consumption sections (Section 13.5.6 and Section 13.6.2) discussed the implications of copper to cyanide consumption and concluded that a median copper in solution of 27 mg/L is an appropriate mid-point of the available data set to use as the operating point.

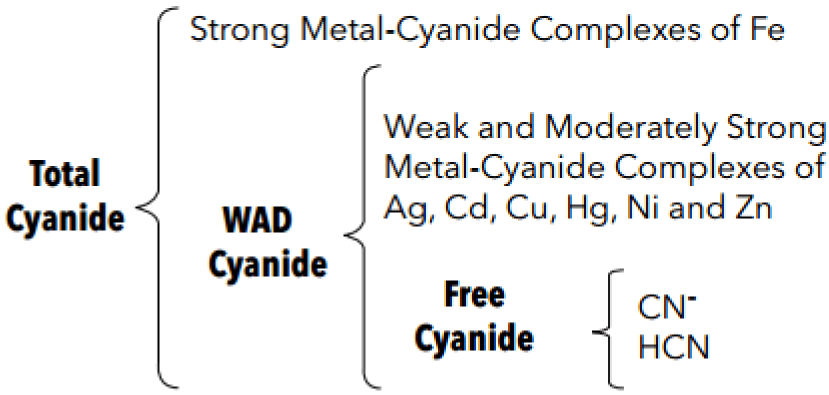

Figure 13.27: Cyanide Speciation Definitions

As a result, SMBS consumption can be estimated by:

- Defining the quantity of WAD CN to be destroyed. i.e. WAD cyanide less target WAD cyanide.
- Applying 3 g SO_2/g WAD CN, or 4.45g SMBS/g WAD CN.

Similarly, additional copper sulphate is only required if an excess of copper in solution does not exist. To estimate the amount of copper required, the composition of the residual leach solution must be considered to account for:

- Presence of iron, which complexes with copper in the presence of cyanide, thereby making the copper unavailable as a catalyst for the cyanide destruction reaction.
- Presence of copper, nickel and zinc in the leach solution, which all behave similarly to the Cu^{2+} ions added to be the catalyst for the cyanide destruction reaction.

Available data on leach liquors suggests that the variability in copper and iron in solution is likely to be the most significant drivers in predicting the addition rates for sodium metabisulphite and copper sulphate. The recommended composition for the leach residual liquor is:

- Slurry density of 50% w/w solids.
- Residual NaCN of 200 mg/L, which would allow for periodic excursion in copper grades that require the residual NaCN in leach liquor to be elevated to ensure effective gold loading onto carbon (Section 13.12).
- Copper in solution of 27 mg/L, median of variability dataset (Section 13.5.6).

- Iron in solution of 11 mg/L, mean of the variability dataset (Section 13.5.6).
- Nickel in solution of 2.6 mg/L, average of solution assays cyanide destruction testwork liquors.
- Zinc in solution of 0.48 mg/L, average of variability dataset (Section 13.5.6).

Although the operational target for WAD cyanide after cyanide destruction is less than 50 mg/L, it is recommended that estimates of sodium metabisulphite addition be ratioed to achieve 20 mg/L WAD cyanide to account for scale-up inefficiencies.

13.14 Paste Testwork

A bulk composite sample was ground to 80% passing 75 µm and provided to MineFill Pty Ltd (MineFill) for paste testwork. The composition of the sample is presented in Table 13.22.

Table 13.22: Composite Sample for Paste Testwork

Sample ID	Drill Hole ID	Interval (m)	Domain / Orebody	Lithology	Mass in Composite (kg)
Comp #02	BNEDD0232	168.4 - 190.7	NEB	Mafic	15
Comp #08	BNEDD0229	128.1 - 143	NEB	Tonalite/Int	10
Comp #11	BNEDD0147	74.1 - 87.6	NEB	Saprolite	15
Comp #13	BNEDD0147	98.32 - 148	NEB	Tonalite	30
Comp #15	BNEDD0162	73.33 - 79	NEB	Shear	6
Comp #21	BNEDD0180	409 - 412.65	NEB	Shear	6
Comp #23	BNEDD0157	62.69 - 157.8	NEB	Tonalite	30

The full description of testwork and conclusions is attached as an appendix (Minefill Services, 2025) and a summary of the testwork and conclusions are presented below.

Because the operation is a combination of open pit and underground, an excess tailing will be generated, over and above the underground paste requirements. As a result, it was decided to deslime the tailings sample which is considered best practise for achieving a suitable paste rheology and superior geotechnical strength. Desliming was completed using a settlement and decant method that reduced the fines as per Table 13.23.

Table 13.23: Particle Size Distribution of Full Stream and Deslimed Tailings

Particle Size (µm)	Full Stream (% Passing)	Deslimed (% Passing)
75	85%	70%
53	76%	51%
38	68%	35%
15	49%	23%
9	41%	19%

The binder used in the testwork was sourced from suppliers based in Tanzania as samples of local Guinean cement were not available, however subsequent review of specifications of locally available cement demonstrated similar properties.

Good quality paste was produced from the partially deslimed tailings with solids content of 69-72% required to achieve a flowable rheology, which is considered within a typical industry standard.

Strength testwork with up to 12% binder content identified that strengths with general purpose 42.5 (GP) cement delivered relatively poor unconfined compressive strengths (764 kPa after 28 days hydration). However, 6% binder addition with a slag-based low heat cement (LH) delivered significantly improved strengths (2,274 kPa after 28ays hydration).

This is a typical result which highlights why slag-based binders are primarily used in global backfill applications. It is highlighted that in some areas of the orebody where paste is undercut by stopes from the panel below, paste strengths exceeding 2,000 kPa are required to prevent excessive dilution.

More specific conclusions from the testwork are:

- Deslimed paste was cohesive and homogenous.
- Paste rheology is not sensitive to paste solids content which is important to generate a robust operating paste system.
- Relatively poor strengths were achieved for the GP cement paste mixes. Significantly higher paste strengths were developed for the low heat (slag based) binder mixes.
- Suitable paste strength can still be expected from GP cement, albeit at significantly higher ratios than have been tested.

The testwork successfully demonstrated that a suitable paste can be made from partially deslimed tailings as shown in Table 13.24. Binder quantities can be varied to match the variable underground paste fill strength requirements.

Table 13.24: Paste UCS Test Matrix and Results after 28 Days

Binder Type	Binder Content (%)	In Situ Binder (kg/m^3)	Solids Content (%)	28 Day UCS (kPa)	Moisture Content (%)	Dry Density (t/m^3)
GP Cement	5.0	73	71.5	200	33.8	1.46
GP Cement	8.0	116	71.5	371	33.1	1.45
GP Cement	12.0	175	78.5	798	32.5	1.45
GP Cement	5.0	78.5	78.5	164	36.8	1.39
Low Heat	2.0	35.6	80.0	693	33.1	1.45
Low Heat	3.0	52.8	80.0	1,274	32.8	1.45
Low Heat	6.0	105.2	80.0	2,274	33.4	1.45

The testwork results were then used to develop a design (Minefill Services, 2025) that integrates the paste fill requirements with the proposed mining method.

A key outcome of this study is the estimated binder quantities required to achieve the geotechnical strength required for various paste fill duties in the underground mining operation, as per Table 13.25. The estimates of binder quantities have assumed the use of GP cement as that is currently the only binder quality that has been identified for the project. However, there exists an opportunity to reduce binder consumption significantly if a local or imported low heat cement can be sourced.

Table 13.25: Paste Fill Binder Contents GP Cement

Stoping Region	Average Strength Target (kPa)	Binder (Dry Weight %)	Binder (kg/m3)
Secondary Stopes Not Exposed	150	5.0%	73
Vertical Exposure Upper 10 m	180	5.0%	73
Vertical Exposure Lower 10 m	255	6.0%	88
Horizontal Exposure	2,625	20%[1]	292

Notes:

1. Value is estimated as no testwork is completed at such binder contents

13.15 Summary of Metallurgical Interpretation for Design

Key design and operating parameters that have been finalised from all testwork to date for use in the DFS include:

- Materials handling testwork (Section 13.7) interpretation concluded that the materials handling properties of saprolitic ores are not amenable to jaw crushing or stockpile reclamation. As a result, a separate feed system is recommended for this ore type. It is recommended that saprolitic ore be crushed using a mineral sizer, then direct fed to the SAG Mill.
- Fresh ore and fresh/laterite blends are both suitable for jaw crushing and stockpile reclamation.
- Grind size (P_{80}) of 75 µm (Section 13.5.1).
- Power demand for the comminution circuit will vary by lithology as per below:
  - Fresh ore, 29.1 kWh/t.
  - Laterite/Saprock/Shear, 13.1 kWh/t.
  - Saprolite, 5.6 kWh/t.
- The inclusion of gravity recovery within the comminution circuit is warranted and will provide an average recovery of gold of 36% (Section 13.5.3).
- Cyanide concentration to target leach residual of 150 to 200 mg/L of NaCN (Section 13.5.3). In normal operation, 150 mg/L can be targeted without an impact on gold extraction, however in the event of elevated copper in solution, higher NaCN concentrations may be required to enable maximum gold loading onto carbon (Section 13.12).

- Air addition to the leach, with no justification for oxygen addition (Section 13.5.4).
- For blends containing less than 75% saprolite, thickener underflows (Section 13.9) can realistically achieve greater than 52% solids (w/w), and rheology (Section 13.8) supports the operation of the leach at 50% solids (w/w).
- For blends containing less than 50% saprolite, filtration moisture of less than 16% (w/w) is realistically achievable with pressure filtration (Section 13.10).
- Cyanide consumption is influenced by the presence of cyanide soluble copper (Section 13.6.2), and review of the variability dataset justifies 0.40 kg/t of cyanide consumption in the leach.
- Lime consumption is influenced by the lithology of the ore (Section 13.6.2) with average lime consumptions of:
  - Fresh ore, 0.33 kg/t.
  - Weathered ore, 2.06 kg/t.
- Gold extraction is influenced by gold head grade as defined by the following relationship:
  - Au Recovery (%) = 0.5145 x [Au head, g/t] + 91.533
- Cyanide destruction is effective (Section 13.13) using the following operating conditions:
  - 3 g SO_2/g WAD CN, added as $Na_2S_2O_5$.
  - 30 mg/L Cu excess, added as $CuSO_4.5H_2O$.
  - pH 8.5.
  - 2-hour residence time.
- The estimation of WAD cyanide is majority driven by free cyanide, and cyanide associated with soluble copper (Section 13.13), where free cyanide is expected to range between 106 and 80 g/L. Soluble copper can only be estimated by taking the median of the variability leach data set (Section 13.5.6). The median copper in solution is 27 mg/L.

13.16 Conclusion

Based on the samples selected, testwork carried out, analysis of the resulting data, the Qualified Person considers the conclusions drawn relating to mineral processing and metallurgical testing to be reasonable and the metallurgical performance predictions to be suitable for the determination of Mineral Reserves and development of production targets relating to the deposits tested. In addition, the design information gathered is considered to be suitable to develop the design of the processing facilities to a level of detail to support the accuracy of the capital and operating cost estimates outlined.

14 MINERAL RESOUCE ESTIMATE

14.1 Lithological Modelling

A conceptual geological model for NEB comprises a set of anastomosing shear structures in a greenstone sequence, focused at the hanging wall contact of a felsic intrusion into an older mafic/metasedimentary sequence.

Leapfrog Geo software was used to create models of these lithologies to fill the resource model space. The weathering surfaces (laterite, mottled zone, saprolite, and saprock) were modelled as DTM surfaces and other lithologies were modelled as solid wireframes. The mafic was not modelled as it is treated as a default lithology above the main shear.

14.2 Domain Modelling

Downhole composites were extracted from the database at lengths of 3 m and 1 m, using a minimum length of 50% of the composite length, with saprolite and fresh mineralisation using 3 m composites, and the laterite mineralisation using 1 m composites. The composite files were trimmed against the interpreted base of laterite DTM surfaces.

Leapfrog grade shells were extracted from the composite files to use as domains for the resource estimates. Smoothing parameters were chosen in an iterative process, after reviewing preliminary shells to create an appropriate continuity.

For NEB, three nested grade domains were defined in the saprolite and fresh mineralisation using Leapfrog software, at nominal 2 g/t Au (high-grade), 0.4 g/t Au (medium-grade), 0.3 g/t (GBE) and 0.2 g/t Au (low-grade) cut-offs from 3 m downhole composites. For the laterite mineralisation, a 0.5 g/t Au cut-off domain was defined from 1 m downhole composites.

The high-grade domain comprised a large zone along the main shear, as well as several sub-parallel smaller zones, mainly based on single intersections. These lesser confidence shapes were reincorporated into the medium-grade domain. The medium-grade and low-grade domains are largely in the footwall of the high-grade domain. The ultimate footwall of the low-grade domain is poorly controlled due to lack of data. Both were also manually post-processed to remove anomalies and shapes based on isolated intersections. As a final post-processing step, the domains were intersected with the base of laterite DTM as an upper constraint. Plans showing the nested domains and the interpreted shears are presented in Figure 14.1 to Figure 14.4.

For BC, three nested grade domains were defined in the saprolite and fresh mineralisation using Leapfrog software, at nominal 1 g/t Au (high-grade), 0.5 g/t Au (medium-grade) and 0.3 g/t Au (low-grade) cut-offs from 3 m downhole composites (Figure 14.5). An interpreted tonalite contact was used as an anisotropy, and the domains were trimmed against the base of laterite DTM For the laterite mineralisation, a 0.5 g/t Au cut-off domain was defined from 1 m downhole composites.

For Fouwagbe, interpreted mineralised trends were used to produce mineralised shells at a 0.4g/t Au cut-off in Leapfrog software and a SW plunge of 30° was interpreted from the continuity of the mineralisation and the disposition of the artisanal workings on the surface (Figure 14.6). A small laterite mineralised body directly above the shear hosted mineralisation is constrained by the interpreted base of laterite and the current topography.

For Sounsoun, the interpreted E-W shear zone was used as an anisotropy for the Leapfrog shells at a 0.3g/t Au cut-off, with a steep NNE plunge down the shear plane (Figure 14.7).

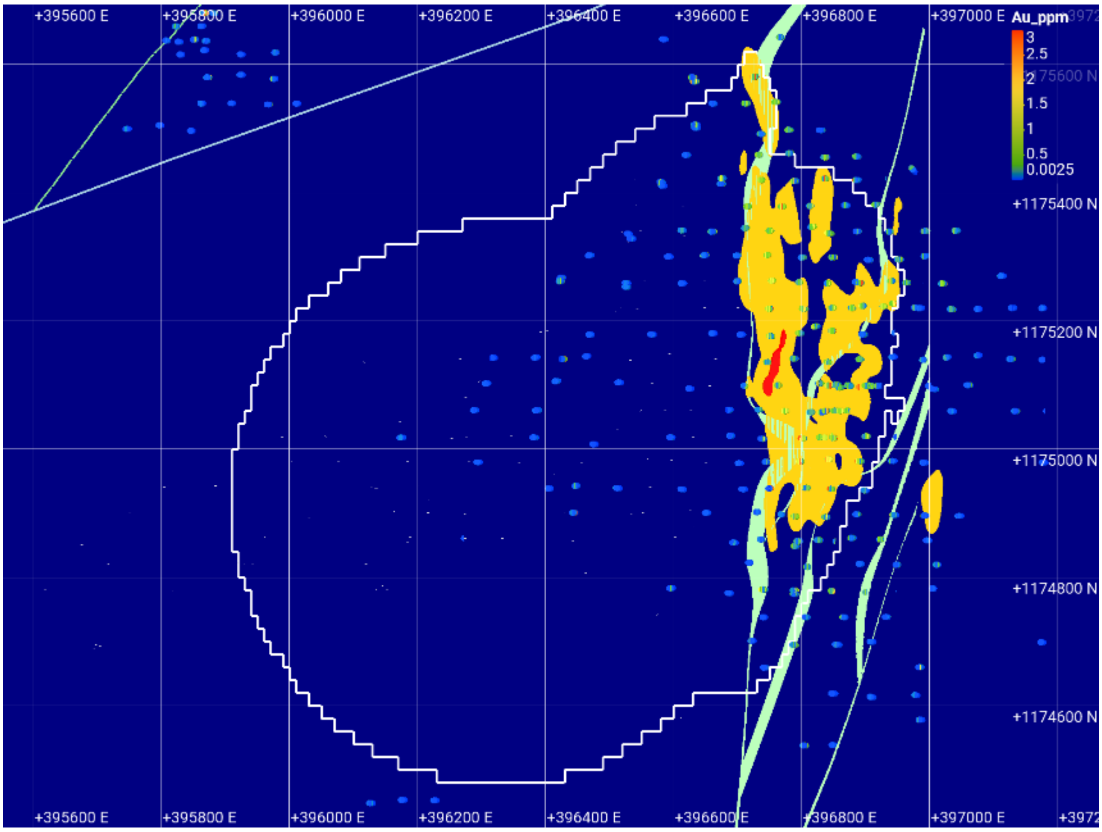

Figure 14.1: 10350mRL NEB Medium-Grade and High-Grade with Shear Zones

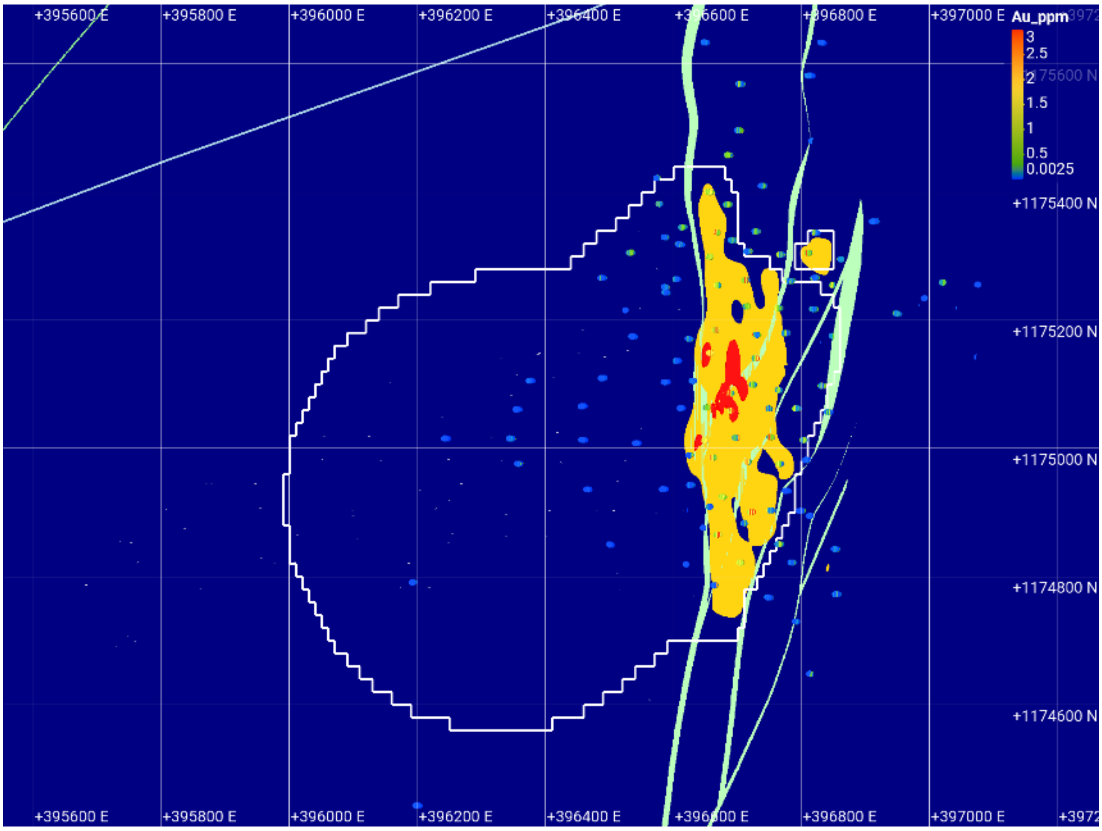

Figure 14.2: 10250mRL NEB Medium-Grade and High-Grade with Shear Zones

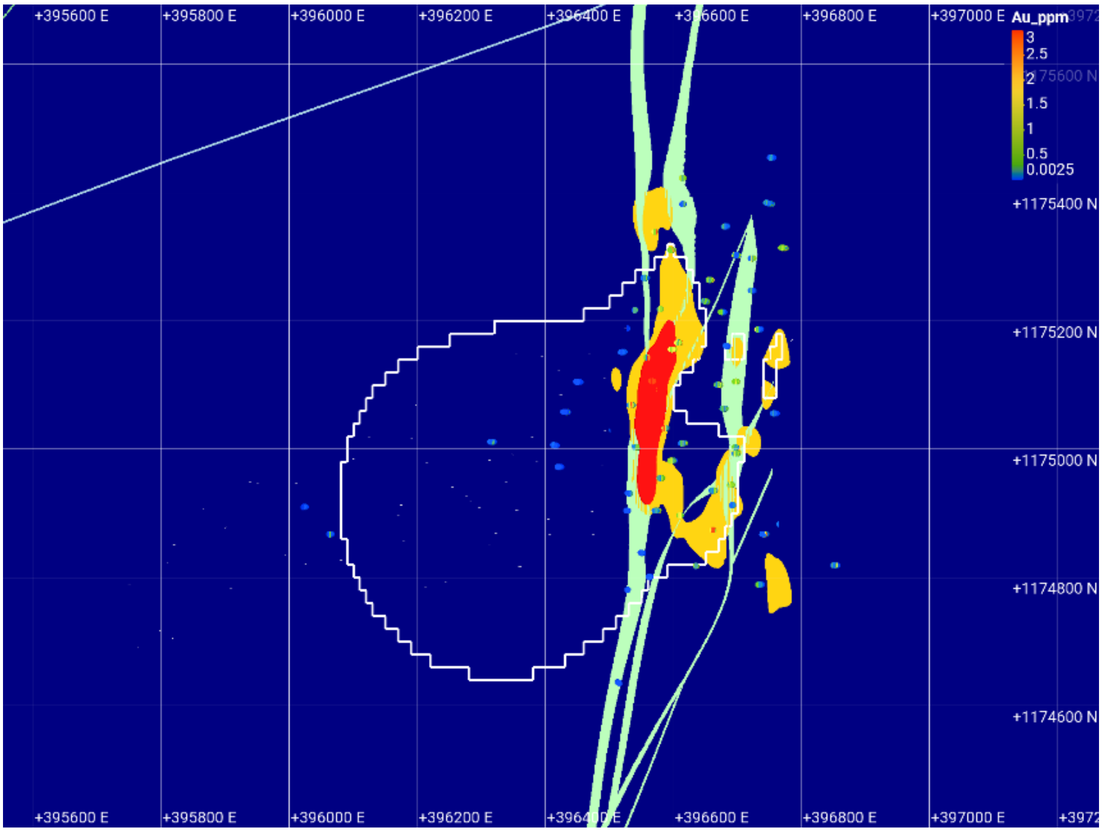

Figure 14.3: 10150mRL NEB Medium-Grade and High-Grade with Shear Zones

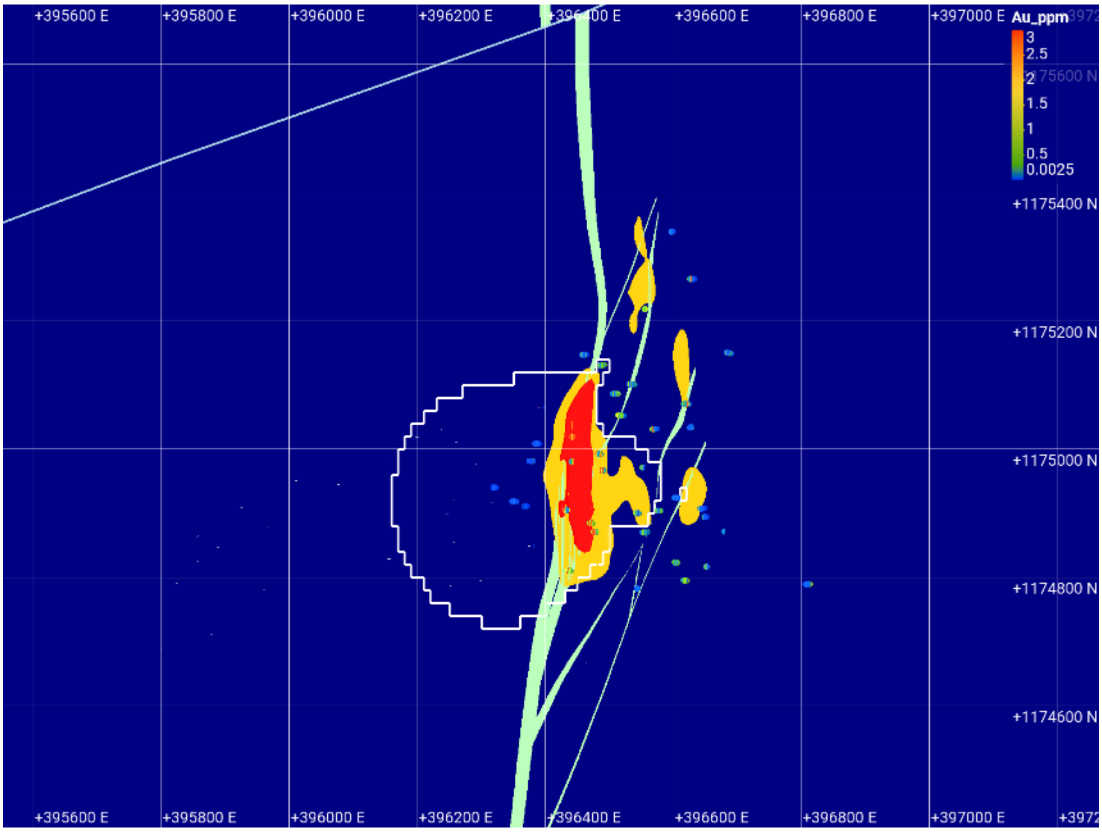

Figure 14.4: 10050mRL NEB Medium-Grade and High-Grade with Shear Zones

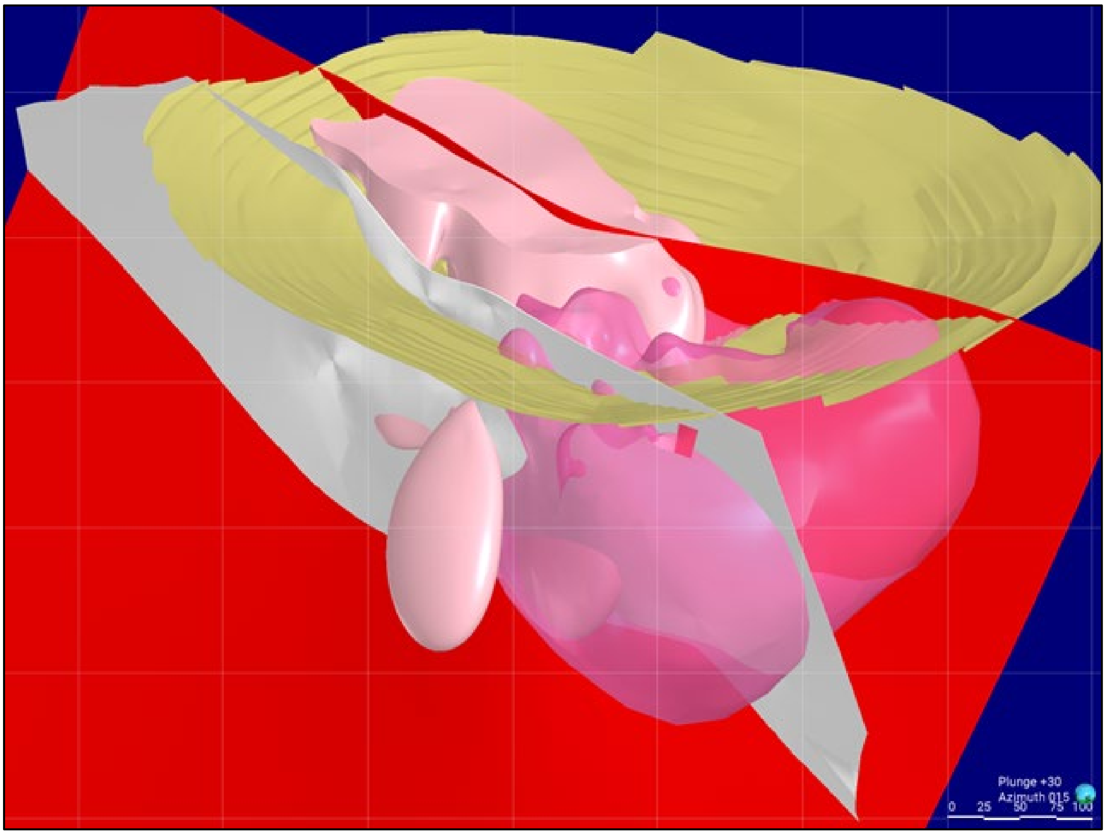

Figure 14.5: BC Lithological and Domain Model with Red, Main Shear; White, Second Order Shear, Purple, Tonalite and Pink, Medium-Grade Domain

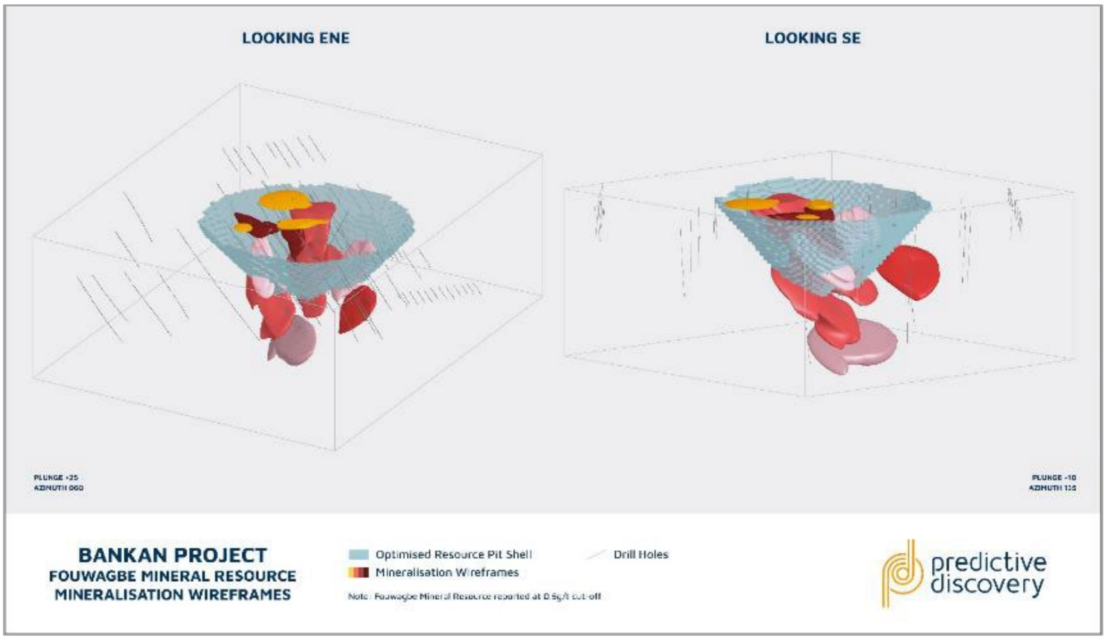

Figure 14.6: Fouwagbe Interpreted Mineralisation Zone (PDI 2025)

Figure 14.7: Sounsoun Interpreted Mineralisation Zones (PDI 2025)

14.3 Mineralised Domain Statistics

To generate the mineralised composite files, the previously generated 3 m or 1 m downhole composite files were intersected with the final domain wireframes in Surpac with composites that had their midpoint within the wireframes were classified as belonging to that wireframe.

High-grade cuts were applied to composites to reduce the influence of extreme outliers. These values, which were determined by statistical analysis including review of coefficient of variation (CV) values, histograms, log-probability plots, and mean-variance plots. The aim of choosing topcuts was to reduce the CV without affecting the overall mean grade of the various mineralised domains. The topcuts adopted, and the reductions in means and CVs are shown in Table 14.1

Composite population statistics for the mineralised domains are shown from Table 14.2 to Table 14.6.

Table 14.1: Topcut Summary

Domain	Top Cut g/t	Mean Reduction	CV Reduction
NEB Low-Grade	N/A	0%	0%
NEB Medium-Grade	30	1.1%	8.3%
NEB High-Grade 1	40	0.6%	2.8%
NEB High-Grade 2	N/A	0%	0%
NEB Laterite	30	9.7%	60.9%
GBE	N/A	0%	0%
BC Low-Grade	7	2.5%	11.3%
BC Medium-Grade	5	0%	1.6%
BC High-Grade	30	9.1%	37.2%

Domain	Top Cut g/t	Mean Reduction	CV Reduction
BC Laterite	15	0%	0%
Fouwagbe	40	13.6%	35.8%
Sounsoun	30	9.9%	21.1%

Table 14.2: NEB Mineralisation Domain, Uncut Au g/t Statistics

Statistic	Low Grade	Medium Grade	High Grade Upper	High Grade Lower	GBE	Laterite
Count	3,019	7,816	662	104	262	1,695
Minimum	0.00	0.00	0.02	0.02	0.02	0.01
Maximum	11.43	67.78	48.67	33.91	11.82	194.88
Mean	0.28	0.99	4.88	3.79	0.87	1.03
Median	0.19	0.59	2.99	2.19	0.62	0.63
Standard Deviation	0.38	1.59	5.52	4.98	1.07	4.98
CV	1.37	1.60	1.13	1.31	1.22	4.83

Table 14.3: NEB Mineralisation Domain, Topcut Au g/t Statistics

Statistic	Low Grade	Medium Grade	High Grade Upper	High Grade Lower	GBE	Laterite
Count	3,019	7,816	662	104	262	1,695
Minimum	0.00	0.00	0.02	0.02	0.02	0.01
Maximum	11.43	30.0	40	33.91	11.82	30
Mean	0.28	0.98	4.85	3.79	0.87	0.93
Median	0.19	0.59	2.99	2.19	0.62	0.63
Standard Deviation	0.38	1.43	5.34	4.98	1.07	1.76
CV	1.37	1.46	1.10	1.31	1.22	1.89

Table 14.4:BC Mineralisation Domain, Uncut Au g/t Statistics

Statistic	Low Grade	Medium Grade	High Grade	Laterite
Count	419	371	349	14
Minimum	0.00	0.00	0.01	0.02
Maximum	20.43	5.66	97.89	3.26
Mean	0.42	0.60	2.43	0.98
Median	0.19	0.31	0.98	0.51
Standard Deviation	1.11	0.76	6.34	1.03
CV	2.66	1.26	2.61	1.06

Table 14.5:BC Mineralisation Domain, Topcut Au g/t Statistics

Statistic	Low Grade	Medium Grade	High Grade	Laterite
Count	419	371	349	14
Minimum	0.00	0.00	0.023	0.02
Maximum	7	5	30	3.26
Mean	0.41	0.60	2.21	0.98
Median	0.19	0.31	1.16	0.51
Standard Deviation	0.97	0.74	3.63	1.03
CV	2.36	1.24	1.64	1.06

Table 14.6:Argo Mineralisation Domains Au g/t Statistics

Statistic	Fouwagbe Uncut	Fouwagbe Topcut	Sounsoun Uncut	Sounsoun Topcut
Count	498	498	238	538
Minimum	0.005	0.005	0.005	0.005
Maximum	148.0	148.0	61.39	61.39
Mean	1.76	1.52	0.97	0.88
Median	0.22	0.22	0.27	0.148
Standard Deviation	7.80	4.33	4.00	2.88
CV	4.43	2.84	4.11	3.28

14.4 Variography

Experimental variograms were produced from the mineralised domain composite datasets using Supervisor software. For all domains, a normal scores transformation was applied to remove short-scale statistical noise and help model the underlying variability.

In general, the variograms are only moderately well structured, even in normal space, with moderate to high nuggets and short ranges. For laterite domains, the maximum continuity is in the horizontal plane; at NEB, both the low-grade and medium-grade domains had the maximum continuity downdip, but a shallow southward plunge to the second direction that may reflect a primary structural feature.

After modelling variograms, the results were back-transformed into sample space, and the final variogram models used for grade estimation.

14.5 Block Models

Surpac block models were created for the Mineral Resource estimates, one for each deposit. The cell size and subcell sizes were chosen to adequately model the wireframes at their resolution; for the Argo deposits the blocks are much smaller than the drillhole spacing and local block estimates are therefore not reliable. The block models are summarised in Table 14.7 to Table 14.10.

Table 14.7: Block Model bankan_ne_202307.mdl Dimensions

	X	Y	Z
Minimum Coordinates	117 3500	394 800	9 500
Maximum Coordinates	117 6560	397 300	10 600
Parent Block Size	20	10	5
Subcell Block Size	10	5	0.625
Rotation	0	0	0

Table 14.8: Block Model bankan_creek_202307.mdl Dimensions

	X	Y	Z
Minimum Coordinates	117 3500	393 000	10 000
Maximum Coordinates	117 4700	394 000	10 450
Parent Block Size	20	10	5
Subcell Block Size	5	5	0.625
Rotation	0	0	0

Table 14.9: Block Model fouwagbe_resource202502.mdl Dimensions

	X	Y	Z
Minimum Coordinates	119 1750	393 000	10 000
Maximum Coordinates	119 3250	395 550	10 500
Parent Block Size	10	10	2.5
Subcell Block Size	2.5	2.5	2.5
Rotation	0	0	0

Table 14.10: Block Model sounsoun202502.mdl Dimensions

	X	Y	Z
Minimum Coordinates	118 8000	391 000	10 000
Maximum Coordinates	119 0000	394 000	10 500
Parent Block Size	10	10	5
Subcell Block Size	2.5	2.5	2.5
Rotation	0	0	0

14.6 Quantitative Kriging Neighbourhood Analysis

Kriging parameters were chosen after a Quantitative Kriging Neighbourhood Analysis of the grade-estimate was completed in Supervisor software. The results of the analyses showed that the effect of increasing the maximum number of composites beyond 32 had marginal effects on the quality of the local block estimate (as measured by the kriging efficiency and slope of regression), although no negative composite weights were assigned.

14.7 Estimation

Gold grades were estimated into the flagged domain blocks using Ordinary Kriging. The kriging estimation parameters were chosen from the kriging neighbourhood analysis; a second pass for the Medium Grade domain at NEB was implemented to ensure all blocks were estimated. The parameters are tabulated in Table 14.11 to Table 14.13.

Table 14.11: NEB Kriging Estimation Parameters

	High-Grade Pass 1	High-Grade Pass 2	Medium -Grade	Low-Grade	GBE Pass 1	GBE Pass 2	Laterite
Minimum composites	8	8	8	4	8	8	8
Maximum composites	22	22	24	32	20	20	22
Rotation about Z	250	250	218	322	220	220	350
Rotation about X	-50	-50	-42	-42	-60	-60	0
Rotation about Y	0	0	-31	31	0	0	0

	High-Grade Pass 1	High-Grade Pass 2	Medium-Grade	Low-Grade	GBE Pass 1	GBE Pass 2	Laterite
Major search	160 m	250 m	150 m	320 m	90 m	270 m	135 m
Semi-major search	160 m	250 m	150 m	320 m	90 m	270 m	135 m
Minor search	64 m	100 m	50	59.3 m	10 m	30 m	135 m
Variogram type	Nested Spherical						
C_0	0.74	0.74	0.37	0.67	0.36	0.36	0.81
C_1	0.19	0.19	0.37	0.22	0.22	0.22	0.12
A_1 Major	8 m	8 m	5 m	8 m	4 m	4 m	3 m
A_1 Semi-major	8 m	8 m	5 m	8 m	4 m	4 m	3 m
A_1 Minor	3.2 m	3.2 m	5 m	8 m	4 m	4 m	3 m
C_2	0.07	0.07	0.20	0.19	0.17	0.17	0.06
A_2 Major	15 m	15 m	25 m	80 m	60 m	60 m	50 m
A_2 Semi-major	15 m	15 m	10 m	80 m	60 m	60 m	50 m
A_2 Minor	6 m	6 m	10 m	24.2 m	10.9 m	10.9 m	11.1 m
C_3			0.06	0.02			0.01
A_3 Major			50 m	160 m			90 m
A_3 Semi-major			50 m	160 m			90 m
A_3 Minor			16.7 m	59.3 m		0.36	75 m

Table 14.12: BC Kriging Estimation Parameters

	High-Grade Pass 1	High-Grade Pass 2	Medium-Grade	Low-Grade Pass 1	Low-Grade Pass 2	Laterite
Minimum composites	8	8	8	8	8	8
Maximum composites	24	24	24	24	24	24
Rotation about Z	90	90	190	20	20	0
Rotation about X	70	70	-50	50	50	0
Rotation about Y	0	0	0	0	0	0
Major search	60 m	60 m	240 m	135 m	300 m	75 m
Semi-major search	60 m	60 m	160 m	135 m	300 m	75 m
Minor search	60 m	60 m	50 m	135 m	300 m	75 m
Variogram type	Nested Spherical					
C_0	0.40	0.40	0.08	0.62	0.62	0.63

	High-Grade Pass 1	High-Grade Pass 2	Medium-Grade	Low-Grade Pass 1	Low-Grade Pass 2	Laterite
C_1	0.29	0.29	0.58	0.21	0.21	0.22
A_1 Major	3 m	3 m	7 m	6 m	6 m	3 m
A_1 Semi-major	3 m	3 m	4.7 m	6 m	6 m	3 m
A_1 Minor	3 m	3 m	1.5 m	6 m	6 m	3 m
C_2	0.26	0.26	0.34	0.14	0.14	0.11
A_2 Major	5 m	5 m	120 m	16 m	16 m	16 m
A_2 Semi-major	5 m	5 m	80 m	16 m	16 m	16 m
A_2 Minor	5 m	5 m	25 m	16 m	16 m	16 m
C_3	0.05	0.05		0.03	0.03	0.03
A_3 Major	12 m	12 m		34 m	34 m	21 m
A_3 Semi-major	12 m	12 m		34 m	34 m	21 m
A_3 Minor	12 m	12 m		34 m	34 m	21 m

Table 14.13: Argo Kriging Estimation Parameters

	Fouwagbe	Sounsoun
Minimum composites	8	8
Maximum composites	24	24
Rotation about Z	215	0
Rotation about X	-45	-70
Rotation about Y	0	0
Major search	200 m	200 m
Semi-major search	80 m	200 m
Minor search	80 m	20 m
Variogram type	Spherical	Spherical
C_0	0.69	0.45
C_1	0.23	0.48
A_1 Major	3 m	8 m
A_1 Semi-major	1.2 m	8 m
A_1 Minor	1.2 m	2.67 m
C_2	0.08	0.07
A_2 Major	10 m	80 m
A_2 Semi-major	4 m	80 m
A_2 Minor	4 m	26.7 m

14.8 Density

The density of selected core samples were measured using an immersion method. Samples of 10 cm to 30 cm of competent core were selected every 30 m to 50 m in waste lithologies, and every 5 m in shear zones. The samples were oven dried, then weighed in air and then immersed in water and density calculated using Archimedes' Principle.

9,704 measurements have been recorded and an analysis of the current density database was made, by classifying by the logged weathering and lithology. From a review of this data, the mean values were similar to those used in the August 2022 resource model, however 114 were identified as problematic, in that their density readings did not match the expected range. These were removed from the dataset before statistical analysis.

The densities applied are:

- Fresh tonalite – 2.8.
- Fresh mafic – 2.9.
- Fresh metasediment – 2.6
- Saprock – 2.3
- Saprolite and mottled zone – 1.6
- Laterite – 2.2

These are typical values for the logged rock types.

14.9 Validation

The resource estimates were validated against the input data. The validation comprised of:

- Visual validation to ensure all blocks were estimated.
- Comparison of the mean composite grade to the mean estimated block grade (Table 14.14 and Table 14.15), with the exception of the BC laterite domain, estimated grades were acceptably close to the input means.

Swath plots were used to demonstrate that the trends in the estimates are similar to the trends in the input data, albeit being highly smoothed in line with the expectations from the wide spaced data and variogram and search parameters.

Table 14.14: NEB Au g/t Validation Statistics

Domain	Composites	Blocks	Difference
Low Grade	0.28	0.18	-35.7%
Medium Grade	0.99	0.85	-14.1%
High Grade 1	4.88	5.36	9.8%

Domain	Composites	Blocks	Difference
High Grade 2	3.79	3.92	3.4%
GBE	0.87	0.80	-8.0%
Laterite	1.03	0.97	-5.8%

Table 14.15: BC Au g/t Validation Statistics

Domain	Composites	Blocks	Difference
Low Grade	0.41	0.44	7.3%
Medium Grade	0.60	0.53	-11.7%
High Grade	2.21	2.21	0.0%
Laterite	0.98	0.95	-3.1%

14.10 Prospect of Eventual Economic Extraction

To test the resource models for reasonable prospects for eventual economic extraction, preliminary open pit optimisations were completed for both NEB and BC. Inputs for the optimisations are largely generic with costs based on similar scale projects in the West African region (Table 14.16), metallurgical recoveries based on preliminary testwork (not the full suite of metallurgical testwork completed and reported in Section 13) and pit slopes based on analogous open pit operations (not on the detailed geotechnical investigation reported in Section 15.4). Note that these resource optimisation inputs are not necessarily the same as those used for the estimation of the Ore Reserve.

The NEB and BC preliminary optimised pits have surficial dimensions of 1,600 m by 900 m and 500 m by 400 m respectively and are located approximately 2.5 km apart. For NEB and BC a gold price of $1,800/oz was used, and for Argo $2,300/oz was used. The NEB and BC preliminary pits were found to be relatively insensitive to the gold price.

For the NEB underground an underground mining cost of $65 per tonne gave a calculated cut-off grade of 1.66 g/t and therefore the entire high-grade interpretation (which uses a cut-off grade of 2.0 g/t) below the preliminary optimised NEB pit shell was reported as a Mineral Resource.

Table 14.16: Preliminary Pit Optimisation Parameters

Parameter	NEB and BC	Argo
Gold Price	US$1,800/oz	US$2,300/oz
Discount Rate	5%	5%
Processing Rate	4 Mtpa	4 Mtpa
Ore Loss	4%	4%
Dilution	5%	5%
Bench Height	5 m	5 m
Laterite Pit Slope	38°	38°

Parameter	NEB and BC	Argo
Saprolite Pit Slope	48°	48°
Fresh Pit Slope	50°	50°
Base Mining Cost	US$1.92/t	US$1.92/t
Laterite Processing cost	US$19.90/t	US$19.90/t
Saprolite Processing Cost	US$21.40/t	US$21.40/t
Fresh Processing Cost	US$24.73/t	US$24.73/t
Laterite Metallurgical Recovery	95%	95%
Saprolite Metallurgical Recovery	94%	94%
Fresh Metallurgical Recovery	94%	94%
Gold Selling cost	US$72/oz	US$72/oz

14.11 Classification

The Mineral Resource was classified as Indicated and Inferred based on the level of geological understanding of the mineralisation, quality of samples, and drillhole spacing.

At NEB the drill spacing across the majority of resource pit shell has been closed to 80 m by 40 m, resulting in 3.90 Moz or 98% of the Open Pit Mineral Resource now being classified as Indicated. Inferred comprises some separate zones in the footwall, any open pit blocks in the low-grade domain above the cut-off, the entire underground resource. The majority of GBE deposit, where the central core of the mineralisation within 70 m of the natural surface is Indicated, with deeper and along strike extensions Inferred pending further infill drilling.

At BC, the drill spacing varies from 40 m by 40 m to wider than 80 m at the bottom of the model. The core area has been classified as Indicated in the upper 70 m of the deposit (above 300 mRL) where the results and interpretation are consistent from hole to hole. At deeper levels, additional drilling is required to confirm the continuity between the several lodes and the Mineral Resource is classified Inferred.

Fouwagbe has been drilled on approximately 50 m by 50 m spacing. Sounsoun has been drilled on two separate grids. The initial holes were drilled to 135° (to the SE) on a 100 m by 50 m grid, infilled to 50 m by 50 m in places. After analysis of the results, it was apparent that the mineralisation was oriented nearly east-west, so subsequent holes were drilled to 090° (to the south) to infill the previous grid to approximately 50 m by 50 m. Despite the drill spacing, there is still significant uncertainty over the geological controls and continuity, as well as insufficient density measurements. The resources are classified Inferred.

The classification reflects the overall level of confidence in mineralised domain continuity based the mineralisation drill sample data numbers, spacing and orientation. Overall mineralisation trends are reasonably consistent within the various lithotypes over numerous drill sections.

14.12 Reasonable Prospects

The Qualified Person considers that there are reasonable prospects for eventual economic extraction on the following basis:

- Drilling, logging and sampling are of a sufficient quality, as demonstrated by the QAQC programme results, to allow interpretation and estimation of a Mineral Resource.
- Size and shape of the mineralisation interpreted from the drilling results is of a scale and shape amenable to open pit mining, and analogous to other similar deposits in West Africa.
- Infill drilling during the project's life has confirmed previous interpretations and added confidence in the geological and grade models.
- Preliminary metallurgical testwork and petrographic examination of the mineralisation suggest that the gold is readily recoverable using commonly used processing methods.
- The project is located in a jurisdiction and a region with a previous history of successfully developing and supporting similar scale open pit gold mines and PDI has engaged in the process to apply the granting of a mining concession from the Government of Guinea, including consideration of mine engineering, processing, mine infrastructure, environmental, social and legal aspects of the Project.
- PDI's tenements are sufficiently extensive that a mining concession large enough for a mine and associated infrastructure could be granted.
- Technical, economic and execution risks associated with the Project are sufficiently expressed by the classification of the resource as Indicated and Inferred.

14.13 Mineral Resource Estimate

The Mineral Resource estimates for the Bankan Gold Project are shown in Table 14.17 and Table 14.18 constrained by open pit optimisations and the high-grade domain below the NEB preliminary optimised pit.

Although the Mineral Resource has been reported within the preliminary open pit optimisation, a decision has been made to optimise the overall Project by mining a smaller open pit at NEB, and plan to mine some of the open pit Mineral Resource from an underground mining operation. As the underground cut-off grade is higher than the open pit cut-off grade, some of the lower grade parts of the open pit resource will be effectively sterilised and is unlikely meet the reasonable prospects test following the commencement of underground mining. When the final decision is made to commence the underground mining operation, the Mineral Resource should be re-stated to reflect the planned sterilisation of this portion of the open pit Mineral Resource.

Table 14.17: Mineral Resource Estimate (NEB and BC)

Deposit	Type	Classification	Cut-off g/t	Tonnes Mt	Grade Au g/t	Contained Metal koz Au
NEB	Open Pit	Indicated	0.5	78.4	1.55	3,900
		Inferred	0.5	3.1	0.91	92
	Underground	Inferred	2.0	6.8	4.07	896
BC	Open Pit	Indicated	0.4	5.3	1.42	244
		Inferred	0.4	6.9	1.09	243
Total Indicated				**83.7**	**1.54**	**4,144**
Total Inferred				**16.8**	**2.28**	**1,231**

Notes:

1. The Mineral Resources is estimated with all drilling data available on 29th July 2023 (NEB and BC).
2. The Mineral Resource is reported in accordance with the JORC Code 2012 Edition and CIM 2014 Definition Standards and has an effective date of 7 August 2023.
3. The Mineral Resources (NEB and BC) are inclusive of Mineral Reserves.
4. The Qualified Person is Phil Jankowski FAusIMM of ERM.
5. The open pit Mineral Resources are constrained by optimised pit shells using a metal price of US$1,800 per ounce Au (NEB and BC) and process recovery of 94%. The NEB underground Mineral Resource is constrained by the high-grade domain below the NEB optimised pit shell.
6. The NEB open pit Mineral Resource is reported using a notional maximal open pit option. Current reserve planning is for a smaller open pit and an accelerated underground operation, where a higher grade (>2g/t) part of the open pit Mineral Resource is mined from underground. If this option is chosen, some of the lower grade open pit Mineral Resource may be effectively sterilised.
7. Rounding may lead to minor apparent discrepancies.

Table 14.18: Mineral Resource Estimate (Fouwagbe and Sounsoun)

Deposit	Type	Classification	Cut-off g/t	Tonnes Mt	Grade Au g/t	Contained Metal koz Au
Fouwagbe	Open Pit	Inferred	0.5	2.2	1.68	119
Sounsoun	Open Pit	Inferred	0.5	0.9	1.19	34
Total Inferred				**3.1**	**1.54**	**153**

Notes:

1. The Mineral Resources is estimated with all drilling data available at 25th of February 2025 (Fouwagbe and Sounsoun).
2. The Mineral Resource is reported in accordance with the JORC Code 2012 Edition and CIM 2014 Definition Standards and has an effective date of 23 April 2025.
3. The Qualified Person is Phil Jankowski FAusIMM of ERM.
4. The open pit Mineral Resources are constrained by optimised pit shells using a metal price of US$2,300 per ounce Au (Fouwagbe and Sounsoun) and process recovery of 94%.
5. Rounding may lead to minor apparent discrepancies.
6. The Mineral Resources reported at Fouwagbe and Sounsoun are not considered further in the DFS.
7. PDI has been made aware that, on 26 May 2025, Guinea's MMG announced the revocation of over 100 exploration permits, including the Argo (which hosts the Fouwagbe and Sounsoun deposits) and Bokoro exploration permits held by PDI group companies. The applications for extension of these permits were submitted to the MMG in 2021 and 2023, respectively. PDI has not received any formal communication from the Guinean government on the matter and intends to work diligently with the MMG to achieve the granting of the renewals.

At the time of the Technical Report, the Qualified Persons are not aware of any environmental, legal, socio-economic, marketing or other relevant conditions, that would materially affect the estimated Mineral Resources of the Bankan Gold Project.

15 MINERAL RESERVE ESTIMATE

15.1 Introduction

PDI engaged Orelogy to estimate a Mineral Reserve for the Project based on the latest available resource model as part of the DFS.

The Mineral Reserve developed by Orelogy as part of the DFS was announced by PDI on 25 June 2025 (Predictive Discovery Limited, 2025). PDI and Orelogy confirm that they are not aware of any new information or data that materially affects the information included in the corresponding market announcement, and that all material assumptions and technical parameters underpinning the Mineral Reserve estimates in the market announcement continue to apply and have not materially changed.

The Mineral Reserve was estimated from the Mineral Resource after consideration of the level of confidence in the Mineral Resource and taking account of material and relevant modifying factors including mining, processing, infrastructure, environmental, legal, social and commercial factors available at the time. The Probable Mineral Reserve estimate is based on Indicated Mineral Resources only. No Inferred Mineral Resource was included in the Mineral Reserve. The Mineral Reserve represents the economically mineable part of the Indicated Mineral Resources. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

It should be noted that the Mineral Resource on which the Mineral Reserve is based was constrained by a large optimisation shell to determine a reasonable prospect for eventual economic extraction (RPEEE). However, the final NEB pit design generated for the Open Pit Mineral Reserve was considerably smaller as it was based on an open pit / underground trade-off optimisation study. Consequently, the portion of Open Pit Indicated Resource below this smaller pit shell was considered for the Underground Mineral Reserve.

15.2 Mineral Reserve Statement

The total Probable Mineral Reserve is estimated at 51.6 Mt at 1.78 g/t Au with a contained gold content of 2,953 koz.

The Mineral Reserve for the Project is reported according to the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (Joint Ore Reserves Committee, 2012) and CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM Standing Committee on Reserve Definitions, 2014) and has an effective date of 31 July 2025. The Mineral Resource was converted by applying modifying factors. The Probable Mineral Reserve estimate is based on the Mineral Resource classified as Indicated. Table 15.1 presents a summary of the Mineral Reserves on a 100% Project basis at a US$1,800/oz gold price.

Table 15.1: Bankan Gold Project Mineral Reserves

Deposit	Mining Method	Classification	Tonnes (Mt)	Grade (g/t Au)	Contained Metal (koz Au)
NEB	Open Pit	Probable	40.2	1.36	1,751
	Underground	Probable	7.9	3.95	1,002
	Total		48.1	1.78	2,753
BC	Open Pit	Probable	3.5	1.78	200
Total Open Pit		Probable	43.7	1.39	1,951
Total Underground		Probable	7.9	3.95	1,002
Total Project		**Probable**	**51.6**	**1.78**	**2,953**

Notes:

1. The Mineral Reserve conforms with and uses the JORC Code (2012) and CIM (2014) definitions and has an effective date of 31 July 2025.
2. The Mineral Resources (NEB and BC) are inclusive of Mineral Reserves.
3. The Mineral Reserve was evaluated using a gold price of US$ 1,800 per ounce.
4. The Mineral Reserve was evaluated using variable cut-off grades as described in Table 15.17 and Table 15.24.
5. Ore block grade and tonnage dilution was incorporated into the model.
6. All figures are rounded to reflect appropriate levels of confidence.
7. Apparent differences may occur due to rounding.
8. The Qualified Person responsible for the Open Pit component of the Mineral Reserve is Mr Ross Cheyne, Principal Consultant with Orelogy Consulting Pty Ltd. Mr Cheyne visited the site in January 2025.
9. The Qualified Person responsible for the Underground component of the Mineral Reserve is Mr Julian Broomfield, Principal Consultant with Orelogy Consulting Pty Ltd.

At the time of the Technical Report, the Qualified Persons are not aware of any environmental, legal, socio-economic, marketing or other relevant conditions, that would materially affect the estimated Mineral Reserve of the Bankan Gold Project.

The Qualified Persons are of the opinion that the proposed mine plan is technically achievable and that all technical proposals made for the operational phase involve the application of conventional technology that is widely utilized in the gold industry in West Africa. In addition, financial modelling completed as part of the DFS shows that the Project is economically viable under current assumptions and all material modifying factors (mining, processing, infrastructure, environmental, legal, social and commercial) were considered during the Mineral Reserve estimation process.

The Mineral Resources, geotechnical investigation and assumptions, hydrogeological investigation and assumptions and optimisation processes that underpin the estimation of the Mineral Reserves are outlined in the following sections.

15.3 Mineral Resources

The Mineral Resource reported for Bankan as of 7 August 2023 was utilised in determining the Mineral Reserve. This is the same model as used for the Project Pre-feasibility Study (Predictive Discovery Limited, 2024).

15.3.1 Resource Block Model Conversion

The Mineral Resource block models were provided in Surpac format (.mdl) with the structure described in Table 15.2.

Table 15.2: Resource Model Parameters

Detail		NEB/GBE	BC
Model Name		bankan_ne_202307_2025.mdl	bankan_creek_202306.mdl
Origin	Easting	394,800	393,000
	Northing	1,173,500	1,173,500
	Level	9500	10000
Rotation	angle	0	0
Extent	Easting (m)	2,500	1,000
	Northing (m)	3,060	1,200
	RL (m)	1,100	450
Parent Block Size	X (m)	10	10
	Y (m)	20	20
	Z (m)	5	5
Minimum Sub-block Size	X (m)	5	5
	Y (m)	10	5
	Z (m)	0.625	2.5

The underground Mineral Reserve estimation used the resource model directly. For the open pit Mineral Reserve estimate, the Mineral Resource block model was converted to an ore parcel model for use in the Hexagon MineSight mine planning software and the subsequent application of dilution and ore loss. The regularised MineSight models had a single block size of 10 m by 10 m by 5 m (X by Y by Z) for both NEB/GBE and BC. The reconciliation of the converted resource model to the original resource model is shown in Table 15.3.

Table 15.3: Bankan Resource Model Parameters

Deposit	Mineralised Material[1]	Waste Material
NEB/GBE	0.1%	0.2%
BC	0.0%	0.5%

Notes:

1. Au greater than 0 g/t.

15.3.2 Resource Classifications

The mineralised material of the Mineral Resource model is classified as Indicated or Inferred in line with the JORC (2012) Code. There were five (5) key lithologies within the resource model that were utilised as part of the mine planning process.

- Laterite.
- Mottled clay.
- Saprolite clay.
- Saprock.
- Fresh.

The ore parcels in the converted model were defined by the initial assumed cut-off grades as follows:

- Laterite/Saprock, 0.49 g/t.
- Saprolite clay/Mottled clay, 0.46 g/t.
- Fresh, 0.56 g/t.

15.4 Geotechnical

Geotechnical investigations were undertaken by Peter O'Bryan and Associates providing a review of work carried out in the PFS and then updating the design parameters based on site investigations carried out as part of the DFS (Peter O'Bryan and Associates, 2025).

15.4.1 Investigations

Ground conditions influencing stability in the proposed open pit and underground mines have been investigated using:

- Current geological interpretations developed through the course of the exploration of the deposits and including preliminary geotechnical logging of the drill core.
- Data contained in geological, structural geological and geotechnical logs compiled by PDI from diamond cored geotechnical investigation boreholes and historic exploration boreholes. This has included:
  - Six PQ3 DDHs drilled specifically and logged in detail in the NEB pit area.
  - One HQ3 DDH drilled specifically and logged in detail in the underground area of the NEB deposit.
  - One PQ3 to HQ3 DDH drilled specifically and logged in detail in the BC pit area.
  - Detailed re-logging of six historic exploration holes, comprising 938 metres of drilling in the underground area of the NEB deposit.
- Laboratory measurement of physical properties of representative samples of country rocks taken from the drilling described above.

- Experience in geotechnical assessment and review in similar geological and geotechnical settings.

15.4.2 Open Pit Geotechnical Design

Geotechnical assessments have been performed for the NEB, GBE and BC pit areas. Geotechnical assessment of proposed open pit and underground mining at the NEB deposit has been carried out to DFS level. Geotechnical assessment of proposed open pit mining at the BC and GBE deposits has been carried out to preliminary assessment level.

15.4.2.1 NEB Open Pit

Empirical classifications of rock mass quality indicate:

- The extremely to completely weathered horizon at the NEB deposit has a mean rock mass rating (RMR) of was 19, indicating very poor rock.
- Highly weathered rocks have an RMR range from 19 to 61, indicating very poor to good rock, with a mean value of approximately 40, indicating poor rock.
- Transitional (moderately weathered) rocks have an RMR range from 20 to 63, indicating very poor to good rock, with an average value of 46, indicating fair rock.
- Slightly weathered rock has an RMR range from 47 to 85, indicating fair to very good rock, with an average of 59, indicating fair rock.
- Overall, fresh rock core was assessed to an RMR range from 39 to 92, indicating poor to very good rock, with an average of 75, indicating good rock.

The depth from surface to the top of fresh rock (TOFR) ranges from approximately 50 m to 77 m.

Some other observations from the logging and testing include:

- Fresh rocks are generally very strong.
- The mean peak and residual shear strengths for natural rock defects are relatively high.
- Based on assessed rock mass conditions, stability within weathered rocks of the uppermost pit wall sectors will be influenced by low strength weathered rocks and relict geological structures, with potential impacts from possible adverse influence of undissipated groundwater pressures.
- The stability of slopes mined in fresh rocks will be governed dominantly by the orientation, persistence and shear strength of geological structures exposed by or located close to pit walls ± possible adverse groundwater pressures.

Preliminary definition of NEB open pit geotechnical design domains has been based on logged rock weathering grades and the orientation of interpreted local mineralisation trends. The two geotechnical designs domains, shown in Figure 15.1, for the NEB pit comprise:

- NEB Domain A, footwall to mineralisation.
- NEB Domain B comprising the remaining pit sectors.

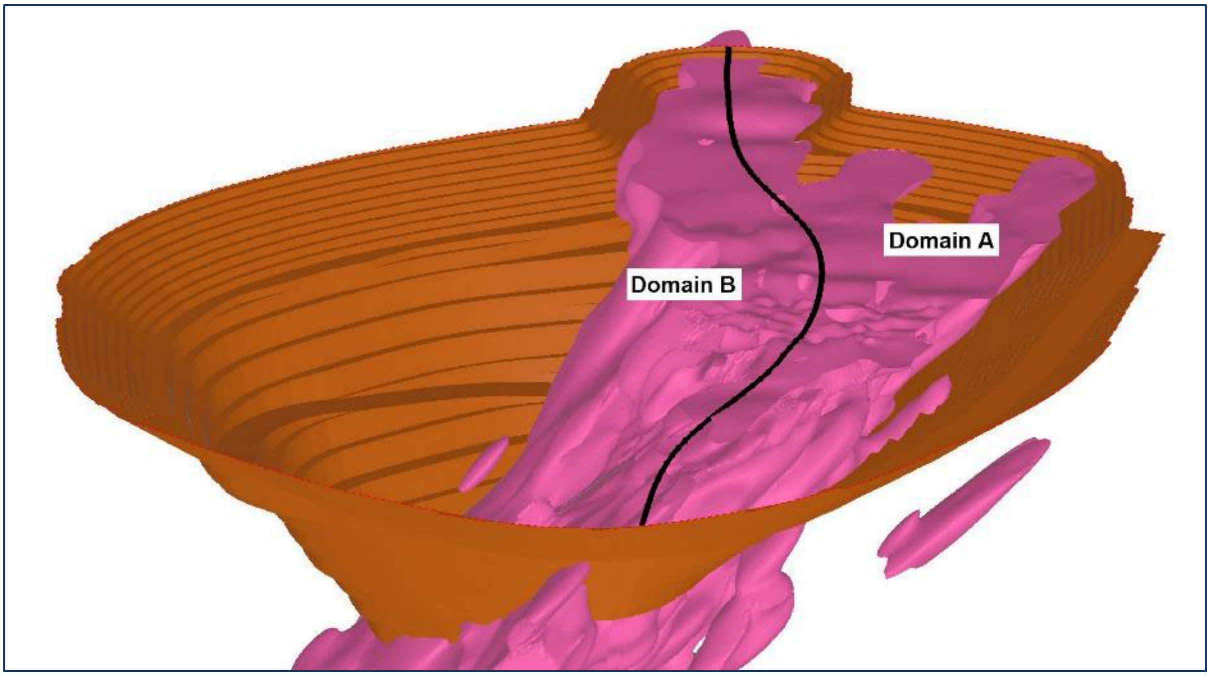

Figure 15.1: NEB Pit Domains (Representative View Looking North)

Base case geotechnical design parameters have been developed for the NEB pit as outlined in Table 15.4 and shown visually in Figure 15.2 and Figure 15.3.

Table 15.4: NEB Pit Geotechnical Design Parameters

Parameter	Units	Domain A		Domain B	
		Surface to TOFR	TOFR to Base of Pit	Surface to TOFR	TOFR to Base of Pit
Batter Height	m	5	10	5	20
Batter Face Angle	°	60	90	60	75
Berm Width	m	4.5	>4[1]	4.5	8
Inter-Ramp Angles	°	34.1	68[1]	34.1	56.3
Bench Stack Berm	m	Not Required	Not Required	Not Required	15 at <80m intervals
Domain Berm Width	m	>15		>15	
Overall Slope Angle (Crest to Toe)	°	39		49	

Notes:

1. Inter-ramp angle in Domain A is limited by dip of mineralisation and which will likely result in berm widths of 12 m resulting in an inter-ramp angle 39.8°.

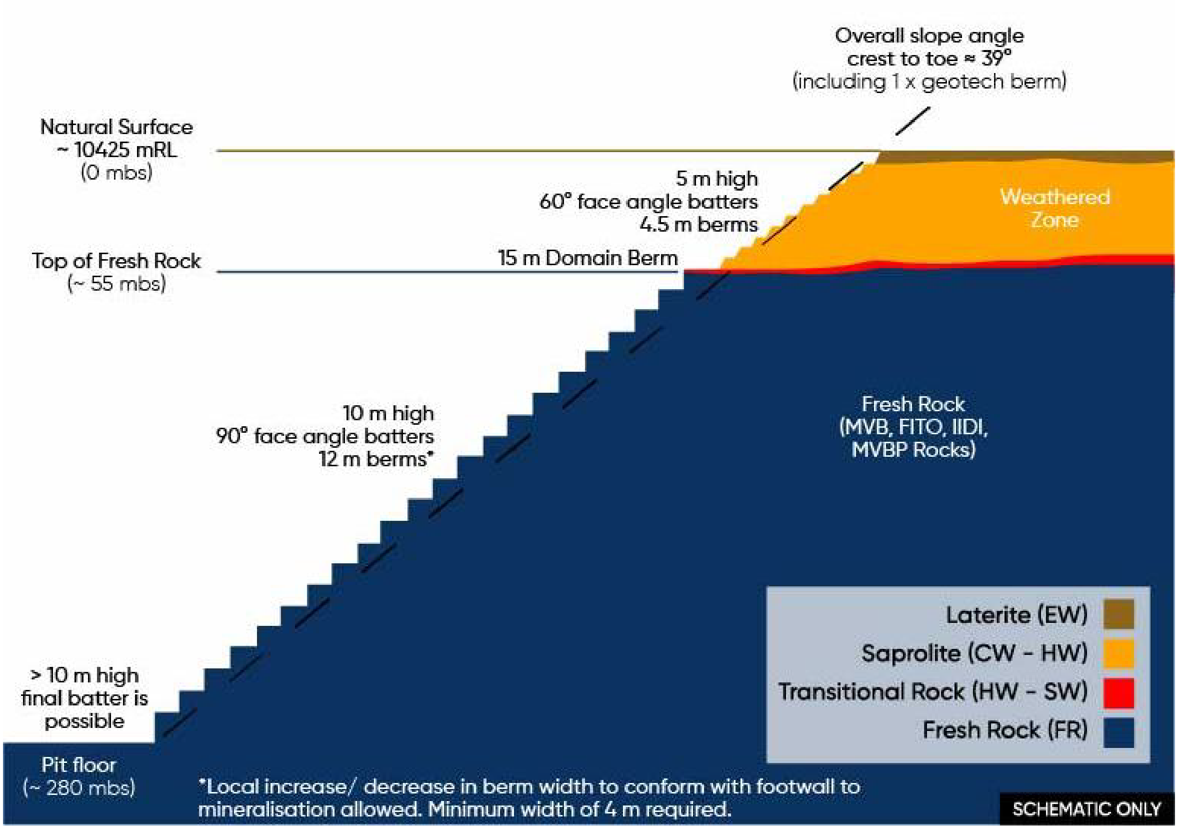

Figure 15.2: NEB Pit Domain A Wall Design Parameters

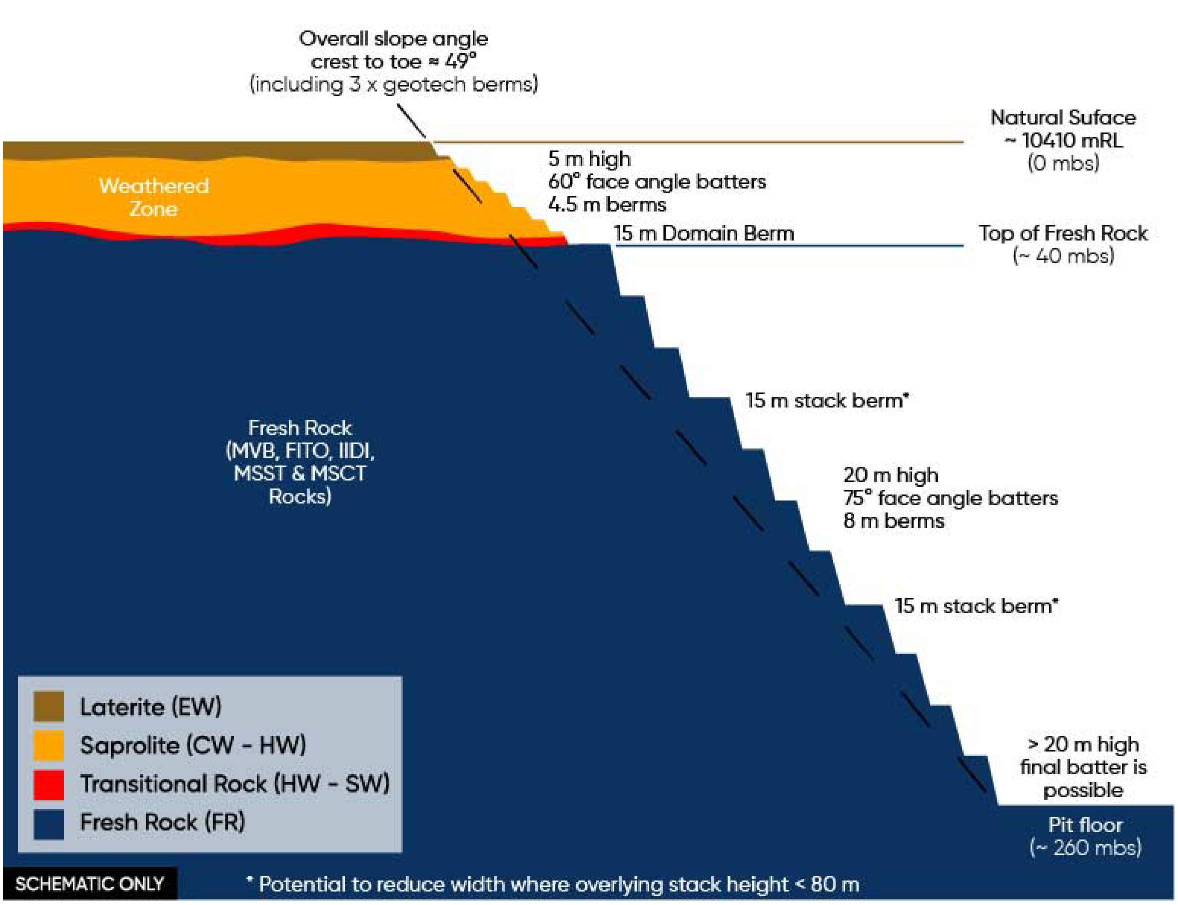

Figure 15.3: NEB Pit Domain B Wall Design Parameters

15.4.2.2 BC Open Pit

Preliminary definition of BC open pit geotechnical design domains has been based on rock weathering and the orientation of local mineralisation trends. The current level of geotechnical assessment for BC is preliminary and further geotechnical investigation and assessment is required prior to the commencement of mining.

The two geotechnical designs domains, shown in Figure 15.4, for the BC pit comprise:

- BC Domain A, footwall to mineralisation.
- BC Domain B, which encompasses the remaining pit sectors.

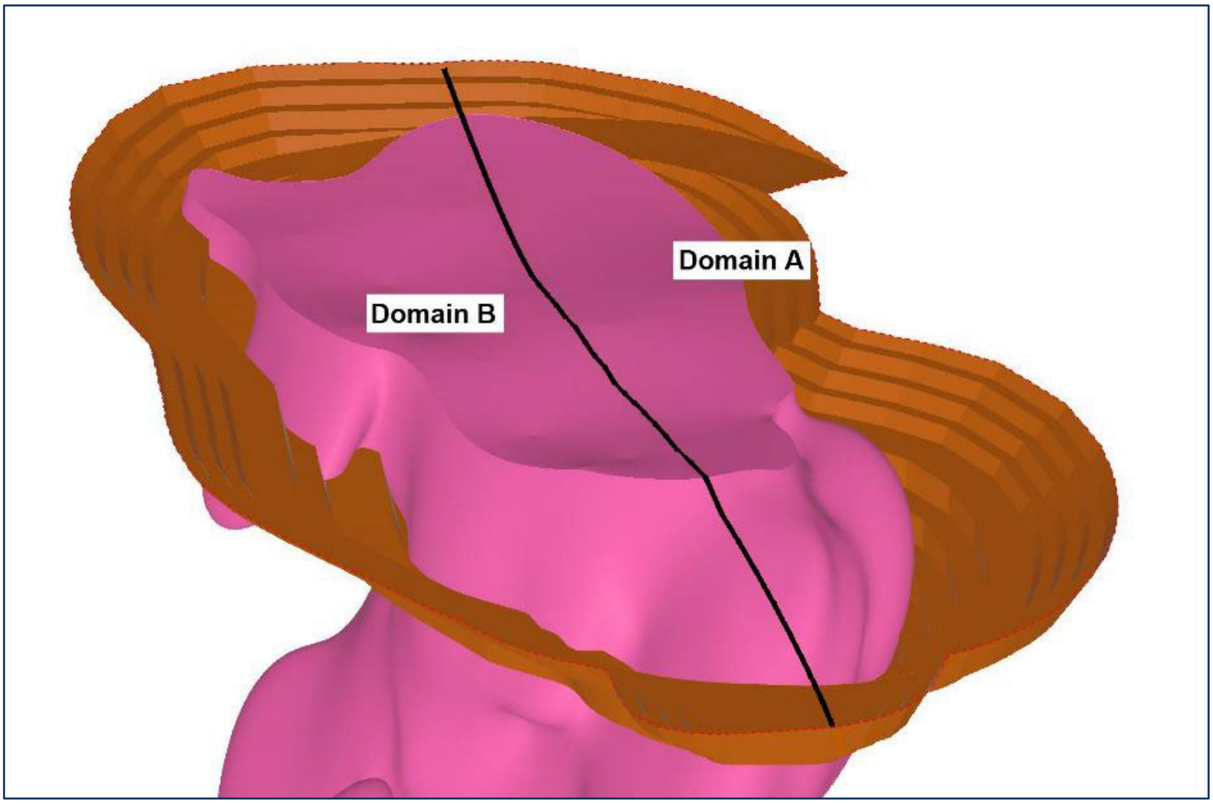

Figure 15.4: BC Geotechnical Pit Domains (Representative View Looking North)

Base case geotechnical design parameters have been developed for the BC pit as outlined in Table 15.5 and shown visually in Figure 15.5 and Figure 15.6.

Table 15.5: BC Pit Geotechnical Design Parameters

Parameter	Units	Domain A		Domain B	
		Surface to TOFR	TOFR to Base of Pit	Surface to TOFR	TOFR to Base of Pit
Batter Height	m	5	10	5	20
Batter Face Angle	°	60	90	60	75
Berm Width	m	4.5	>4[1]	4.5	8
Inter-Ramp Angles	°	34.1	68[1]	34.1	56.3
Bench Stack Berm	m	Not Required	Not Required	Not Required	15 at <80m intervals
Domain Berm Width	m	>15		>10	
Overall Slope Angle (Crest to Toe)	°	44		50	

Notes:

1. Inter-ramp angle in Domain A is limited by dip of mineralisation and which will likely result in berm widths of 8.5 m resulting in an inter-ramp angle 49.6°.

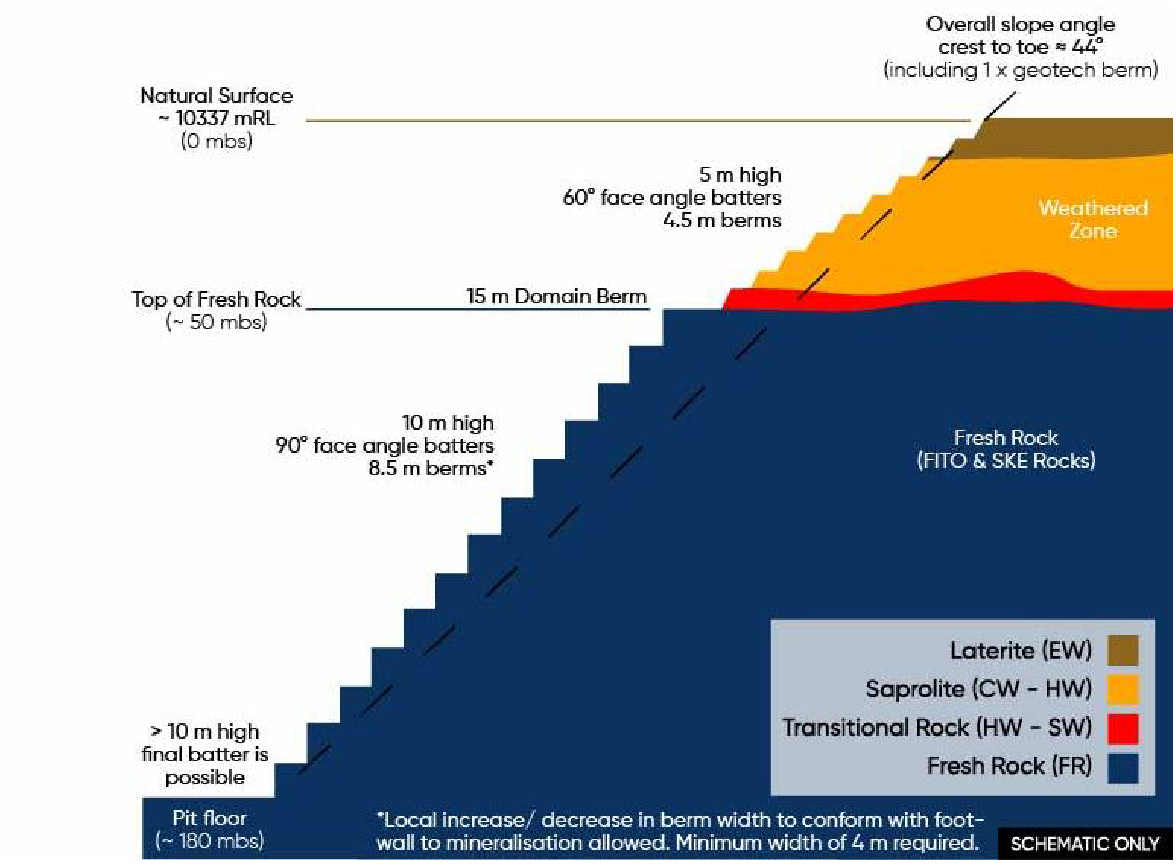

Figure 15.5: BC Pit Domain A Wall Design Parameters

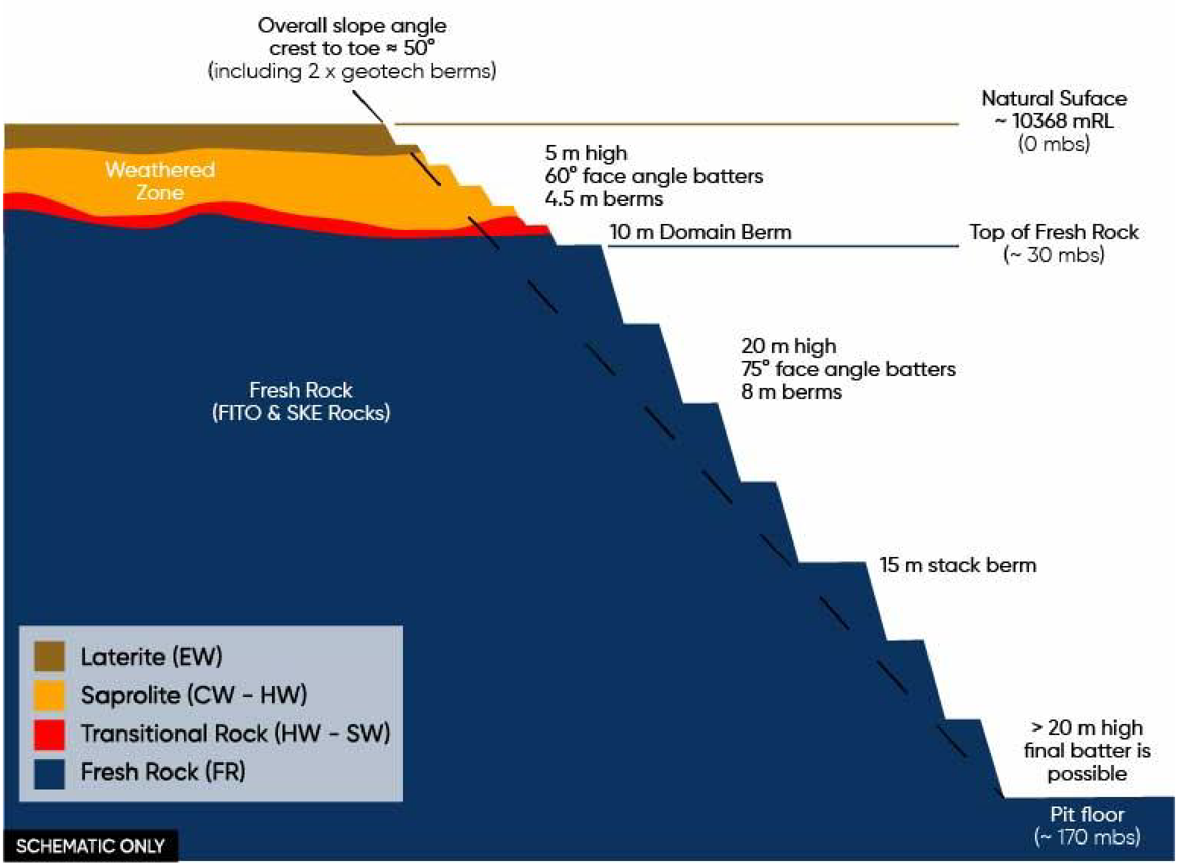

Figure 15.6: BC Pit Domain B Wall Design Parameters

15.4.2.3 GBE Open Pit

Current proposed open pit mining at the GBE deposit is relatively shallow and is anticipated to be limited to less than 25 m below the TOFR, gaining sufficient depth to allow establishment of a portal to provide decline underground access to the NEB deposit.

Preliminary definition of GBE open pit geotechnical design domains has been based on rock weathering only. The current level of geotechnical assessment for GBE is preliminary and further geotechnical investigation and assessment is required prior to the commencement of mining.

Base case geotechnical design parameters have been derived for all sectors of the GBE pit, as outlined in Table 15.6 and shown visually in Figure 15.7.

Table 15.6: GBE Pit Geotechnical Design Parameters

Parameter	Units	All Sectors	
		Surface to TOFR	TOFR to Base of Pit
Batter Height	m	5	20
Batter Face Angle	°	60	75
Berm Width	m	4.5	8
Inter-Ramp Angles	°	34.1	56.3
Bench Stack Berm	m	Not Required	Not Required
Domain Berm Width	m	>15	
Overall Slope Angle (Crest to Toe)	°	39	

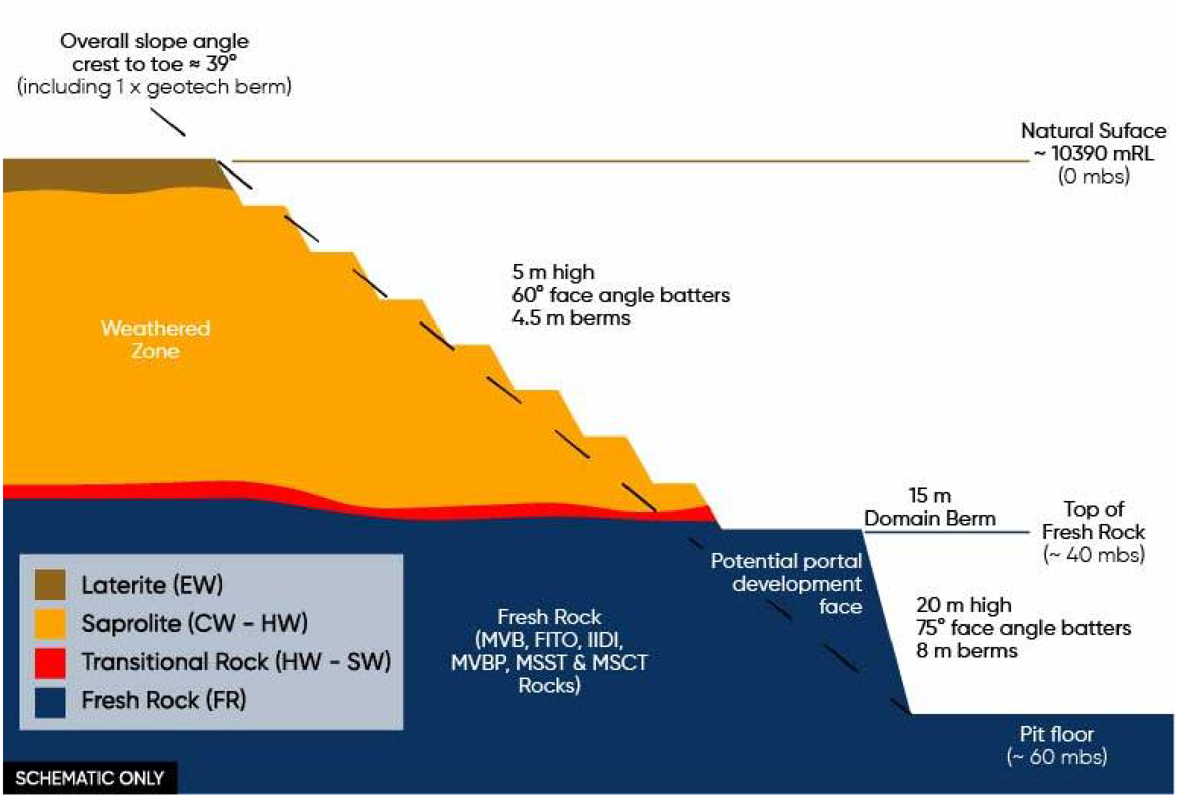

Figure 15.7: GBE Pit Wall Design Parameters

15.4.3 Underground Geotechnical Design

Assessment and analysis of future underground excavation stability has used:

- Interpretations made from review of drill logs and core photographs of relevant borehole cores.

- Empirical methods to estimate stable spans and ground support and reinforcement requirements.
- Experience-based assessment of rock mass conditions influencing underground development.

Some other observations from the logging and testing include:

- Intact fresh rocks are generally very strong.
- Rock defect shear strengths for disturbed natural rock defects are high.
- Rock mass conditions within and near planned stoping areas range from extremely poor to very good and are generally fair.
- Rock quality along the alignments of level access development is generally fair.
- Rock quality along the alignments of decline development ranges from very poor to very good.
- Localised intervals of extremely poor rock quality have been intersected by drilling.

Preliminary geotechnical assessment indicates that stoping can be performed via longhole open stoping (LHOS) in conjunction with use of cemented paste backfill.

The base case parameters outlined in Table 15.7 have been developed for underground stope design and configurations.

Table 15.7: NEB Underground Geotechnical Stope Design Parameters

Depth (mRL)	Maximum HW Height (m)	Maximum Strike Length (m)	Maximum Backs Width (m)
10160 – 10020	20	20	30
10020 – 9880	15	20	25
9880 – 9740	15	20	20
9740 – 9620	12	20	15

All stope voids are to be backfilled prior to progressive increase of total void dimensions.

In addition, base case ground support and reinforcement designs for NEB underground development and stopes have been developed and are summarised in Table 15.8.

Table 15.8: NEB Underground Geotechnical Ground Support and Reinforcement Requirements

Development Type/ Area	Ground Support & Reinforcement Requirements
Decline 5.8 m High x 5.5 m Wide	Mesh surface support to within ≤ 3.5 m of floor level. Pattern bolting using ≥ 2.4 m long friction bolts* installed at ≤ 1.4 m bolt spacings in backs and sidewalls to within ≤ 3.5 m of floor level.
Vent Drive 5.5 m High x 5.5 m Wide	Mesh surface support to within ≤ 3.5 m of floor level. Pattern bolting using ≥ 2.4 m long friction bolts* installed at ≤ 1.4 m bolt spacings in backs and sidewalls to within ≤ 3.5 m of floor level.
Ore Drive 5.0 m High x 5.0 m Wide (Except Sill Pillar Level)	Mesh surface support to within ≤ 3.5 m of floor level. Pattern bolting using ≥ 2.4 m long friction bolts* installed at ≤ 1.4 m bolt spacings in backs and sidewalls to within ≤ 3.5m of floor level.
Ore Drive Sill Pillar Levels 5.0 m High x 5.0 m Wide	≥ 50 mm FRS across backs and sidewalls to within 1.5m of floor. Pattern bolting using ≥ 2.4 m long Sandvik D47 MD Bolts installed at ≤ 1.2 m bolt spacings in backs and sidewalls to ≤ 1.5m of floor level.
Development in Weathered and/ or Poor Ground Conditions	≥ 50 mm FRS across backs and sidewall to within 1.5m of floor. Pattern bolting using ≥ 2.4 m long Sandvik D47 MD Bolts installed at ≤ 1.2 m bolt spacings in backs and sidewalls to ≤ 1.5 m of floor level.
All Intersections	≥ 6 m long twin bulbed strand cable bolts installed in backs on a 2.5 m x 2.0 m pattern through intersection span. All cable bolts to be full column grouted, plated and post tensioned to 5t.
Stope Hanging Wall (from HW Drive)	Rings two (2) of ≥ 8 m long (embedment in stope HW) twin bulbed strand cable bolts installed at bolt spacings (at hanging wall surface) of 2.0 m and at 2.5 m ring spacings. All cable bolts to be full-column grouted. Plating and post-tensioning to 5t to be carried out where access allows.
Stope Brows	Rings of three (3) ≥ 6 m long twin bulbed strand cable bolts installed in backs at 1.5 m bolt spacings. All cable bolts to be full-column grouted, plated and post-tensioned to 5t.
General	Minimum required pull load for 2.4m long friction bolts to be greater than 10t.

15.5 Hydrogeology

15.5.1 Geographical Setting

The site is drained in a southward direction towards the Niger River by a network of ephemeral streams which flow predominantly during the wet season. The topography of the area is gently undulating, with occasional rises of elevations of up to ~450 m above mean sea level (AMSL), approximately 100 m above the drainage lines. Bankan Creek drains the western portion and the proposed BC pit area from north to south where it joins the west to east flowing Niger River. A second tributary drains the NEB pit area, flowing eastwards, towards Kouroussa to the east and then enters the Niger River. Another minor tributary, just southeast of the planned accommodation village, flows in a south-easterly direction, through Kouroussa, and thereafter enters the Niger River.

Surface drainage lines are a source of groundwater recharge during the rainy season and, depending on the location of infrastructure, could pose a significant risk during flood events.

15.5.2 Groundwater Regime

As part of the DFS, an additional 16 boreholes were drilled and constructed, of which 14 underwent pump testing to build on the conceptual hydrogeological model, developed during the PFS based on previous drilling and pump testing. In addition, hydraulic permeability testing (packer testing) was conducted on deep geotechnical exploration boreholes.

Together with the geological model developed from extended mineral exploration drilling by PDI, the hydrogeological work completed for the DFS improved the overall understanding of the local and regional aquifer systems.

The groundwater assessment identified four main hydro-stratigraphic units:

- Weathered aquifer.
- Saprock or transition aquifer.
- Fresh bedrock and fractured rock aquifer.
- Alluvium aquifer.

The weathered aquifer (upper saprolite) has a high storage and low permeability with varying thickness ranging from 10 m to 80 m across site. As such it has significant storage and forms a leaky confining unit above the thinner, more permeable transition zone. Extensive shear zones may extend vertically through the hydrostratigraphic units, potentially creating hydraulic connections between the units they intersect.

In the case of NEB, where shear zones extend close to the surface, recharge of the transition zone aquifer below the saprolites occurs more readily and this in turn extends to fractures associated with the shear zone in the fresh rock below. These vertically extensive fault systems (shear zones) likely have the potential to compartmentalise aquifers, as the surrounding host rock, excluding the shear zones or fractures, exhibits significantly lower permeability.

In the lower lying areas, minor alluvial aquifers occur along drainage lines and are common along the Niger River flood basin.

Groundwater levels have been monitored monthly since the initiation of the baseline monitoring program in 2023. The program includes observations from 17 locations across the area, comprising community boreholes and wells, and also test boreholes drilled during the PFS. The groundwater level monitoring data indicates the following ranges:

- Minimum level, 0.6 metres below ground level (mBGL).
- Maximum level, 29 mBGL.
- Average level, 8.4 mBGL.

Groundwater recharge is expected to occur predominantly along shear zones which extend close to surface. Overall, recharge is estimated to range between 2% and 5% of annual rainfall.

As indicated earlier, hydraulic properties were derived from the aquifer testing programmes of both the PFS and DFS pump test results as well as from packer tests on deep inclined geotechnical boreholes. The pump test program comprised of constant rate discharge tests ranging from 12 h to 24 h. Once analysed, the data presented transmissivity ranges between 1 and 28 m^2/day, and the estimated hydraulic conductivity (K) ranges between 0.002 and 1.2 m/day. The low values typically represent the saprolite and fresh rock matrix, and the higher values the transition zone and shear zone. Importantly, and consistent with the conceptual model outlined in the PFS, hydraulic conductivity decreases with depth.

Water quality analyses from the hydrocensus and baseline monitoring shows that the groundwater in the Project area is generally of good to marginal quality, based on the World Health Organisation (WHO) guidelines for drinking water. However, sulphate, iron, manganese, arsenic, nitrate, zinc, nickel, pH level and aluminium concentrations are elevated above the WHO guideline for drinking water in some boreholes.

15.5.3 Predictive Numerical Groundwater Modelling

A scripted MODFLOW6 numerical groundwater model was developed, informed by the PFS model simulation results and the updated DFS groundwater data. It was designed to simulate groundwater hydrodynamics in greater detail both spatially and temporally for the life of mine and post mining period.

The model is made up of 20 m square cells across 15 layers (compared to 62.5 m cells and three used in the PFS) in the Project area and along known shear zones, allowing a more accurate representation of the pit shells and the underground works. The saprolite and transition layers are represented by a single layer each with the remaining 13 representing the fresh and fractured bedrock.

Quarterly temporal resolution is used throughout, and the model simulates transient open pit and underground development from yearly forecasted pit shells and yearly forecasted tunnelling and underground mining plans.

The filtered TSF will be fully lined with HDPE, and as such, seepage and associated mass transport are expected to be negligible and therefore not included in the predictive numerical model.

15.5.3.1 Mine Pits Inflows and Dewatering

A primary output for the numerical model was estimating the plausible range of mine pit inflows and dewatering volumes. For the base case the following groundwater inflow and dewatering estimates for the combined pits and underground, over 14 years from the commencement of site construction works to the end of the mining operations are as follows:

- Maximum inflow and dewatering of 8,800 m^3 per day in the second year of construction, the year prior to the commencement of processing.
- Minimum inflow and dewatering of 3,500 m^3 per day in year five of production.
- Average inflow and dewatering of 5,000 m^3 per day.

The pit inflows and dewatering bores form an integral part of the groundwater management.

15.5.4 Geochemical Assessment

Twelve rock samples were tested for acid base accounting during the PFS (six from BC and six from the NEB/GBE combined deposit referred to as NEB). To supplement the PFS geochemical assessment, a further 99 rock samples underwent geochemical assessment during the DFS. The rock samples represented proposed waste rock disposal materials from both the NEB and BC pit rock formations.

At NEB pit the waste rock will originate from the fresh mafic lithology (42%), saprolite lithology (22%) and tonalite lithology (14%). The remaining 22% is made up from the metasediments including saprock, mottled zone, shear zone, and laterite.

At BC pit, waste rock originates from tonalite lithology (35%), mafic lithology (25%) and saprolite lithology (19%) with the remainder made up of metasediments.

Acid base accounting, total sulphur and deionised water leachate testing was conducted on all of the samples. From the 99 samples, 60 were tested for mineral composition using x-ray diffraction (XRD).

From the acid base accounting it was concluded that:

- Overall risk of potential acid forming (PAF) rock material at the Project site is low, with most samples falling within the non-acid formation (NAF) category.
- From the 21 BC and 78 NEB samples, 89% of samples are NAF, 8% are uncertain, and 3% are PAF. The samples which tested as PAF are found in BC (mafic lithology) and in NEB (low-grade shear lithology). The uncertain behaviour of samples would have a low capacity to produce acid drainage since the net acid generation (NAG) pH is greater than 4.5.
- Total sulphur concentrations are generally low, less than 0.3%, which indicates a low risk of generating acid drainage.
- Carbonates will have capacity to neutralise the acidity that may be generated by sulphide oxidation and provide neutralisation. Siderite was also present, however, it can be dissolved, releasing iron and CO_2, influencing the overall acid drainage chemistry.
- Leaching analysis indicated that at neutral to alkaline pH, aluminium and arsenic are the main likely constituents of interest (consistent with regional ambient groundwater quality).
- Distilled water extract testing indicated that the leachate from the waste rock has a low overall metal content.
- Samples indicated low salinity, so saline drainage is not of concern.

On this basis, only high-level controls are required to manage the risk of acid generation in the waste rock.

Although elevated concentrations of various metals, including arsenic and aluminium, were identified in groundwater samples across the Project area, several monitoring boreholes will be installed hydraulically downgradient of the waste rock areas. Additionally, waste dump leachate will be contained and monitored to determine if it can be released into the environment.

15.5.5 Groundwater Management

Groundwater management at the Project will be stipulated by a groundwater monitoring and management plan (GWMMP), which will be guided by a trigger, action, response plan (TARP).

Mine dewatering for the NEB and BC pit areas is proposed in two stages:

- Pre-mining, using dewatering boreholes, to allow additional time for drainage of the less permeable geological units (i.e. saprolites).
- Operational dewatering through conventional dewatering methods.

Since the underground mine will be constructed and mined in conjunction with the NEB open pit (which is situated above the underground mine), primary dewatering will be in place for the upper geological zones (saprolite and transition zone), which are regarded as the regionally sensitive zones. Additional dewatering will be required when the decline tunnel and mine workings reach depths below the pit interface. Dewatering of underground workings is often undertaken with a combination of vertical wells, horizontal drains behind the working face and collection sumps within the mine workings.

Approximately 15 dewatering boreholes for the NEB and underground area and nine for the BC area will be installed. In addition, a further 11 out of pit monitoring boreholes will be installed to facilitate the ongoing monitoring of the ground water. Some of these boreholes have already been drilled as part of the PFS and DFS testing programmes. These dewatering bores will be operational early in the construction phase relevant to each pit to minimise impacts on the mining start up and to provide water for construction activities.

15.6 Open Pit Optimisation

The first stage of the conversion of a Mineral Resource into a mineable open pit Mineral Reserve is the open pit optimisation process. It is at this stage that all the most up-to-date physical, technical and economic parameters are applied to the orebody to generate the "ideal" open pit excavation geometry.

The Whittle™ open pit optimisation software tool was utilised to undertake this component of the study. Whittle™ is a recognised, off the shelf, optimisation package, and to a degree is considered the industry standard for open pit optimisation.

15.6.1 Overview

In broad terms, the process that Whittle™ undertakes is to vary the base input price by a range of factors (referred to as the revenue factors), up and down from a base value of 1. For any given revenue factor Whittle™ then produces a three-dimensional shape, or "shell", that generates the maximum possible value for all the input parameters and the associated factored price.

Lower factored prices will produce smaller shells and the higher price factor, the larger the shell. This results in a set of "nested" shells, with each shell lying inside the shell of the next largest revenue factor.

The value of each shell is reported at the base price. The effect of this is that:

- Shells with a revenue factor less than 1 are smaller and have less ore than the shell with revenue factor of 1. This reduces the revenue generated and therefore they will have a lower value.
- Shells with a revenue factor greater than 1 are bigger and have relatively more waste. This increases the costs and therefore they will have a lower value.
- The result is the classic Whittle™ cash flow curve that generally peaks at the base price (i.e., revenue factor of 1). The robustness of this shell is reflected in how quickly the value falls away either side of this peak.

These nested shells are important for several reasons, including:

- Smaller shells indicate the areas of highest value in the ore body and therefore give a guide as to where mining should commence.
- Larger shells provide an indication of how much additional mineralisation may become economic, or alternatively what current ore may become unviable, should input parameters change in the future.
- They permit Whittle™ to develop a "schedule" for mining the deposit over time and therefore allow a discounted cash flow (DCF) to be generated.

It is important to have some appreciation of how Whittle™ generates a DCF as these values are used in this study as a guide to both sensitivity and the shell selected as the basis for pit design. It should be noted that Whittle™ cash flows differ from accounting cash flows which include the effects of debt servicing, depreciation and taxation.

Whittle™ generates the following two standard DCF's:

- Worst Case DCF, which assumes that, for any given shell, extraction is undertaken bench by bench sequentially from the top to bottom of the pit. This results in a significant amount of any overlying overburden being removed prior to presentation of ore (i.e., a large pre-strip), and there are no initial shells or stages that access higher value ore earlier in the schedule. This is clearly not a preferred pit extraction scenario from a value perspective.
- Best Case DCF, which assumes that, for any given shell, extraction is undertaken sequentially from the smallest shell generated by the optimisation, through all the intervening shells out to the shell selected. This approach focuses on the highest value areas of the orebody and generates the least amount of waste stripping. As such it provides the highest DCF for the selected shell. However, this approach is constrained by practical considerations as it is effectively mining the orebody using an impractically high number of small incremental pushbacks that could not be achieved as:
  - Distances between successive shells are invariably too narrow to mine practically.

– Bench turnover required to mine the shell in the time frame is not achievable.

– The base of the shell is too small and constricted to allow for practical access or safe extraction.

Figure 15.8 illustrates a schematic representation of the worst case and best case DCF mining sequences. The figures indicate the amount of waste that needs to be moved to gain access to the orebody, and as can be seen the worst-case approach requires significantly more waste movement than the best case, hence two divergent cash flows are generated.

In essence, these two DCF'S provide the extremities or theoretical value that can be generated from a deposit. The shell that represents what will happen will in fact lie somewhere between these two endpoints.

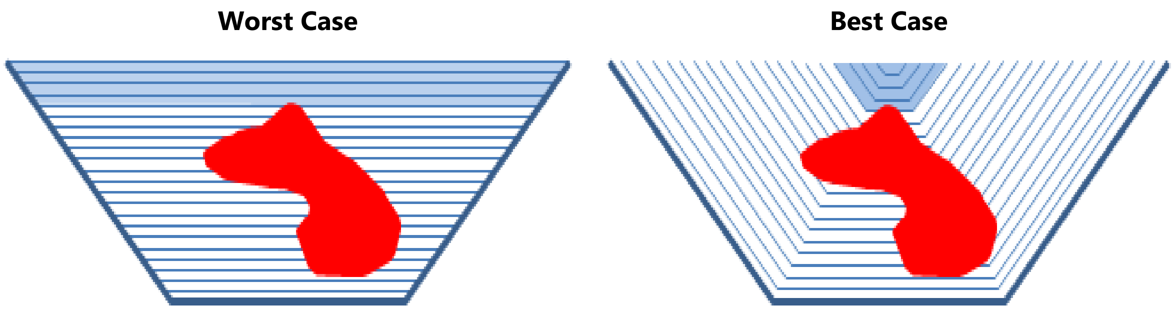

Figure 15.8: Whittle™ Mining Sequence

15.6.2 Diluted Mining Block Model

A primary component of the mine planning cycle is the conversion of a resource block model to a diluted "mining model" that is suitable for mine planning purposes. There are several approaches currently utilised in the mining industry to model dilution, including but not limited to:

- Fixed factors – i.e. the standard "Whittle approach".
- Block regularisation – Accumulating granular resource models to a regular block size reflecting a selective mining unit (SMU) size and accumulating all material within the regular sized block.
- Dilution skins or envelopes – Defining ore within a solid volume and then expanding that solid by a defined "skin width" and flagging the material captured as ore (i.e. diluent material).

Orelogy has developed an in-house proprietary approach to dilution that utilises an "ore parcel" model and then applies an internal dilution within each block that replicates an edge dilution, in a similar fashion to the "skin" approach but does this mathematically within the block. This approach does not require the generation of complex geometries and then re-modelling within them.

The Orelogy approach uses the following steps:

- Regularise the bock model.
- Determine a suitable skin or "mixing zone".
- Swapping of material within mixing zone.

Regularising the resource model to one block size maintains ore and waste proportions or "parcels" within the regular blocks. In this way the granularity of the sub-celled resource model is preserved within a regular block that is suitable for Whittle optimisation. The block size for regularisation does not need to reflect an SMU, it is more important that the block is large enough across strike that it captures the edge of orebody. Generally, the block height selected is the design bench height to align with the mining increment. This is shown diagrammatically in Figure 15.9.

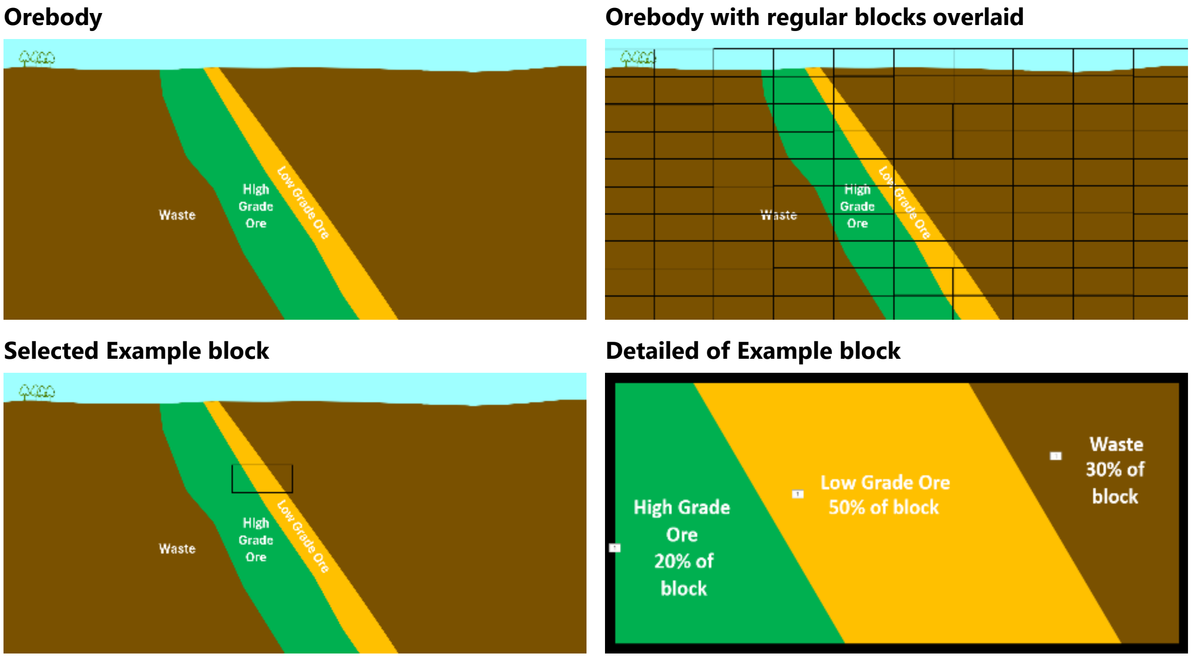

Figure 15.9: Regularisation Process to a Parcel Model (Orelogy 2025)

Determination of a suitable skin or "mixing zone" between the different parcels is based on the following:

- Bench height.
- Equipment size.
- Degree of blasting movement (i.e. free dig through to high powder factor blasting).
- Dip of the orebody.
- Strike of the orebody.

This is show in diagrammatically in example in Figure 15.10:

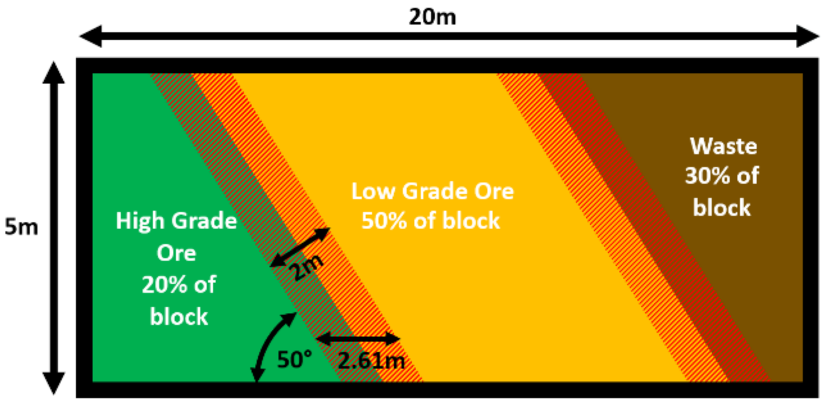

Figure 15.10: Determination of Mixing Zone (Orelogy 2025)

The key parameters used in this example are:

- Block Dimension of 20 m by 20 m by 5 m (X by Y by Z).
- True Mixing Width of 2m.
- Orebody Dip of 50°.

This yields an apparent mixing width and mixing volume determined as follows:

Apparent Mixing Width = 1 m / sin (50°) = 2.61m

Mixing Volume = 2.61 m (X) x 20 m (Y) x 5 m (Z) = 261m³ (i.e.13.1%)

Half of the mixing volume (referred to as the swap volume) is then swapped out of each parcel and into the adjacent parcel as indicated diagrammatically in Figure 15.11.

Figure 15.11: Swapping of Material within Mixing Zone (Orelogy 2025)

The grades and densities associated with each swap volume are carried into each parcel. This ensures the following:

- Diluent grades are accounted for, including both value driving and / or deleterious elements.
- Tonnes and metal balance are maintained on a block-by-block basis (i.e. no tonnes or metal are gained or lost globally).

The Bankan resource models (NEB/GBE and BC) had a parent block size of 10 m E by 20 m N by 5 m RL and a minimum sub-cell size of 5 m E by 10 m N by 0.625 m RL. An ore parcel model was created for the Bankan models based on a block size of 10 m E by 10 m N by 5 m RL (i.e. 500 m³).

The mixing widths were based on the selected digging equipment for the project, with 1.5 m assumed for the smaller (120 tonne) unit and 2.5 m for the larger (220 tonne) unit. It is assumed NEB/GBE ore will be excavated with a combination of the two sized units whereas BC, being smaller and more structurally complex, will only be mined with the smaller digging unit.

The mixing volume calculations are detailed in Table 15.9.

Table 15.9: Mixing Width Calculation by Deposit

Parameter		Unit	NEB/GBE	BC
120 t excavator	Mixing width	m	1.5	1.5
	% of ore mined	%	50%	100%
220 t excavator	Mixing width	m	2.5	2.5
	% of ore mined	%	50%	0%
Average Mixing Width		m	2.0	1.5
Dip		Degree	50	45
Strike		Degree	45	70
Apparent Mixing Width		m	2.8	3.0

Additional operations that are carried out as part of the block dilution calculations include:

- Blocks that are 100% ore are assumed to be within the orebody and have no “edge” and therefore have no dilution applied.
- Ore parcels where the blocks on either side are 100% waste is treated as an “isolated” block with two edges and the swapping volume is doubled.
- Where any parcel proportion is smaller than the required swap volume, they are insufficient in size to undertake the full “swap”. In this case the two proportions are accumulated together.

As a result of the approach described above:

- The degree of dilution varies on a block-by-block basis.
- All the diluent material carries some grade that also gets transferred between the parcels (i.e. the diluent material is not necessarily barren).
- As it is a volume swap, the degree of dilution or ore loss on a tonne basis can vary considerably if there is a significant density variation between ore and waste.

Therefore, to report a global ore loss and dilution can be problematic. Dilution and ore loss is therefore calculated on a block-by-block basis and then these values are treated as a grade item and the weighted average is then reported as the global number. As this is locational it varies from resource to pit shell to design depending on what blocks are being captured.

For the entire resource, the weighted average dilution and ore loss generated for the Project are presented in Table 15.10.

Table 15.10: Global Resource Ore Loss and Dilution (Weighted Average)

Model	Ore Loss	Dilution
NEB/GBE	4.3%	4.5%
BC	4.7%	4.9%

15.6.3 Optimisation Parameters

The open pit optimisation used as the basis for the pit designs utilised mining costs derived from a combination of initial mining contractor submissions and the PFS mining costs. A final validation optimisation was carried out using mining costs based on the preferred submission from the first-round request for budget pricing. The following sections detail the parameters and results of the validation optimisation that supports the pit designs and associated reserves.

15.6.3.1 Mining Dilution and Ore Loss

Ore loss and dilution were applied in the block model as described in Section 15.6.2. The optimisation used the diluted grades in the diluted mining model. Dilution was therefore set to 0% in Whittle. However, ore recovery in Whittle was set to 0.99 (i.e. 99%) to account for an additional 1% ore loss related to operational losses (i.e. material sent to wrong destination during mining operations).

15.6.3.2 Geotechnical Parameters

Preliminary pit wall design criteria were provided by Peter O'Bryan and Associates (Peter O'Bryan & Associates, 2025). These parameters were reviewed and not changed over the course of study and comprised:

- Face height.
- Face angle.
- Berm width.
- Geotech berm width at base of weathered profile.
- Stack berm width and stack height in fresh profile.

These parameters were used to calculate the overall slopes angle for the initial optimisation on which the pit design was based. The overall slope angles were then reviewed and modified on the basis of final feedback from Peter O'Bryan and Associates (Peter O'Bryan and Associates, 2025) as outlined in Section 15.4, and the final ramp layout of the designs. These were then utilised to generate a final optimisation to validate the designs produced. The slopes calculated for the final optimization of GBE/NEB and BC are detailed in Table 15.11.

15.6.3.3 Metallurgical Recoveries

Based on the metallurgical testwork, and as outlined in Section 13.6, PDI supplied the following regression for the calculation of recovery based on gold head grade:

$$\% \text{ Recovery} = (0.5145 \times [Au, g/t] + 91.533)/100$$

Table 15.11: Overall Slope Angle Calculation

Pit	Wall	Weath	Bearing[1]	Face Height	Face Angle	Berm Width	Inter-ramp Angle	Wall Height[2]	Geotech / Stack Berm Width	Two Way Road Width	One-Way Road Width	Overall Slope Angle
			(°)	(m)	(°)	(m)	(°)	(m)	(m)	(m)	(m)	(°)
NEB/GBE	Footwall (East)	Weath.	50	5	60	4.5	34.1	50	1 x 15			30.7
			135	5	60	4.5	34.1	50	1 x 15	1 x 29		24.7
		Fresh	95	10	90	12.0	39.8	205		2 x 29	1 x 19	36.7
	Hanging Wall (West)	Weath.	225	5	60	4.5	34.1	50	1 x 15			30.7
			320	5	60	4.5	34.1	50	1 x 15			30.7
		Fresh	275	20	75	8.0	56.3	205	2 x 15			55.1
BC	Footwall (East)	Weath.	30	5	60	4.5	34.1	50	1 x 15	1 x 29		28.0
			115	5	60	4.5	34.1	50	1 x 15			30.7
		Fresh	55	10	90	8.5	49.6	130			1 x 19	44.3
	Hanging Wall (West)	Weath.	205	5	60	4.5	34.1	40	1 x 15		1 x 19	25.4
			300	5	60	4.5	34.1	40	1 x 15		1 x 19	30.1
		Fresh	235	20	75	8.0	56.3	140	1 x 15		1.5 x 19	48.7

Notes:

1. Used in Whittle slope rosette set-up.
2. Estimate based on previous optimisations

15.6.3.4 Mining Costs

The mining costs were developed on a dollars per total tonne mined basis where applicable using the most up-to-date mining contractor cost submission at the time (May 2025). Fixed costs were converted into a unit cost using an initial LOM schedule to generate the total cost divided by a total tonnage. These are shown in Table 15.12 below.

Drill and blasts costs by material type are shown in Table 15.13 below. The costs for explosives were based on a submission from an in-country supplier.

Load and haul costs (inclusive of ancillary fleet) are shown by pit by bench level in Table 15.14 below. Fuel price was supplied by PDI based on in-country pricing.

Table 15.12: Fixed Costs ($/t mined)

Item	Cost ($/t mined)
Monthly Management Fees	$0.485
Rehab	$0.011
Infrastructure Works[1]	$0.124
TOTAL	**$0.621**

Note:

1. Clearing & grubbing, topsoil removal, roadbuilding

Table 15.13: Drill and Blast Costs ($/t mined)

Material	Density (t/m^3)	Rate ($/t)	PF (kg/m^3)	Explosives ($/t)	Presplit ($/t)	TOTAL ($/t mined)
Laterite	2.14	$0.758	0.45	$0.253		**$1.010**
Mottled	1.72	$0.364	0.23	$0.162		**$0.526**
Saprolite	2.27	$0.243	0.16	$0.082		**$0.324**
Saprock	1.63	$0.758	0.45	$0.332		**$1.089**
Fresh	2.80	$1.090	0.81	$0.345	$0.086	**$1.521**

The GBE mining costs used for the validation optimisation were based on the preferred mining contractor request for budget pricing submission at the time. However, the final approach used for the DFS financials was based on the mining of GBE pit being undertaken by the bulk earthworks (BEW) contractor engaged at the start of the construction period. The impact of using the mining contractor costs for the optimisation of this pit was considered immaterial as:

- The primary target of the GBE pit design was a fresh interface for establishing the underground portal. As such it was always assumed that it would be pushed beyond any optimal shell generated to achieve this.

- It's contribution to the Mineral Reserve and the overall project NPV is minimal. It is effectively a pre-production capital cost.

Table 15.14: Load and Haul (incl. Ancillary) by Bench by Pit ($/t mined)

RL	NEB	GBE	BC	RL	NEB	GBE	BC
10430	$2.45			10030	$4.39	$4.51	$4.39
10420	$2.62			10020	$4.50	$4.62	$4.50
10410	$2.86			10010	$4.61	$4.73	$4.61
10400	$3.02			10000	$4.72	$4.84	$4.72
10390	$3.32	$2.57		9990	$4.83	$4.96	$4.83
10380	$3.27	$2.88	$2.47	9980	$4.95	$5.07	$4.95
10370	$3.15	$3.42	$2.55	9970	$5.07	$5.19	$5.07
10360	$2.93	$3.45	$2.84	9960	$5.19	$5.32	$5.19
10350	$2.57	$3.34	$3.02	9950	$5.32	$5.44	$5.32
10340	$2.38	$2.66	$2.50	9940	$5.44	$5.57	$5.44
10330	$2.38	$2.50	$2.43	9930	$5.57	$5.70	$5.57
10320	$2.41	$2.54	$2.40	9920	$5.71	$5.83	$5.71
10310	$2.42	$2.54	$2.40	9910	$5.84	$5.97	$5.84
10300	$2.46	$2.58	$2.41	9900	$5.98	$6.10	$5.98
10290	$2.50	$2.62	$2.49	9890	$6.12	$6.24	$6.12
10280	$2.53	$2.66	$2.57	9880	$6.26	$6.38	$6.26
10270	$2.57	$2.70	$2.58	9870	$6.41	$6.53	$6.41
10260	$2.63	$2.75	$2.69	9860	$6.55	$6.68	$6.55
10250	$2.63	$2.80	$2.73	9850	$6.70	$6.83	$6.70
10240	$2.73	$2.85	$2.75	9840	$6.86	$6.98	$6.86
10230	$2.72	$2.91	$2.78	9830	$7.01	$7.13	$7.01
10220	$2.84	$2.96	$2.85	9820	$7.17	$7.29	$7.17
10210	$2.90	$3.02	$2.90	9810	$7.33	$7.45	$7.33
10200	$2.98	$3.08	$2.96	9800	$7.49	$7.61	$7.49
10190	$3.04	$3.15	$3.02	9790	$7.66	$7.78	$7.66
10180	$3.08	$3.21	$3.09	9780	$7.82	$7.95	$7.82
10170	$3.18	$3.28	$3.16	9770	$7.99	$8.12	$7.99
10160	$3.26	$3.35	$3.23	9760	$8.17	$8.29	$8.17
10150	$3.41	$3.43	$3.30	9750	$8.34	$8.46	$8.34
10140	$3.39	$3.50	$3.38	9740	$8.52	$8.64	$8.52
10130	$3.34	$3.58	$3.46	9730	$8.70	$8.82	$8.70
10120	$3.54	$3.67	$3.54	9720	$8.88	$9.00	$8.88

RL	NEB	GBE	BC	RL	NEB	GBE	BC
10110	$3.63	$3.75	$3.63	9710	$9.07	$9.19	$9.07
10100	$3.71	$3.84	$3.71	9700	$9.25	$9.38	$9.25
10090	$3.80	$3.93	$3.80	9690	$9.44	$9.57	$9.44
10080	$3.89	$4.02	$3.89	9680	$9.64	$9.76	$9.64
10070	$3.99	$4.11	$3.99	9670	$9.83	$9.96	$9.83
10060	$4.09	$4.21	$4.09	9660	$10.03	$10.15	$10.03
10050	$4.18	$4.31	$4.18	9650	$10.23	$10.35	$10.23
10040	$4.29	$4.41	$4.29	9640	$10.43	$10.56	$10.43

15.6.3.5 Ore Related Costs

The costs applied to material defined as ore by the optimisation process are summarised in Table 15.15.

Table 15.15: Ore Related Costs ($/t ore)

Item	Cost ($/t ore)
Mining	
Grade Control	$0.580
Assaying	$0.314
ROM Management & Rehandle	$1.209
Processing	
General and Administration	$0.231
Process and Maintenance Labour	$0.305
Reagents and Operating Consumables	$6.191
Fixed Power	$2.545
Variable Power – Laterite/Saprock	$2.700
Variable Power – Saprolite/Mottled	$1.160
Variable Power - Fresh	$6.140
Maintenance Consumables	$1.077
Tailings Management	$2.134
Tailings Rehandle	$3.39

Item	Cost ($/t ore)
General, Admin and NPI	
General & Administration	$2.598
Operating Consumables	$0.114
General & NPI Maintenance	$0.168
Power	$0.572
Labour	$0.559
G&A Sustaining	$0.044
Total Ore Cost	
Laterite/Saprock	**$24.732**
Saprolite/Mottled	**$23.192**
Fresh	**$28.172**

15.6.3.6 Revenues

The gold price and associated selling costs are detailed in Table 15.16.

Table 15.16: Revenue Parameters

Category	Unit	Value
Base Price	$/oz	$1,800
Government Royalty	% of Revenue	5.0%
Local Development Contribution	% of Revenue	1.0%
Refining Charge	$/oz	$4.00
Net Price	**$/oz**	**$1,688**
	$/g	**$54.27**

15.6.3.7 Capital Costs

Initial capital was not included in the optimisation setup as the breakpoint between the open pit and underground methods is unaffected by capital

15.6.4 Cut-Off Grade

The Mineral Reserve was reported using variable cut-off grades as processing costs vary by material type and metallurgical recovery varies by diluted head grade as outlined in Section 15.6.2.

For open pit mining, ore was flagged on a block-by-block basis within the model. Using the following relationships:

Breakeven Cut-off Grade (g/t) = Total Ore Cost / Net Price x Recovery

Breakeven Cut-off Grade (g/t) = Total Ore Cost / Net Price x (0.5145 x [Au grade] + 91.533)/100

The resulting breakeven cut-off regressions are:

- Laterite/Saprock, 0.49786 -0.00276 x (g/t).
- Saprolite/Mottled, 0.46686 -0.00259 x (g/t).
- Fresh, 0.56711 -0.00314 x (g/t).

Table 15.17 provides the average breakeven cut-off grade for open pit mining by material type based on the average mined head grade.

Table 15.17: Mineral Reserve Cut of Grade – Open Pit Mining

Material	Processing Cost (US$/t)	Head Grade (g/t)	Recovery (%)	Breakeven COG (g/t)
Laterite	$20.63	0.98	92.0%	0.41
Saprolite / Mottled	$19.09	1.03	92.1%	0.38
Saprock	$20.63	1.34	92.2%	0.41
Fresh – Open Pit	$24.07	1.50	92.3%	0.48

15.6.5 Open Pit Underground Transition

As the Project is proposed as a combined open pit and underground mining operation, it is important to evaluate the optimal transition between the open pit and underground designs.

The approach that Whittle uses is, for any given block that can be mined by either above-ground methods or underground methods, the value given to that block during open pit optimisation process is the difference between its value when mined above-ground and its value when mined underground. This is because the benefit of mining the block by above-ground methods is only its value over and above mining it from underground, as the block will still make the underground-based value if the block is omitted from the open pit.

It is important to note that this evaluation determines which blocks should be mined by open pit methods and which by the underground. However, when Whittle generates the value for the optimisation it uses the full block value from the open pit approach. Therefore, while Whittle will generate its normal range of nested shells at different revenue factors, the revenue factor 1 shell is essentially the “optimal” shell. A single underground mining cost of US$70 per tonne of ore was used as an initial estimate for the purpose of evaluation the open pit/underground transition point.

15.6.6 Optimisation Results

15.6.6.1 Open Pit Underground Transition Optimisation Results

The following Table 15.18 and Table 15.19 provide the outcomes of the combined open pit/underground validation optimisation as described above.

Table 15.18: Open Pit/Underground Transition Optimisation Results – Physicals

Shell	Revenue Factor	Ore							Waste	Total	Strip Ratio
		Weathered		Fresh		Total					
		(Mt)	(g/t)	(Mt)	(g/t)	(Mt)	(g/t)	(koz)	(Mt)	(Mt)	
36	1.00	10.8	1.05	34.3	1.52	45.1	1.41	2,043	80.4	125.5	1.78

Table 15.19: Open Pit/Underground Transition Optimisation Results – Financials

Shell	Revenue Factor	Costs			Revenue	Cashflow	Discounted Cashflow			Mine Life
		Mining	Ore	Selling			Best	Worst	Avg	
		($M)	($M)	($M)	($M)	($M)	($M)	($M)	($M)	(yr)
36	1.00	-$565	-$1,221	-$212	$3,407	$1,408.7	$1,102.9	$960.1	$1,031.5	10.0

Shell 36 (revenue factor 1) is the shell that is used to validate the pit designs which based on earlier optimisation results. The exact shape/depth of the open pit to underground transition shell will be dependent on the open pit and underground mining costs.

15.6.6.2 Sensitivity Analysis

In terms of the conventional optimisation sensitivity assessment to changes in cost, price, recovery etc., using the underground trade-off approach in Whittle effectively negates this sensitivity as the benefit or penalty from the parameter change is applied to both open pit and underground, and therefore the trade-off is not affected, and the transition point remains the same.

The only sensitivity that will affect the shell generated are changes to the mining costs. Table 15.20 shows the sensitivity of the shell to variations in open pit or underground mining costs. As can be seen the transition optimisation appear less sensitive to increases in open pit mining cost, than it is to a decrease. For a 10% increase in open pit mining cost, ore reduces by -3% and total pit size reduces by -5%. However, a 10% decrease in open pit mining cost increases ore by +5% and pit size by +11%.

Table 15.20: Open Pit/Underground Transition Optimisation Results – Sensitivity to Mining Cost

Scenario	OP Mining Cost	Ore							Waste	Total	Strip Ratio
		Weathered		Fresh		Total					
	% Var.	(Mt)	(g/t)	(Mt)	(g/t)	(Mt)	(g/t)	(koz)	(Mt)	(Mt)	
SCNP04A (Base Case)	0%	10.8	1.05	34.3	1.52	45.1	1.41	2043	80.4	125.5	1.78
SCNP04A-1	+10%	10.8	1.05	33.0	1.51	43.8	1.40	1,968	75.0	118.8	1.71
SCNP04A-2	-10%	10.8	1.05	36.4	1.57	47.2	1.45	2,203	92.3	139.5	1.95
% Var. from base Case											

Scenario	OP Mining Cost	Ore							Waste	Total	Strip Ratio
		Weathered		Fresh		Total					
	% Var.	(Mt)	(g/t)	(Mt)	(g/t)	(Mt)	(g/t)	(koz)	(Mt)	(Mt)	
SCNP04A-1	10%	0%	0%	-4%	-1%	-3%	-1%	-4%	-7%	-5%	-4%
SCNP04A-2	-10%	1%	0%	6%	3%	5%	3%	8%	15%	11%	10%

An optimisation was carried out on an open pit only basis using the validation optimisation parameters (i.e. without the underground cost trade-off) to determine the ultimate pit geometry. The results of this optimisation are presented in Table 15.21, Table 15.22 and Figure 15.12 below.

Table 15.21: Open Pit Only Optimisation Results – Physicals

Max. Best Case DCF (yellow) | Max. Worst Case DCF (green) | Max. Average Case DCF (pink)

Shell	Revenue Factor	Ore							Waste	Total	Strip Ratio
		Weathered		Fresh		Total					
		(Mt)	(g/t)	(Mt)	(g/t)	(Mt)	(g/t)	(Moz)	(Mt)	(Mt)	
21	0.7	8.4	1.12	27.2	1.94	35.6	1.75	1,998	91.7	127.3	2.58
22	0.72	8.5	1.12	27.4	1.93	35.9	1.74	2,009	92.2	128.1	2.57
23	0.74	8.7	1.12	27.6	1.92	36.3	1.73	2,020	92.4	128.7	2.55
24	0.76	8.9	1.11	29.3	1.90	38.2	1.71	2,103	100.1	138.2	2.62
25	0.78	9.1	1.10	30.2	1.89	39.4	1.70	2,158	105.9	145.3	2.69
26	0.8	9.3	1.09	30.6	1.89	39.9	1.70	2,186	109.7	149.6	2.75
27	0.82	9.6	1.09	31.6	1.86	41.2	1.68	2,230	112.7	153.9	2.74
28	0.84	9.7	1.08	32.5	1.84	42.3	1.66	2,260	113.6	155.9	2.69
29	0.86	10.0	1.07	32.9	1.83	42.8	1.66	2,280	115.5	158.3	2.70
30	0.88	10.2	1.07	36.0	1.84	46.2	1.67	2,478	149.2	195.4	3.23
31	0.9	10.3	1.06	38.5	1.84	48.8	1.68	2,632	175.7	224.5	3.60
32	0.92	10.6	1.05	41.2	1.78	51.9	1.63	2,720	180.9	232.8	3.49
33	0.94	10.7	1.05	43.3	1.78	54.1	1.63	2,839	201.6	255.7	3.73
34	0.96	10.8	1.05	43.9	1.77	54.6	1.63	2,855	202.9	257.5	3.71
35	0.98	10.8	1.05	44.4	1.76	55.2	1.62	2,871	203.9	259.1	3.69
36	1	10.8	1.05	45.8	1.77	56.6	1.63	2,971	226.5	283.0	4.00
37	1.02	10.8	1.05	46.1	1.76	56.9	1.63	2,979	226.9	283.8	3.99
38	1.04	10.9	1.05	48.3	1.77	59.2	1.64	3,120	257.5	316.8	4.35
39	1.06	10.9	1.04	48.8	1.76	59.7	1.63	3,132	258.3	318.0	4.33
40	1.08	10.9	1.04	48.9	1.76	59.9	1.63	3,136	258.6	318.5	4.32
41	1.1	11.0	1.04	49.7	1.76	60.6	1.63	3,170	265.5	326.1	4.38

Table 15.22: Open Pit Only Optimisation Results - Financials

Shell	Revenue Factor	Costs			Revenue	Cashflow	Discounted Cashflow			Mine Life
		Mining	Ore	Selling			Best	Worst	Average	
		($M)	($M)	($M)	($M)	($M)	($M)	($M)	($M)	(yr)
21	0.7	-$579	-$965	-$208	$3,342	$1,591.1	$1,232.9	$1,154.4	$1,193.6	7.91
22	0.72	-$582	-$973	-$209	$3,360	$1,591	$1,233	$1,154	$1,194	7.98
23	0.74	-$585	-$982	-$210	$3,378	$1,596	$1,236	$1,154	$1,195	8.06
24	0.76	-$630	-$1,034	-$219	$3,516	$1,600	$1,239	$1,153	$1,196	8.48
25	0.78	-$663	-$1,068	-$225	$3,609	$1,633	$1,261	$1,161	$1,211	8.75
26	0.8	-$684	-$1,082	-$228	$3,656	$1,654	$1,274	$1,164	$1,219	8.9
27	0.82	-$704	-$1,117	-$232	$3,729	$1,663	$1,280	$1,163	$1,222	9.2
28	0.84	-$713	-$1,146	-$235	$3,778	$1,677	$1,288	$1,158	$1,223	9.4
29	0.86	-$724	-$1,161	-$237	$3,812	$1,685	$1,293	$1,154	$1,223	9.5
30	0.88	-$902	-$1,254	-$258	$4,144	$1,689	$1,296	$1,152	$1,224	10.5
31	0.9	-$1,043	-$1,327	-$274	$4,403	$1,731	$1,319	$1,136	$1,227	11.5
32	0.92	-$1,080	-$1,412	-$283	$4,547	$1,759	$1,332	$1,120	$1,226	12.1
33	0.94	-$1,193	-$1,474	-$295	$4,748	$1,772	$1,338	$1,093	$1,216	12.8
34	0.96	-$1,201	-$1,489	-$297	$4,774	$1,786	$1,345	$1,074	$1,210	12.9
35	0.98	-$1,208	-$1,505	-$299	$4,800	$1,787	$1,346	$1,069	$1,207	12.9
36	1	-$1,327	-$1,543	-$309	$4,967	$1,788	$1,346	$1,063	$1,205	13.7
37	1.02	-$1,330	-$1,553	-$310	$4,981	$1,788	$1,346	$1,043	$1,195	13.7
38	1.04	-$1,494	-$1,618	-$325	$5,218	$1,788	$1,346	$1,041	$1,194	14.8
39	1.06	-$1,500	-$1,632	-$326	$5,238	$1,781	$1,342	$998	$1,170	14.9
40	1.08	-$1,503	-$1,636	-$326	$5,245	$1,780	$1,342	$992	$1,167	14.9
41	1.1	-$1,541	-$1,657	-$330	$5,302	$1,780	$1,341	$990	$1,166	15.1

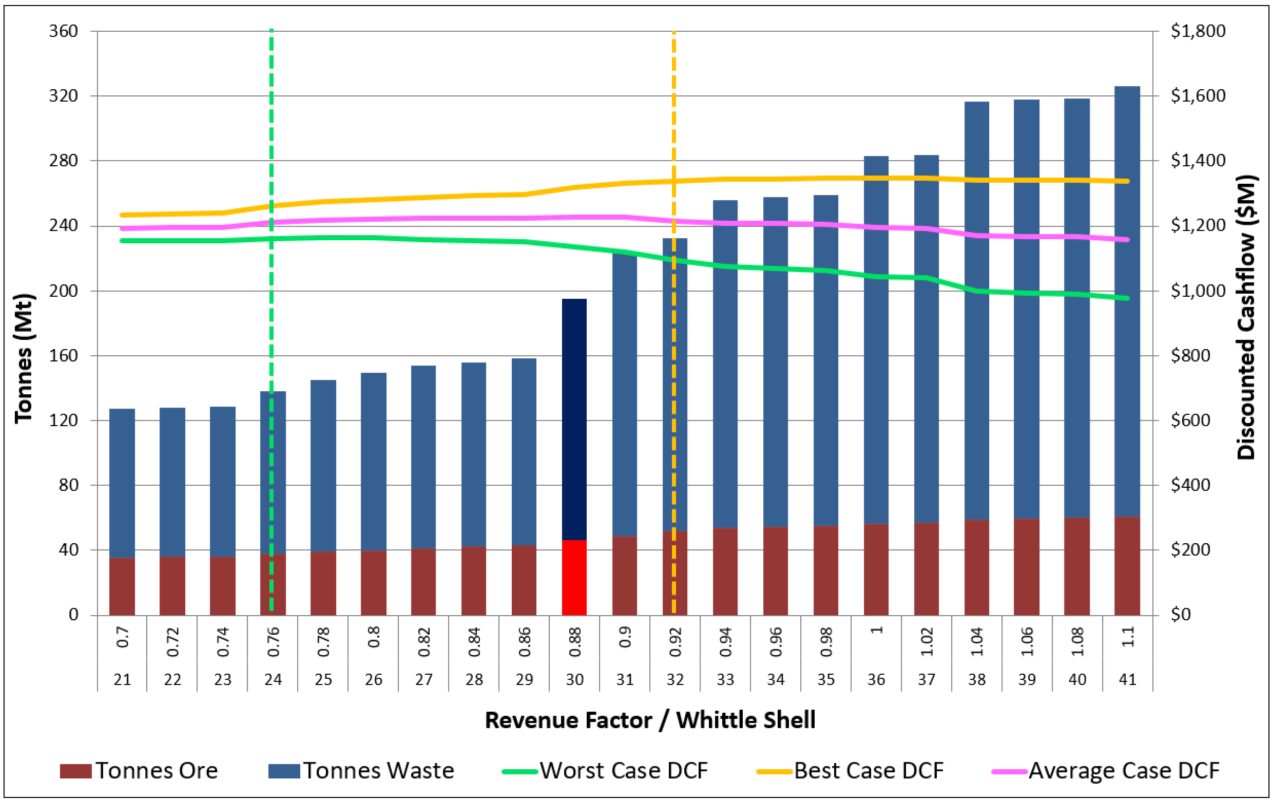

Figure 15.12: Open Pit Only Optimisation Results – Tonnes / Value Curves

The average case DCF shell can be considered a reasonable selection basis for a shell on which to base a "open pit only" Mineral Reserve design. It contains approximately the same ore tonnes as the open pit / underground transition shell 36 (within 2.5%) but at a much higher grade (1.67 g/t versus 1.41 g/t or an approximate increase of 18.5%) for considerably more waste (increase of approximately 85%). Visual comparison of the shell shows the "no underground" shell drives to the deeper high grade beneath the open pit/underground transition shell (approximately 80 m deeper) but excludes the lower grade material on the footwall that the open pit/underground transition shell recovers. In additional the GBE and BC pits are also smaller (reductions of 45% and 10% respectively). A plan and section view through the NEB shells are presented in Figure 15.13 which clearly shows where the unconstrained ore is sourced from, and the lower grade material excluded.

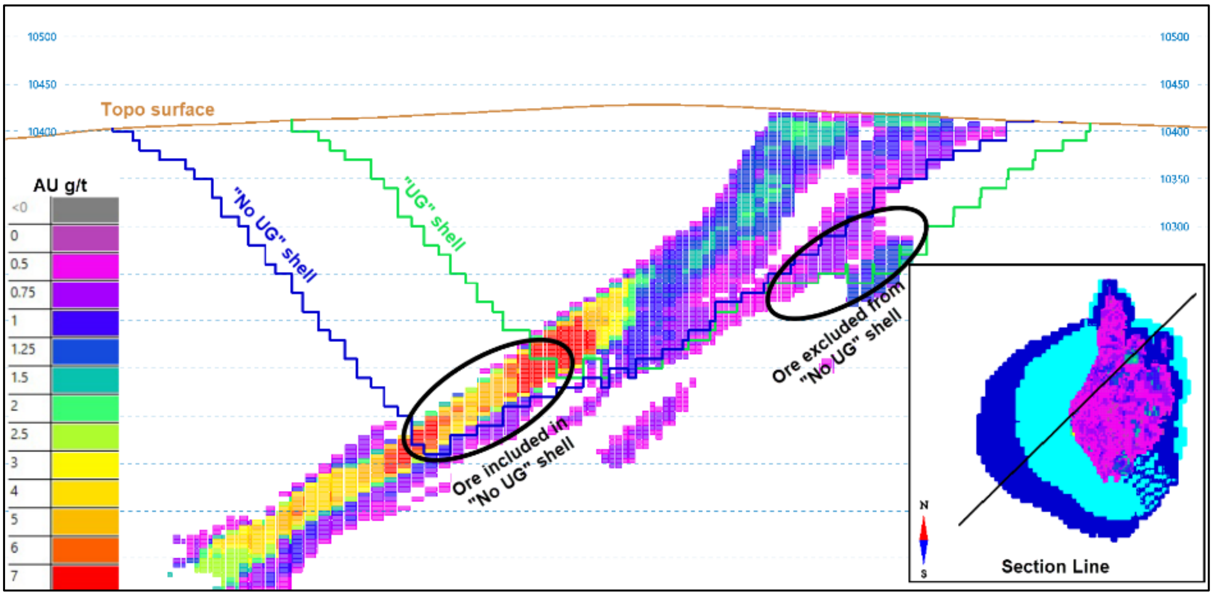

Figure 15.13: Plan and Section Comparing "No Underground" Shell 30 to "Open Pit/Underground Transition" Shell 36

15.7 Underground Optimisation

15.7.1 Introduction

The underground optimisation process was conducted using Deswik.SO, a strategic mine planning tool to automate the design of stope shapes for a range of stoping methods. The approach taken was to run a preliminary optimisation throughout the entire NEB resource model and range key inputs and parameters to conduct sensitivity analysis on the resultant inventories.

The algorithm of Deswik.SO replicates the workflow typically undertaken by a mining engineer. It commences by generating polylines on sectional views, which are then linked to form a three-dimensional wireframe, which is then evaluated against the block model. The outcome is an optimised stope shape that maximises recovered resource value above a defined cut-off grade.

The design process accounts for practical mining constraints, including minimum and maximum mining widths, anticipated wall dilution, minimum and maximum wall angles, and minimum separation distances between parallel or sub-parallel stopes. It also incorporates constraints on minimum and maximum stope heights and widths.

Deswik.SO uses the slice method, which optimises stoping by projecting stope shapes along the strike and height in the transverse direction of the orebody. Essentially, this approach slices the orebody perpendicular to its width, forming stope shapes that satisfy criteria such as cut-off grade and stopes geometry as shown in Figure 15.14.

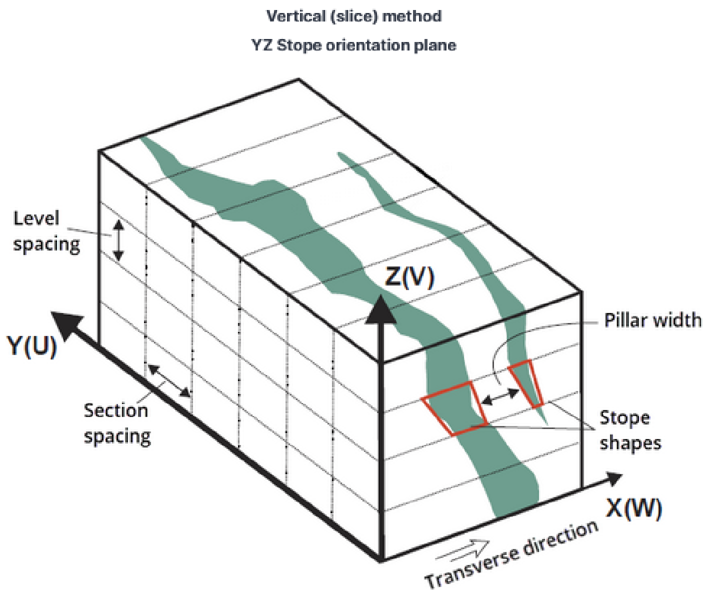

Figure 15.14: Vertical Slice Method Applied to a Vertical Orebody (Orelogy 2025)

As the NEB orebody strikes in the north-south direction (length, Y-axis) and is vertically oriented (height, Z-axis), the stope annealing process occurs along the east-west direction (width, X-axis). The NEB orebody geometry is shown in Figure 15.15.

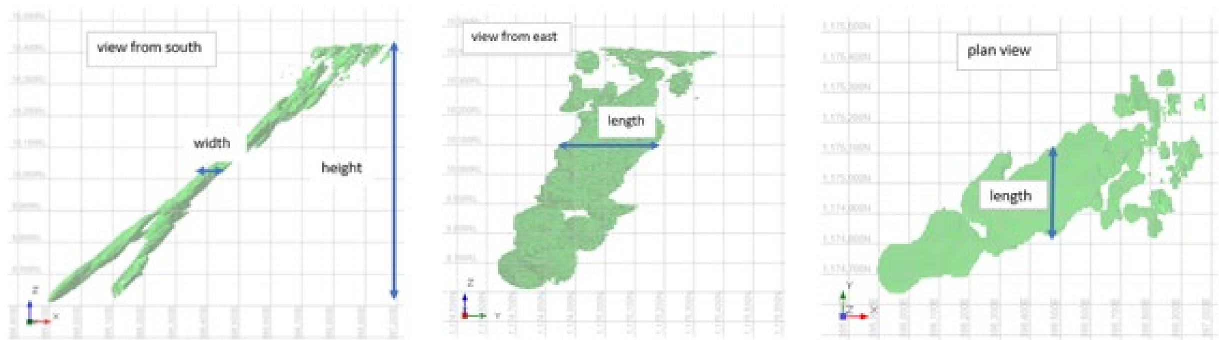

Figure 15.15: NEB Orebody Geometry

It should be noted that the optimisation and design of the underground was completed including consideration of potential future mining of the Inferred Mineral Resources, which is not included in the Mineral Reserve, for strategic planning purposes at the request of the Company. The Qualified Persons confirm that this approach has not impacted on the mining recovery of Mineral Reserves as the Inferred Mineral Resources are spatially separate and located below the Mineral Reserves.

15.7.2 Cut-Off Grade

The mining cut-off grade has been generated based on a combination of mining and non-mining costs calculated on first principal analysis and PDI economic constrains. These incorporated key operational parameters including the selected mining method, ground support requirements, stope dimensions, drive dimensions, as well as both fixed and variable costs. Establishment of this cut-off grade provided the base in which to run the stope optimisation process. The mining costs were calculated as detailed in Table 15.23.

Table 15.23: Mining Costs Calculations

Description	Units	Value
Nominal Ore Production	t/a	1,400,000
Panel Mining Cost (including Stoping & Ore Development)	US$/t	13.43
Haulage	US$/t	4.69
Filling (Paste)	US$/t	14.61
UG Contractor Overheads	US$/t	26.22
UG Owner Power	US$/t	8.25
UG Owner Fuel – Contractor	US$/t	2.07
UG Tech Services	US$/t	2.80
Accommodation, Messing & Flights	US$/t	1.40
Geology/Grade Control	US$/t	1.00
Total Unit UG Mining Cost	**US$/t**	**74.47**

Merging mining costs and business parameters the cut-off grade was calculated to be 2.0g/t. Table 15.24 summarises the input values and assumptions used in the calculation.

Table 15.24: Cut-Off Grade Inputs

Parameter	Units	Value
Maximum Ore Production	t/a	1,400,000
Gold Recovery	%	92.6
Discount Rate	%	5
Gold Price	US$/oz	1,800
Government Royalty	%	6
Selling Cost	US$/oz	4.00
Total Selling Price	US$/oz	111.76
Net Gold Price	US$/oz	1,688.24
	US$/g	54.28
Grade Control	US$/t	incl.
G&A Cost	US$/t	2.28
Processing Cost	US$/t	22.34
Underground Mining Cost	US$/t	74.47
Underground Cut-off grade	**g/t**	**1.97**

Sensitivity analysis of the cut-off grade was conducted by varying the following four key parameters by 80% to 120% (5% increments):

- Gold price.
- Recovery.
- Mining costs.
- Processing costs.

The analysis indicates that the cut-off grade varies within a range of 1.64 g/t to 2.46 g/t as shown in Table 15.25 and Figure 15.16.

Table 15.25: Underground Cut-off Grade Sensitivity

Variation	Gold Price	Recovery	Mining Cost	Processing Cost	Production Rate	G&A Cost
80%	2.47	2.46	1.68	1.88	2.34	1.96
85%	2.32	2.32	1.75	1.90	2.23	1.96
90%	2.19	2.19	1.82	1.93	2.14	1.97
95%	2.08	2.08	1.90	1.95	2.05	1.97
100%	**1.97**	**1.97**	**1.97**	**1.97**	**1.97**	**1.97**
105%	1.88	1.88	2.05	1.99	1.90	1.97
110%	1.79	1.79	2.12	2.02	1.84	1.98
115%	1.71	1.71	2.19	2.04	1.78	1.98
120%	1.64	1.64	2.27	2.06	1.72	1.98

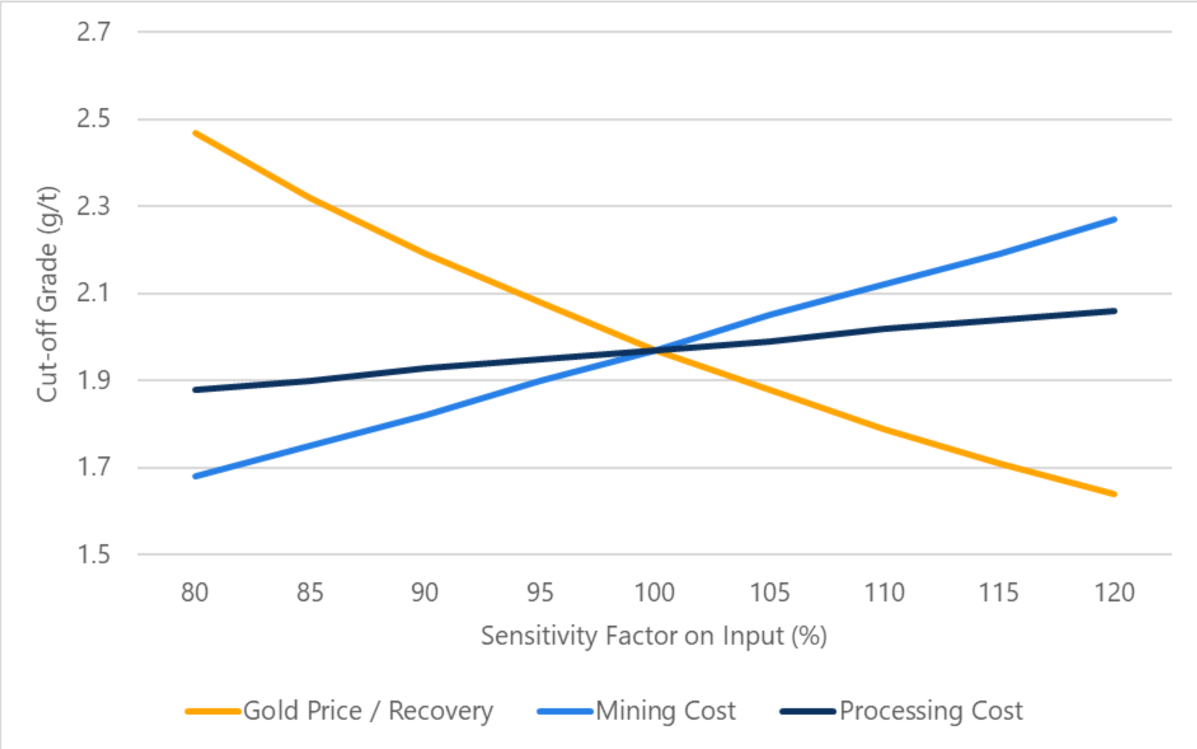

Figure 15.16: Cut-off Grade Sensitivity Analysis

15.7.3 Optimisation Parameters

Optimisation parameters follow the orebody geometry and geotechnical guidelines, which are variable according to depth of the orebody and cut-off calculated. Following these characteristics resulted in the optimisation parameters summarised in Table 15.26.

Table 15.26: Stope Optimisation Input Parameters

Level intervals	Parameter	Unit	Value
	Mining method		Transverse, LH stoping with paste fill
	Cut-off grade	g/t	2.00
	Rotated model	yes/no	No
	Rotated framework	yes/no	No
	Framework orientation	method	Slice (YZ Plane)
10160 mRL to 10020 mRL	X	origin/distance/rotation	395850 / 1200 / 0
	Y	origin/distance/rotation	1174600 / 900 / 0
	Z	origin/distance/rotation	10020 / 140 / 0
10020 mRL to 9880 mRL	X	origin/distance/rotation	395850 / 1200 / 0
	Y	origin/distance/rotation	1174600 / 900 / 0
	Z	origin/distance/rotation	9880 / 140 / 0
9880 mRL to 9740 mRL	X	origin/distance/rotation	395850 / 1200 / 0

Level intervals	Parameter	Unit	Value
	Y	origin/distance/rotation	1174600 / 900 / 0
	Z	origin/distance/rotation	9740 / 140 / 0
9740 mRL to 9500 mRL	X	origin/distance/rotation	395850 / 1200 / 0
	Y	origin/distance/rotation	1174600 / 900 / 0
	Z	origin/distance/rotation	9500 / 240 / 0
10160 mRL to 10020 mRL	Strike length (U)	metre	20
10020 mRL to 9880 mRL			15
9880 mRL to 9740 mRL			15
9740 mRL to 9500 mRL			12
	Level height (V)	metre	20
	Slice interval	metre	NA
	Stope width	metres (minimum/maximum)	5 / 100
	Stope dilution	metres (HW/FW)	0.5 / 0.5
	Minimum pillar	metres	30
	Dip angles	° (minimum/maximum/change)	40 / 70 / 2
	Strike angles	° (minimum/maximum/change)	-0.2
	Stope thickness ratio	ratio (tb / lr)	2.4 / 2.4
	Sub shapes	yes/no	No
	Vertical refinement	number of points	NA
	Material exclusions	yes/no	NA

15.7.3.1 Grade Sensitivity Analysis

By varying the cut-off grades by 0.1 g/t from 1.2 g/t to 2.2 g/t the grade variation was analysed to determine the sensitivity of the orebody, as shown in Figure 15.17.

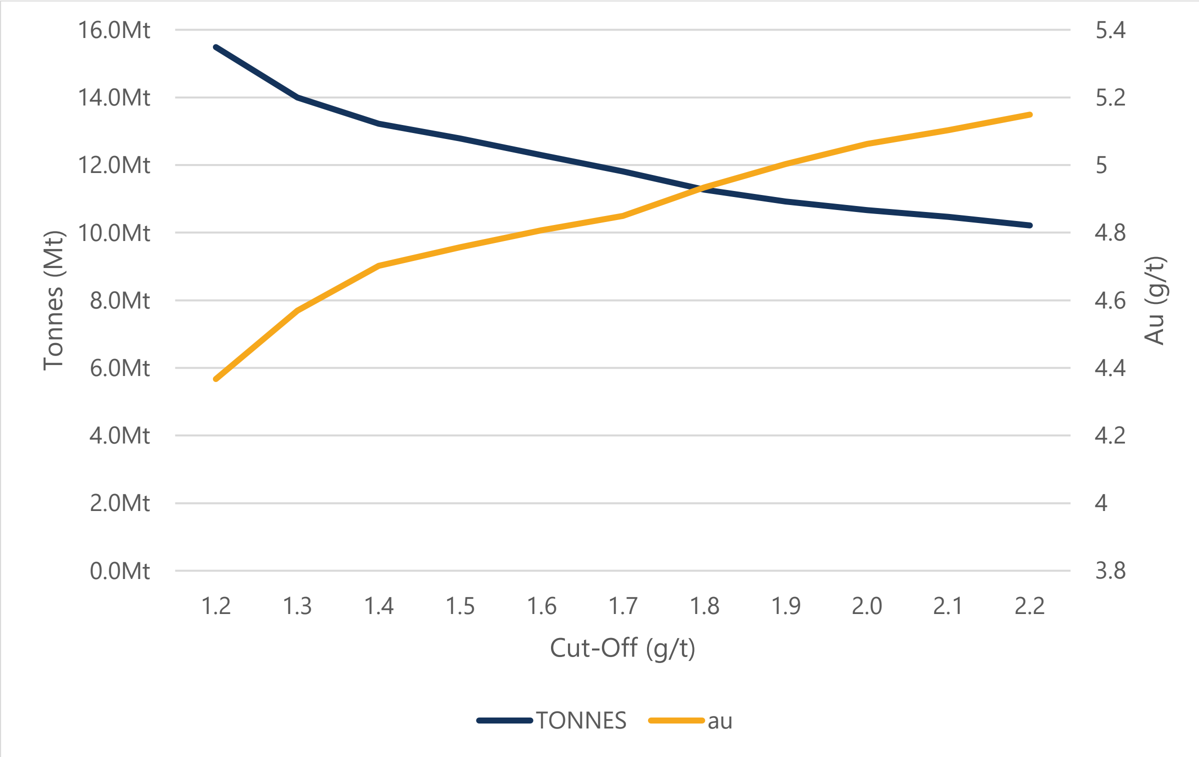

Figure 15.17: Underground Grade and Tonnage versus Cut-off Grade

An inverse relationship between grade and tonnage was observed, with an overall variance of 34% in tonnes, corresponding to a 13% variance in contained ounces. To better understand the source of the variance, the analysis was broken down by resource classification, as shown in Figure 15.18 and Figure 15.19.

Tonnes in the Indicated section show low sensitivity to cut-off grade changes, varying only 7% above 1.8 g/t, compared to 16% in the Inferred material. In terms of contained ounces, the same range results in a 3% variation for Indicated and 7% for Inferred.

The sensitivity analysis confirms that the selected cut-off grade of 2.0 g/t is technically and economically justified. The Indicated portion of the orebody demonstrates limited sensitivity to cut-off grade variations, with both tonnes and contained metal showing stable responses. This indicates a robust and well-defined resource that supports reliable mine planning.

While the Inferred resource, which was only investigated for strategic planning purposes, displays greater variability, particularly in tonnes, it remains within acceptable ranges for evaluation. The analysis further highlights the lower grade nature of the Inferred material and its higher sensitivity to economic inputs, reinforcing the need for further drilling and conversion to higher confidence classification prior to incorporation within production schedules.

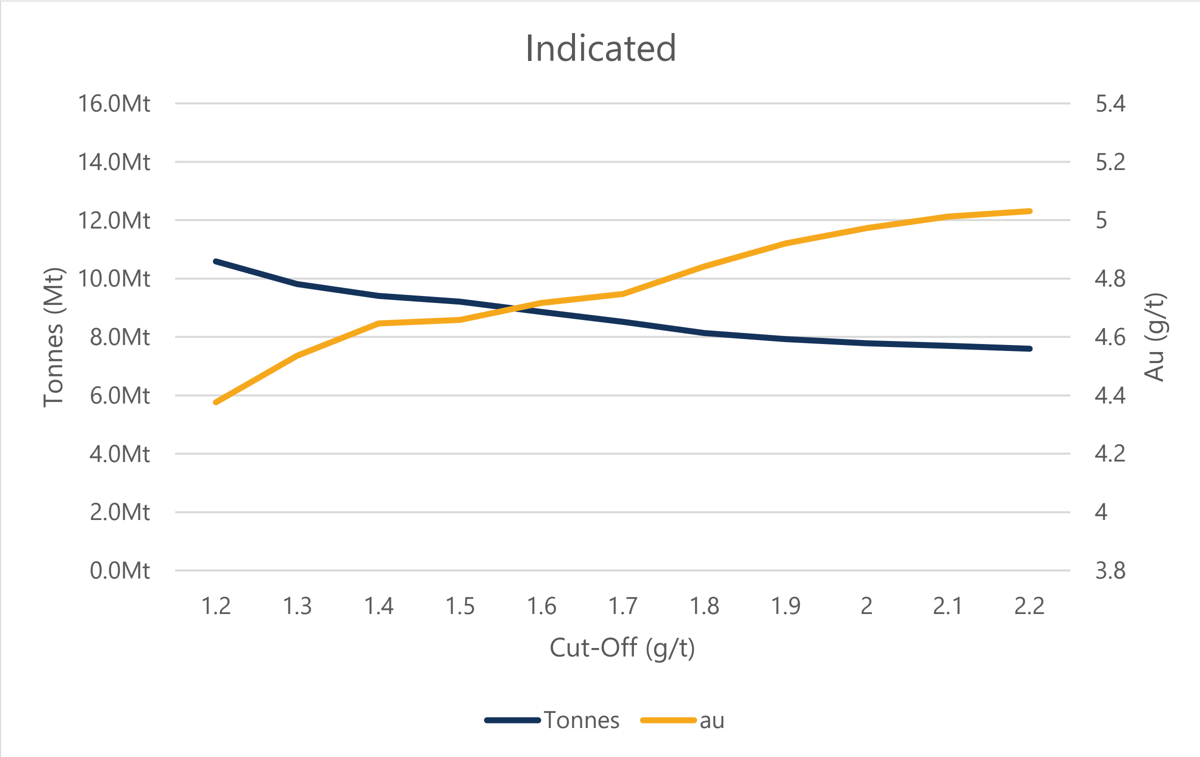

Figure 15.18: Indicated Mineral Resources Tonnes and Grade versus Cut-off Grade

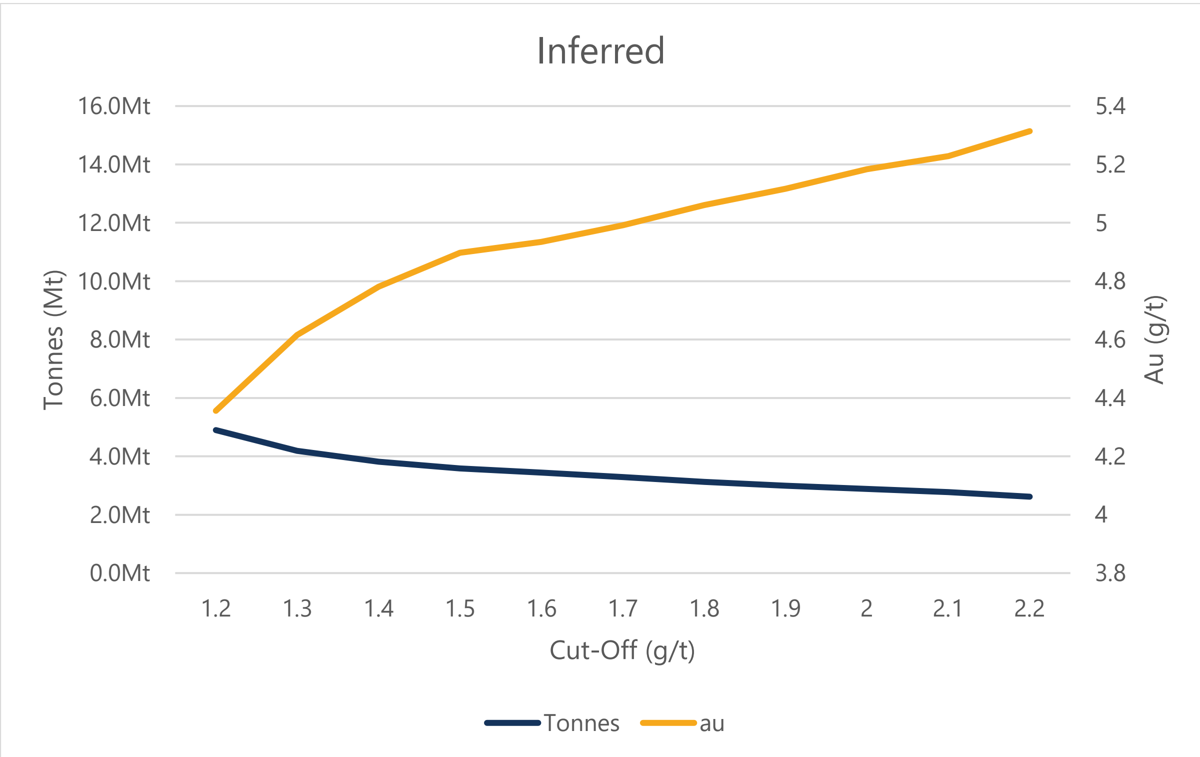

Figure 15.19: Inferred Mineral Resources Tonnes and Grade versus Cut-off Grade

15.7.3.2 Stope Configuration Trade-Off

As part of the mining method selection and stope optimisation process, a trade-off analysis was conducted to evaluate the performance of two transverse stope configurations.

- Single lift 20 m height.
- Double lift 40 m height.

The objective was to determine the most suitable approach based on geotechnical constraints, operational efficiency, and overall project economics.

The analysis indicated that the double lift configuration would have an approximately 10% higher mining rate than the single lift, primarily due to the ability to conduct concurrent mining activities, despite slower downhole drilling rates. In both configurations, bogging and backfilling remained the most time-intensive components of the stope cycle. From a cost perspective, the two configurations are broadly comparable; however, the double lift demonstrates a marginal efficiency advantage by reducing the number of ore drives required within a 60 m panel. To achieve the target production rate of 1.4 Mtpa, both configurations necessitate three active stopes and three boggers, resulting in equivalent equipment requirements. Figure 15.20 and Figure 15.21 below depicts the analysed scenarios.

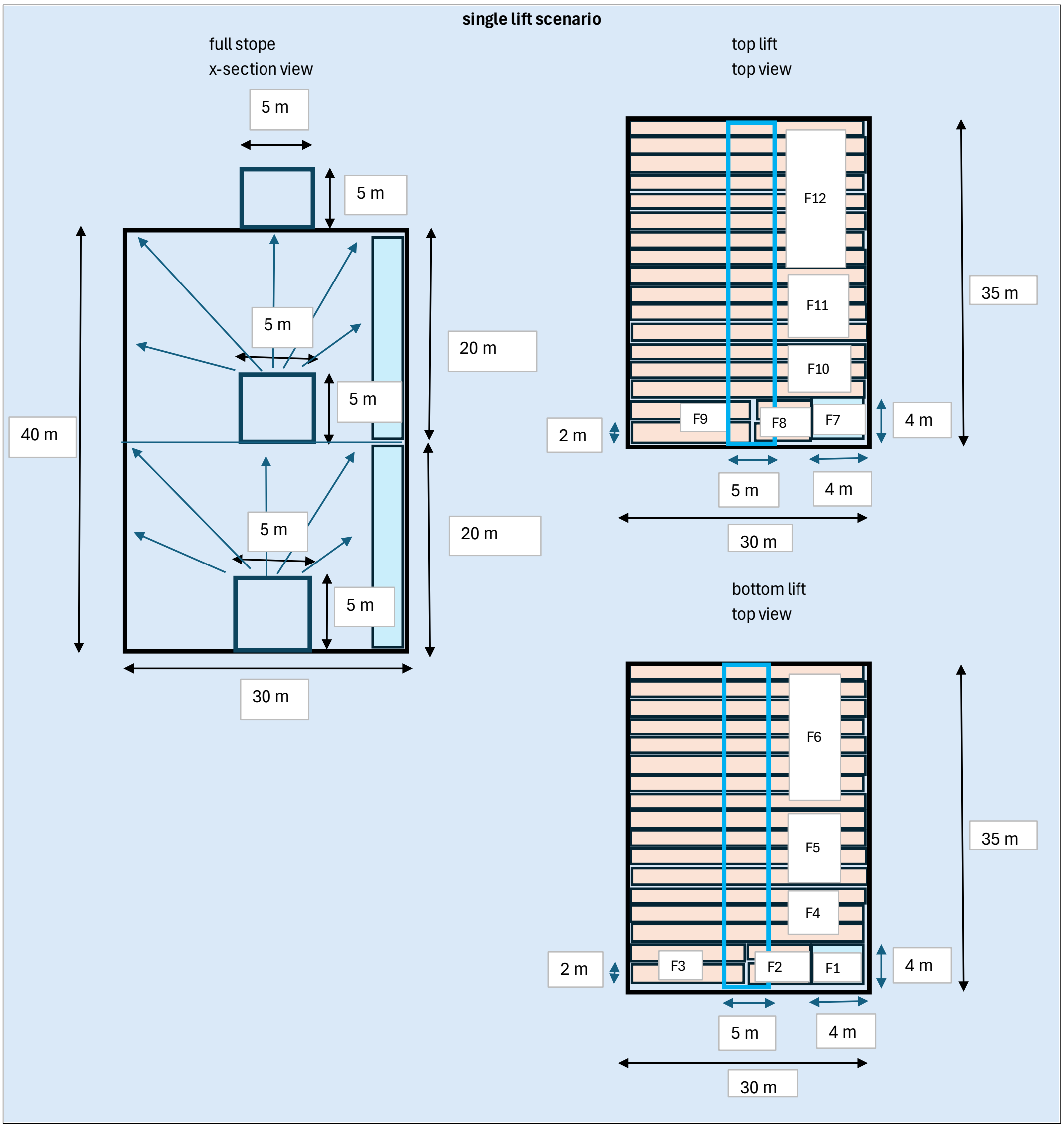

Figure 15.20: Single Lift Scenario (Orelogy 2025)

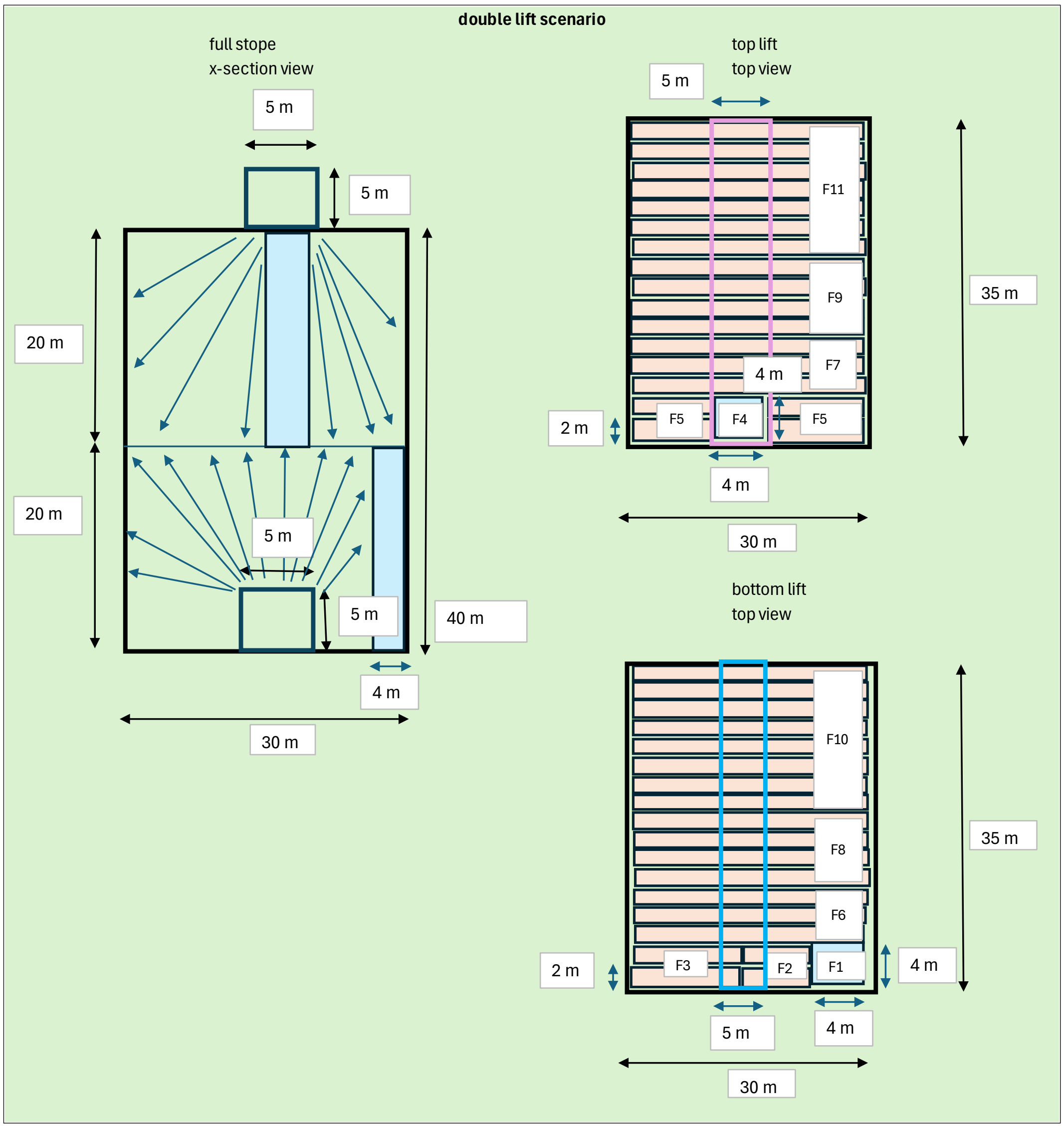

Figure 15.21: Double Lift Scenario (Orelogy 2025)

However, while the double lift configuration offers time and cost efficiencies by reducing development, it exceeds the recommended one-month hangingwall exposure, reaching approximately 1.5 months. This increases geotechnical risk and may breach design standards. In contrast, the single lift configuration, using 20-metre stopes, stays within exposure limits, ensuring better stability and compliance. While it requires more ore drives and slightly longer mining time, it provides a safer and more geotechnically sound option.

Another key advantage of the single lift approach lies in the inherent flexibility of the orebody, which permits the subdivision of stopes into smaller, more manageable panels without compromising ore accessibility or overall productivity. This adaptability enables more precise control over stope sequencing and contributes significantly to maintaining ground stability. By leveraging this flexibility, the operation can reduce exposure times, mitigate geotechnical risks, and support a safer and more sustainable production environment, particularly critical in areas with variable ground conditions or during production scale-up.

Overall, the single lift method presents a more robust and geotechnically resilient solution, aligning with long-term mine planning objectives and safe execution standards and was therefore selected for the underground optimisation and mining method.

15.7.4 Optimisation Results

Optimisation output at 2.0 g/t cut-off grade generated an inventory of 7.9 Mt at 3.95 g/t for 1,002 koz based on Indicated Mineral Resources only.

There is a clear boundary between Inferred and Indicated material, resulting from the drilling density and orebody continuity. This is shown in Figure 15.22 below with Indicated material in green and Inferred material in red.

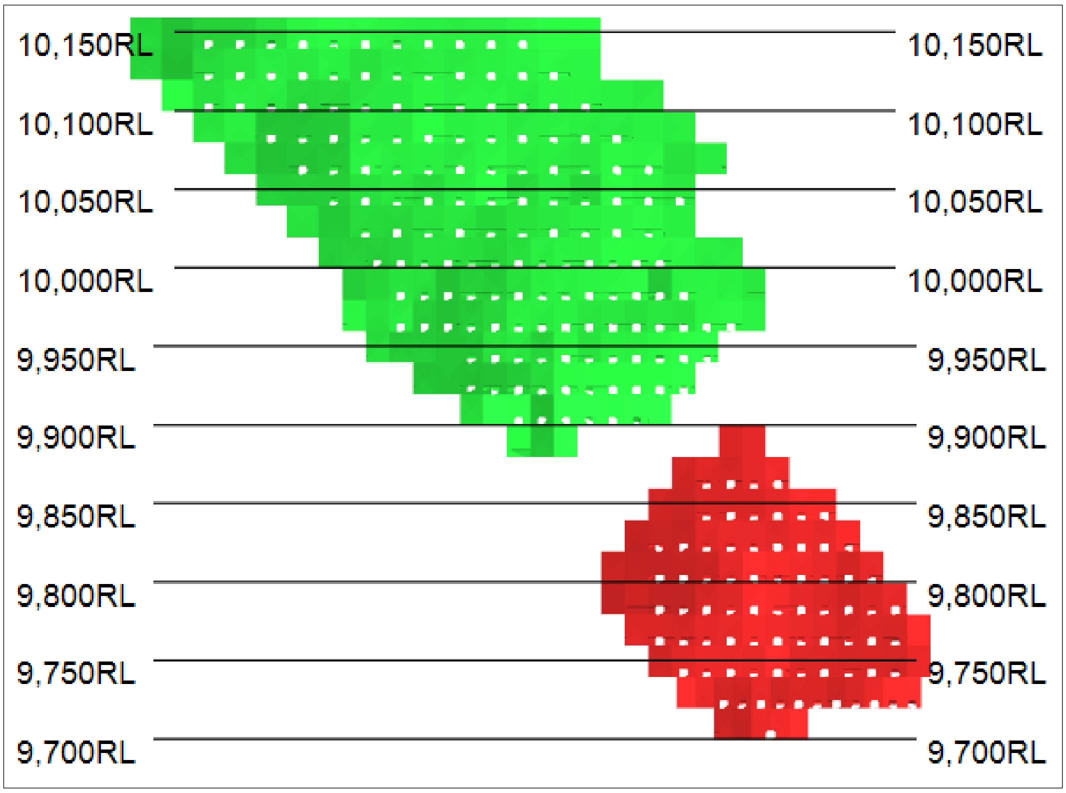

Figure 15.22: Stope Shape Results Looking East

It is worth noting that while the optimisation process included Inferred Mineral Resources, this was only used to for strategic planning purposes and because of the clear boundary between Indicated Mineral Resources and Inferred Mineral Resources, the inclusion of the Inferred Mineral Resources did not influence the economic viability of the Indicated Mineral Resources included in the stope optimisation. No Inferred Mineral Resources have been included in the Mineral Reserves or LOM production schedule.

16 MINING METHODS

16.1 Overall Mining Strategy

An assessment of several project development strategies was carried out prior to the DFS commencing and this informed the final approach for the DFS. The assessment results indicated that bringing the underground operation online as early as possible generated considerable benefit to the project as it provided:

- Higher grade plant feed from the start of production.
- Fresh ore feed to the comminution circuit, which reduces the complexity of designing a circuit to accommodate both 100% weathered ore and 100% fresh ore.

As such, the underground needs to be established during the pre-production period to ensure continuity of fresh ore feed from underground once plant production commences. The project strategy developed includes open pit mining from NEB, BC and GBE.

The GBE pit is mined during the first nine months of the two-year pre-production period. This pit is specifically designed to target the fresh rock interface adjacent to the NEB orebody, from where the underground access portal will be established. This has the key benefits of:

- Mitigating the risks of establishing the underground decline access through the geotechnically challenging weathered clay zones (approximately 60 vertical metres).
- Generating ore as part of the development process, which can subsequently be used as part of the plant commissioning stocks.
- Providing bulk fill material for construction purposes.

Underground development then progresses from Month 11 of Year 1 of pre-production, targeting the delivery of a minimum of 25% sustainable fresh ore feed to the plant from the start of production. Pre-stripping of NEB begins in the second half of Year 2 of pre-production to ensure sustainable ore feed is available from the start of production. During this period GBE waste material is rehandled to construct the ROM pad ready for ore delivery.

Mining of BC is deferred until the end of the mine life due to the added cost and complexity of establishing surface water management, including the diversion of Bankan Creek, around this pit.

There is a considerable delay between the completion of the GBE pit and the commencement of pre-strip mining of the NEB pit (approximately nine months), therefore it was not considered cost effective to mobilise the mining contractor for the mining of GBE to then either demobilise or have equipment on standby for the nine months until production mining in NEB commenced. Therefore, the DFS approach assumes that the bulk earthworks contractor engaged for the construction component of the project would also be utilised to mine GBE pit.

16.2 Mining Method Selection

16.2.1 Open Pit Mining

The selected open pit mining method for the Project is using a conventional truck and shovel approach, which is a proven mining method for open pit mining in West Africa. The operation will follow a typical cycle of operation as shown in Figure 16.1.

Mining areas will be cleared of vegetation and topsoil removed and stockpiled for use in rehabilitation of the site. Grade control will be carried out in advance of mining utilising RC drilling. The mining method and grade control practises to be employed at site are aimed at mining the ore zones selectively using backhoe configured excavators on a 2.5 m flitch to minimise dilution and ore loss.

It has been assumed that all material will require drill and blast, except for the saprolite clay and mottled clay material which has been assumed as 75% and 50% free dig respectively, however, this material may require some dozer ripping and equipment has been incorporated into the mining study to accommodate this.

Continuous open pit mining operations will commence during the last six months of the construction period in the NEB Stage 1 pit. Construction of ex-pit mine roads, ROM pad, TSF starter embankment and other infrastructure will be carried out using stockpiled GBE waste and run-of-mine waste from NEB. During production, waste will be hauled to the nearest waste rock dump, other than periodic requirements for bulk fill for the TSF embankments and road construction as required.

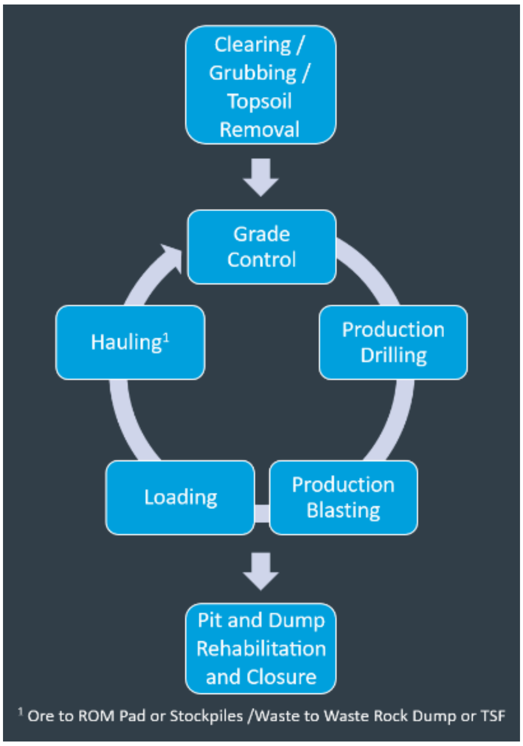

Figure 16.1: Open Pit Mining Cycle (Orelogy 2025)

Ore delivered to the ROM pad will be stored in stockpile 'fingers' from where it will be rehandled to the crusher using a front-end loader (FEL). Low-grade ore will be placed in long-term stockpiles adjacent to the pits from where it will be rehandled to the ROM pad using an FEL/truck combination and directly loaded into the crusher dump pockets.

Rehabilitation of the waste rock dumps will commence as soon as areas of the dumps are completed to final limits.

16.2.2 Underground Mining

Underground mining for the Project will employ proven, industry-standard equipment and methods, including twin-boom jumbos, long hole drills, load-haul-dump units (LHDs), and haul trucks, ensuring safe and reliable operation.

Figure 16.2 below shows the location of the underground workings relative to the portal location in the GBE pit and the NEB open pit, looking east. Figure 16.2 shows the inferred material below 9,900RL despite this material being excluded from the Mineral Reserve and LOM production schedule with it only being included in the underground optimisation and mine design for strategic planning purposes.

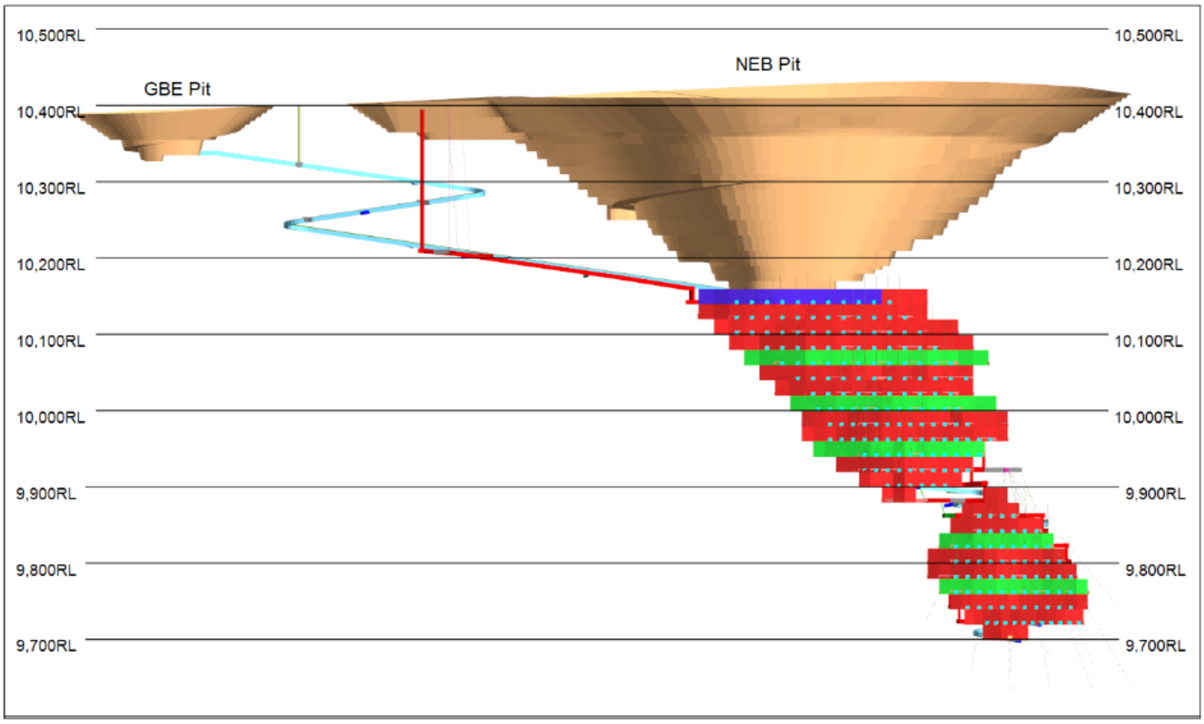

Figure 16.2: Access to Underground from the GBE Open Pit Looking East

The underground mining method was selected based on a comprehensive evaluation of geological, geotechnical, operational, and economic factors to optimise ore recovery and ensure sustainable operations. The key considerations included:

- Cost-effective bulk mining. Employing bulk mining techniques, such as up-hole transverse and longitudinal long hole open stoping, enabling efficient extraction of large ore volumes, reducing unit costs and effectively managing short-term grade variability.
- Operational flexibility. Designing multiple mining areas to enable simultaneous extraction activities across different locations, ensuring the achievement of minimum annual production rates and mitigating the risk of production bottlenecks.
- Enhanced ore recovery with paste backfill. Utilising paste backfill technology to improve ground stability and maximise ore recovery, particularly in high-grade zones.

This integrated approach ensured economic efficiency, operational safety, and environmental responsibility in the underground mining operations.

Mining selection underwent a rigorous analysis process of the orebody, considering geotechnical qualification of the ore and surrounding rocks, dip angles, depth and thickness. Using the UBC Mining Method Selector (Miller-Tait, Pakalnis, & Poulin, 1995), sub-level stoping was identified as the most suitable mining method for the orebody.

Due to the required underground production rate, the internal dilution and hanging wall exposure, two variations of the sub-level stopping method were investigated and chosen, transverse long hole open stoping (TLHOS) and longitudinal long hole open stoping (LLHOS).

16.2.2.1 Transverse Long Hole Open Stoping

Transverse long hole open stoping is a mining method in which the stope is oriented perpendicular to the strike of the orebody, with the mining direction being from the hanging wall toward the footwall. This approach enables larger stope dimensions, as the hanging wall exposure remains constant.

The method is best suited to wider sections of the orebody, however for the Project, a stope width limit was applied based on geotechnical constraints.

Due to the potential for larger stope sizes, this method can offer higher productivity, with reduced drilling requirements and a faster mining cycle. However, it requires greater development, as each stope must have individual access and a larger number of long hole slots and raises due to the geotechnical constraints placed on the stope dimensions.

16.2.2.2 Longitudinal Long hole Open Stoping

Longitudinal long hole open stoping is a mining method in which stopes are developed along the strike of the orebody, parallel with the hanging wall and foot wall drives. This method was selected in areas where the orebody thickness was insufficient to support a transverse approach, and where greater control was required to minimise dilution along the footwall. While this method offers improved selectivity, it is generally associated with smaller stope dimensions and higher drilling densities, which result in longer stope cycles and increased operational effort.

16.2.2.3 Mine Backfill

Given the high-grade nature of the orebody, maximising resource recovery was a critical objective of the mine design. This necessitated minimising ore loss and ensuring the complete extraction of stopes. The use of backfill was identified as a key enabler to achieve this goal.

Backfill methods can vary and include rock fill, cemented rock fill, and engineered fill such as paste. Rock fill is typically employed in operations where haulage constraints exist and where void confinement is not required for pillar recovery. However, in this case, where preserving pillars for stability would compromise ore recovery, engineered backfill was considered essential.

PDI engaged MineFill (Minefill Services, 2025) (Minefill Services, 2025) to conduct an assessment of engineered fill options, with a particular focus on paste fill. Several test programs and analyses were undertaken to evaluate the suitability of tailings to produce a consistent and reliable paste backfill material. These results are summarised in Section 13.14.

The adoption of paste fill also had implications for the mining sequence. As exposure of cured paste within stopes can lead to dilution if not managed correctly, so the mine plan was created to enable implementation a bottom-up extraction sequence. This sequencing ensured both geotechnical stability and optimal ore recovery while minimising the potential for backfill-induced dilution.

16.3 Mine Design Basis and Optimisation

The design basis relating to the Mineral Resource Model, geotechnical considerations, groundwater and the optimisation of the open pit mining and underground mining are outlined in Sections 15.3 to 15.7.

16.4 Open Pit Mine Design

The open pit designs aim to optimise resource extraction by using the selected optimisation shell as a guide, while also ensuring safety, efficiency, and environmental sustainability. The following sections outlines the key aspects of the open pit mine design criteria, pit and dump designs, pit inventories and water management design.

16.4.1 Design Criteria

16.4.1.1 Geotechnical Parameters

The wall slope design parameters used are detailed in Section 15.4.2 and summarised in Table 16.1 below for clarity.

Table 16.1: Pit Wall Design Criteria

Wall	Weathering	Face Height	Face Angle	Berm Width	Geotech Berm[1]	Stack Berm[2]
		(m)	(°)	(m)	(m)	(m)
Footwall (East)	Weathered	5	60	4.5	15	N/A
	Fresh	10	90	12.0	N/A	N/A
Hanging Wall (West)	Weathered	5	60	4.5	15	N/A
	Fresh	20	75	8.0	N/A	15

Notes:

1. At fresh material boundary.
2. Every vertical 80 metres.

16.4.1.2 Pit Access

When the pit designs were being undertaken it was assumed the contractors would most likely select a dump truck in the 100 to 150 tonne class range and therefore, to ensure the pit designs could cater for any possible final equipment selected, the roads were design for the 150 tonne class truck, specifically the Caterpillar 785G.

The calculation of the road widths for in-pit ramps and ex-pit roads are presented in Table 16.2 below. The road width calculation is based on the following parameters:

- Truck type, Caterpillar 785G.

- Truck operating width (TOW), 6.71 m.
- Truck tyre type, 33R51.
- Tyre outside diameter (OTD), 3.06 m.
- Safety bund height, bund height = ½ OTD, 1.53 m.
- Top width of bund, 0.1 m.
- Bund width (BW) at a slope of 37.5°, 4.1 m.
- Drain width (DW), 1.5 m.

Table 16.2: Design Ramp and Roads Widths

Road Type	Clearance		Total Multiplier	Running Width	Total Width
	Between Trucks	Edge			
	(# Trucks)	(# Trucks)	(# Trucks)	(m)	(m)
Dual Lane Ramp	1.00	0.25	3.5	23.50	29.0
Single Lane Ramp	N/A	0.50	2.0	13.40	19.0
Dual Lane Surface road[1]	1.00	0.25	3.5	23.50	31.5

Notes:

1. Double bunded.

Typical schematics of the road layouts are provided in Figure 16.3 to Figure 16.5.

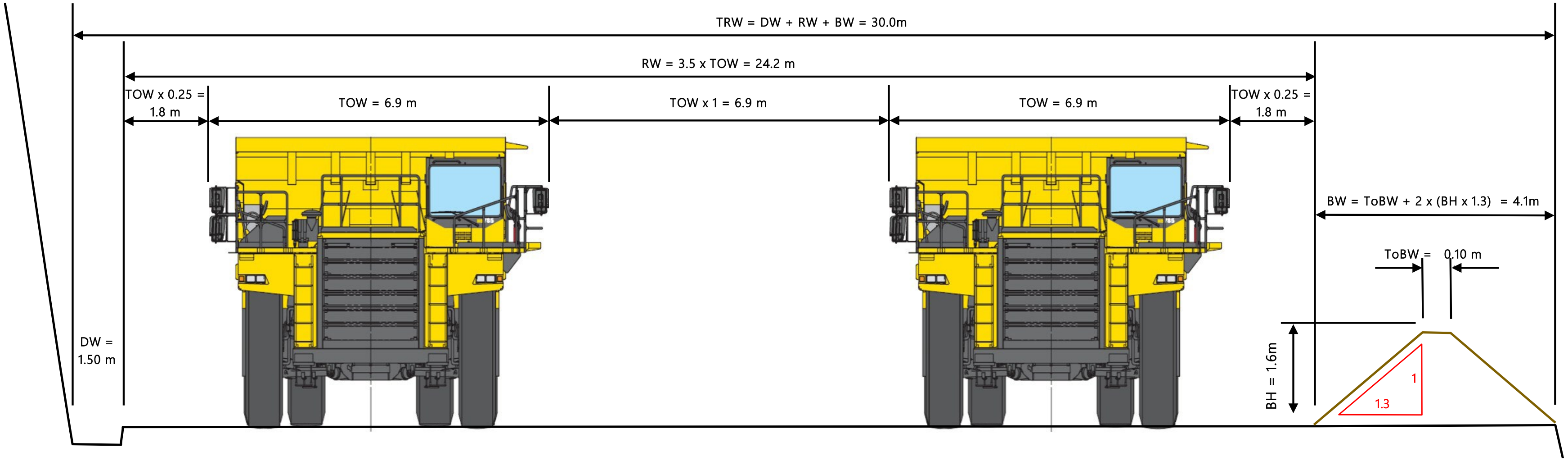

Figure 16.3: Dual-Lane In-Pit Ramp Layout

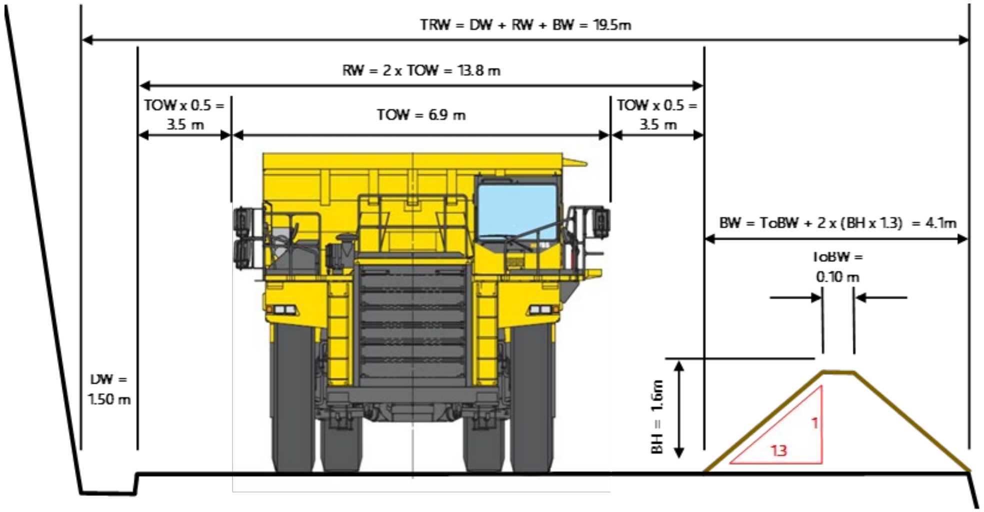

Figure 16.4: Single-Lane In-Pit Ramp Layout

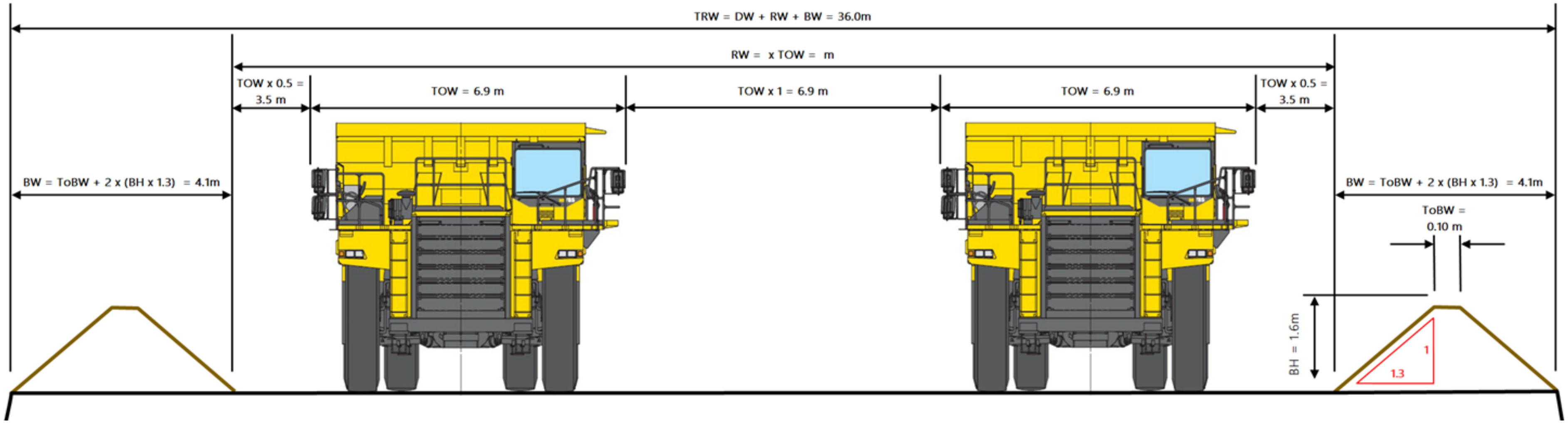

Figure 16.5: Dual-Lane Ex-Pit Road Layout

The truck selected in the preferred contractor submission is a Caterpillar 777 which is a 100 tonne class machine. This truck has an operating width of approximately 6.10 m (depending on variant) which results in a calculated dual lane ramp width of 26.5 m being 2.5 m less than the design width. Therefore, there is an opportunity to reduce the design road width during final design and implementation to suit the narrower truck and generate marginally less waste rock. However, the differential is considered immaterial to the overall material movement of the designs completed.

16.4.1.3 Mining Dimensions

A minimum mining width of 50 m was used for pushbacks and 40 m in the base of pits. There are only small sections on the footwall of NEB (less than 200 m) where the minimum mining width is reached and applied, otherwise the pushback width is greater than the minimum mining width.

16.4.1.4 Waste Rock Dump Parameters

There were no specific design recommendations for waste rock dump slopes from either a geotechnical or statutory standpoint, for either construction or final landform. Based on reasonable industry practise a reprofiled rehabilitation slope of 20˚ or less was considered appropriate. The design parameters for the construction and rehabilitated slopes and shown in Table 16.3 and Figure 16.6.

Table 16.3: Waste Rock Dump Design Criteria

Slope Configuration	Face Height	Face Angle	Berm Width	Overall Slope
	(m)	(°)	(m)	(°)
Construction	10	35	16.5	18
Final Landform	20	20	7	18

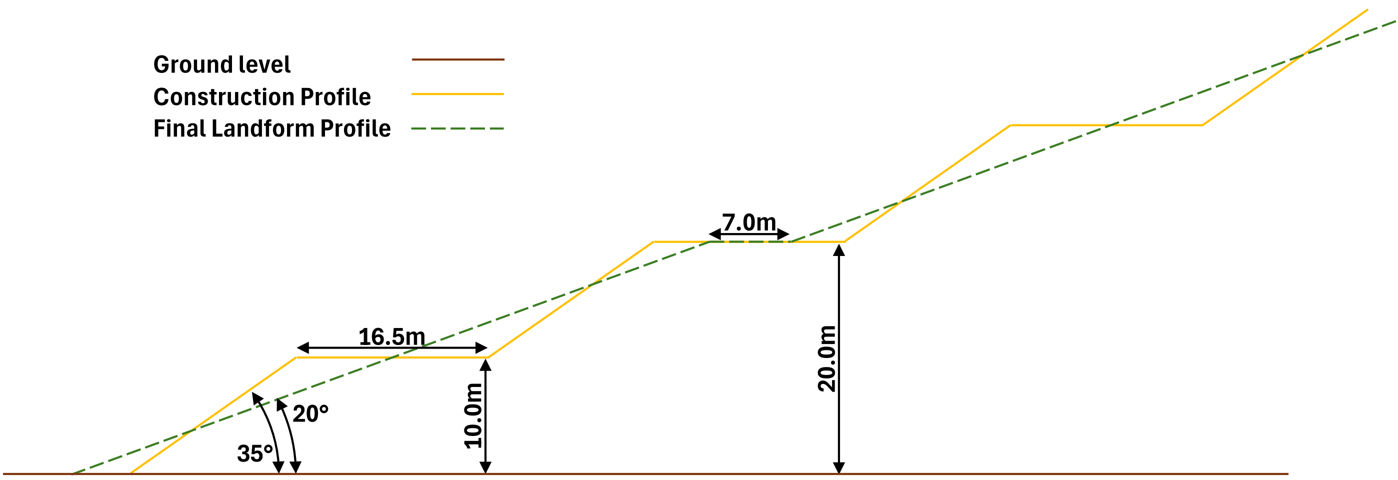

Figure 16.6: Waste Rock Dump Design Criteria (Construction and Final Landform)

The NEB waste rock dump averages 50 m to 65 m above ground level, with the BC dump only extending 20 m to 25 m above ground level.

16.4.2 Open Pit Mine Designs

The Project consist of three open pits, NEB, BC and GBE. The following sections provide detail on the pit designs and waste rock dump designs developed for the DFS and associated Mineral Reserve.

16.4.2.1 GBE Pit Design

The GBE pit design is approximately 250 m by 250 m as shown in Figure 16.7. The pit is circular with a clockwise spiral single lane ramp from the northeastern corner to the base of the pit on the assumption that the bulk earthworks contractor will be using smaller trucks than the main mining fleet. The pit is developed down to fresh rock interface (approximately 65 m below surface) to enable the establishment of the underground portal access in fresh rock while still mining some ore for later processing. As there is a delay between mining this pit and realising any value from the ore generated, the pit design focused on making the pit as small as practical to limit the early mining expenditure, while still providing sufficient space at the base of the pit for the establishment of the underground mine. There is a flat bench in the south of the pit base that is approx. 50 m by 60 m from which the portal access will be stablished. There is a 7.5 m deep sump in the north of the pit base to allow in-pit water to be managed and minimise ingress into the underground workings.

The open pit optimisation only extends to a depth of 35 m for the revenue factor 1 shell as shown in Figure 16.8, however, the primary purpose of the GBE pit is to act as a box-cut for the portal access point to the main underground development decline as the risk of establishing the decline is considerably reduced if the development is carried out only in fresh rock, as opposed to through the overlying saprolitic clays. Therefore, the GBE design was extended a further 30 m to the fresh boundary to ensure the portal could be established in competent rock.

While the GBE pit design produces approximately 560 kt of ore at a grade of 0.73 g/t, and therefore does generate revenue, it generates a negative discounted value as:

- The pit is sub-optimal as described above and therefore has an excessive strip ratio for the revenue generated.
- Mining costs are accrued in almost two years ahead of production with any revenue delayed until the commencement of production.

The NPV of the GBE pit, accounting for the revenue delay, is approximately US$8m, however the additional cost to develop the underground decline from surface to the base of the GBE pit is approximately US$16m, approximately double the GBE pit value. More crucially using the GBE pit for access considerably lowers the risk to UG development and associated project timeline.

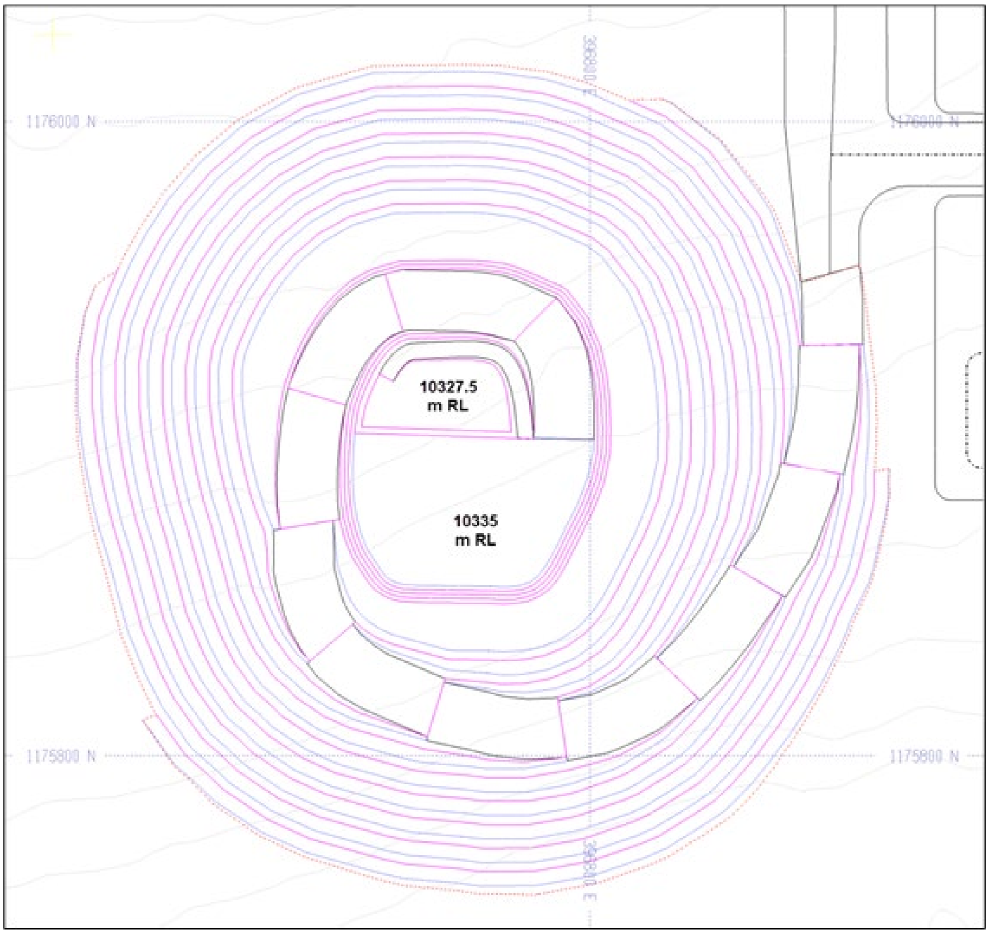

Figure 16.7: GBE Pit Design

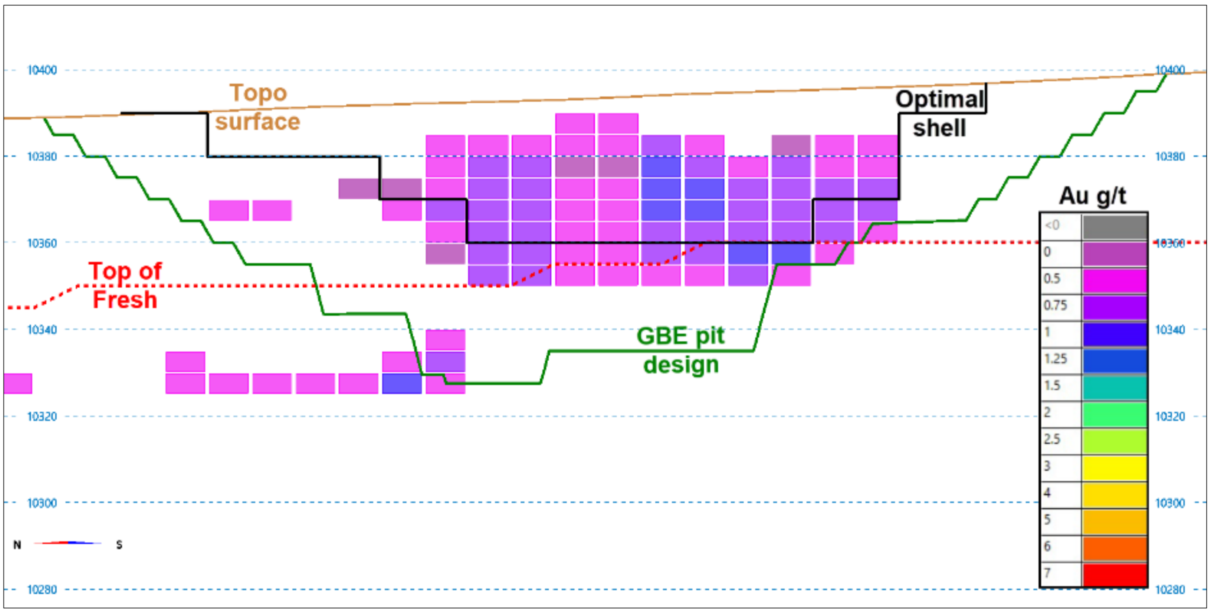

Figure 16.8: Cross Section through GBE Pit Design (10370E)

16.4.2.2 NEB Pit

NEB provides over 90% of the total ore inventory from the Project open pits. It has been designed in three stages as shown in Figure 16.9 to Figure 16.11.

NEB Stage 1 develops to a depth of approx. 105 m below surface (10320m RL) as shown in Figure 16.9. It has a north / south extent of approx. 630 m and an east / west extent of approx. 460 m. It has one dual access ramp from surface on the eastern footwall that proceeds clockwise to the 10345m RL, and then a single lane access to the pit base.

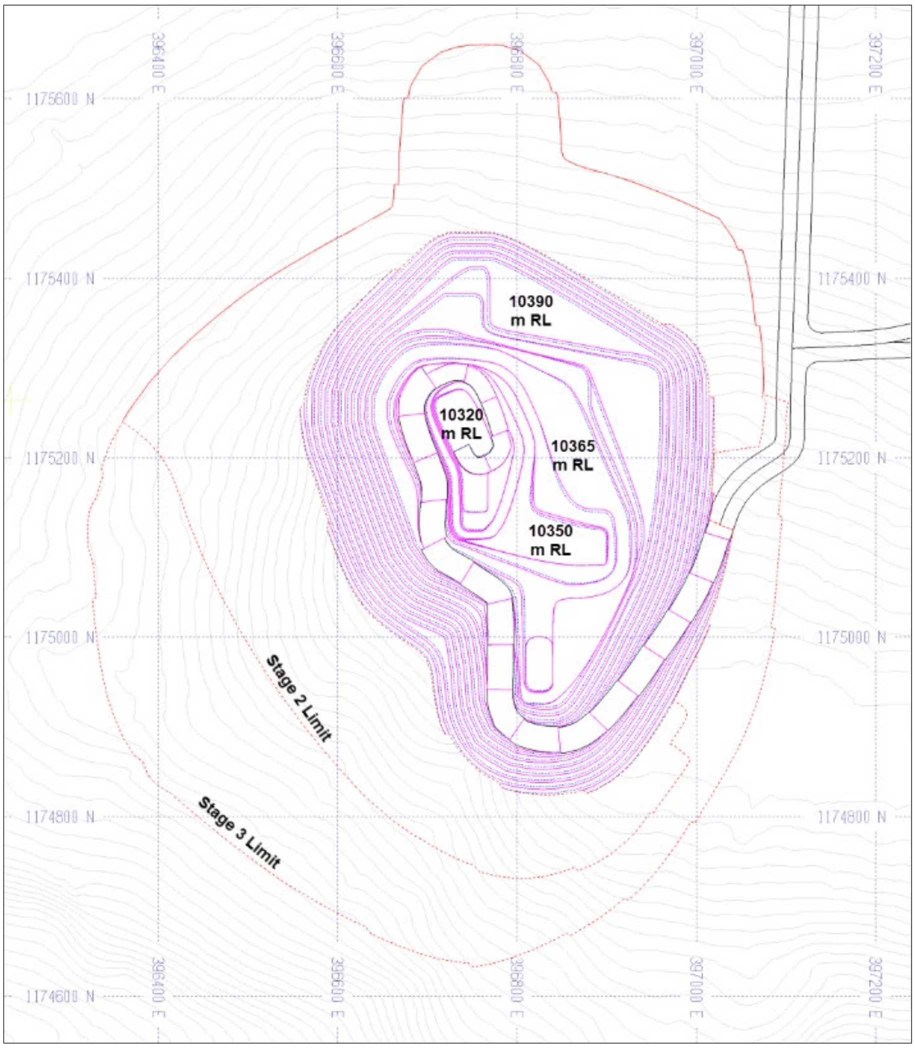

Figure 16.9: NEB Stage 1 Pit Design

NEB Stage 2 develops to a depth of approximately 145 m below surface (10280m RL) as shown in Figure 16.10 below. It has a north / south extent of approx. 930 m and an east / west extent of approximately 720 m. It has one dual access ramp from surface on the south-eastern footwall that proceeds clockwise to the 10350m RL on the north-western hanging wall. It then switches back down the western hanging wall to the 10325m RL and then switch backs again to the 10305m RL on the north-western hanging wall. A single lane access then proceeds to the pit base. Stage 2 extend to final pit limits in the north to north-east quadrant of the pit. Elsewhere it maintains a minimum 50 m mining width from the Stage 1 pit and the ultimate pit limit.

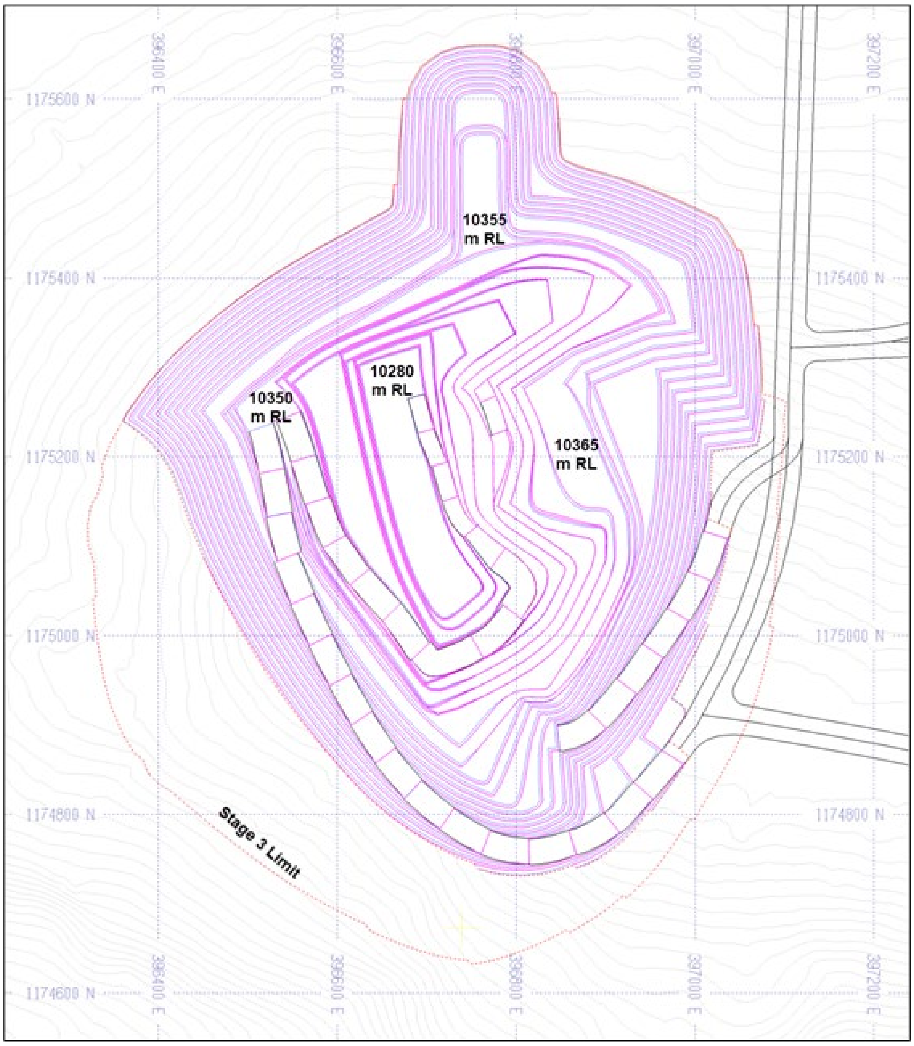

Figure 16.10: NEB Stage 2 Pit Design

Stage 3 is a final push-back on the western hangingwall of the deposit that targets high grade fresh ore at depth, as shown in Figure 16.11 below. The pit extends to approximately 250 m to 260 m from surface and to a final size of 780 m by 1,030 m. All the stages are serviced by a single dual lane ramp, which converts to single lane access at the base of the pit.

Figure 16.12 below provides a cross-section through the NEB pit, showing the geometry of the three mining stages and the orebody, along with the top of fresh surface material.

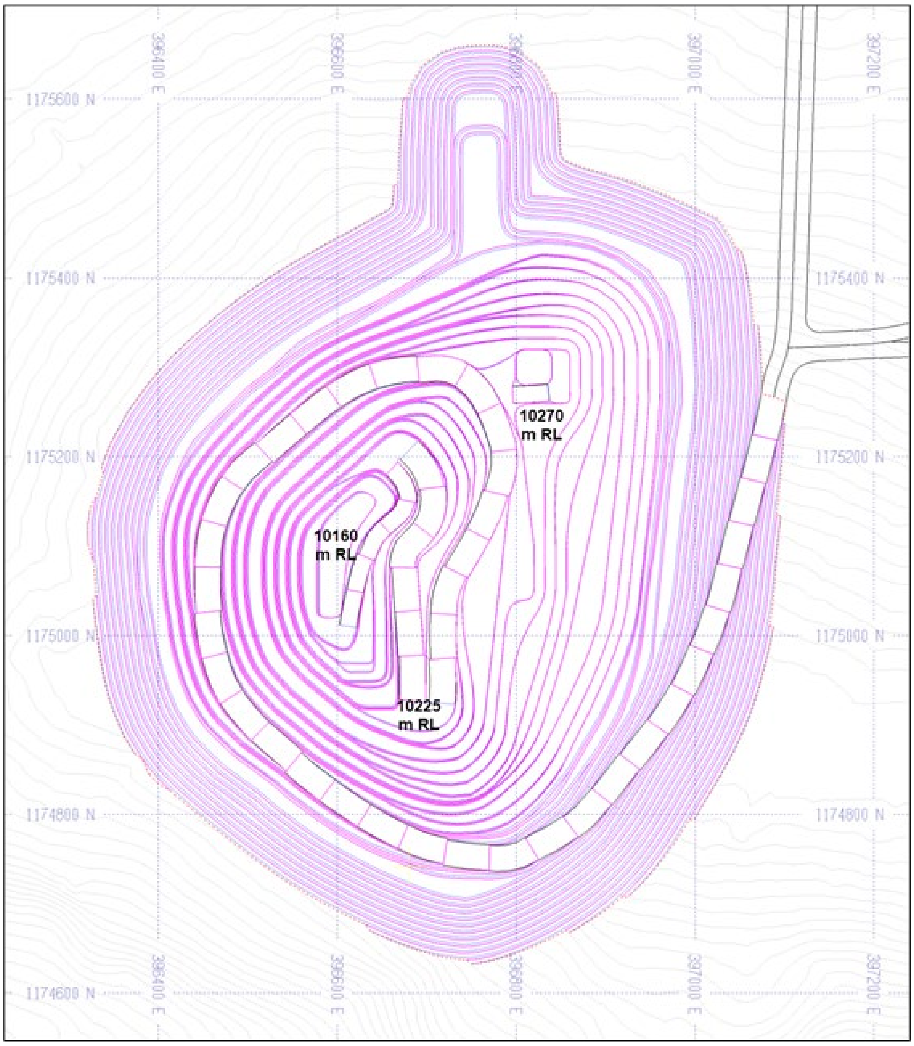

Figure 16.11: NEB Stage 3 Pit Design

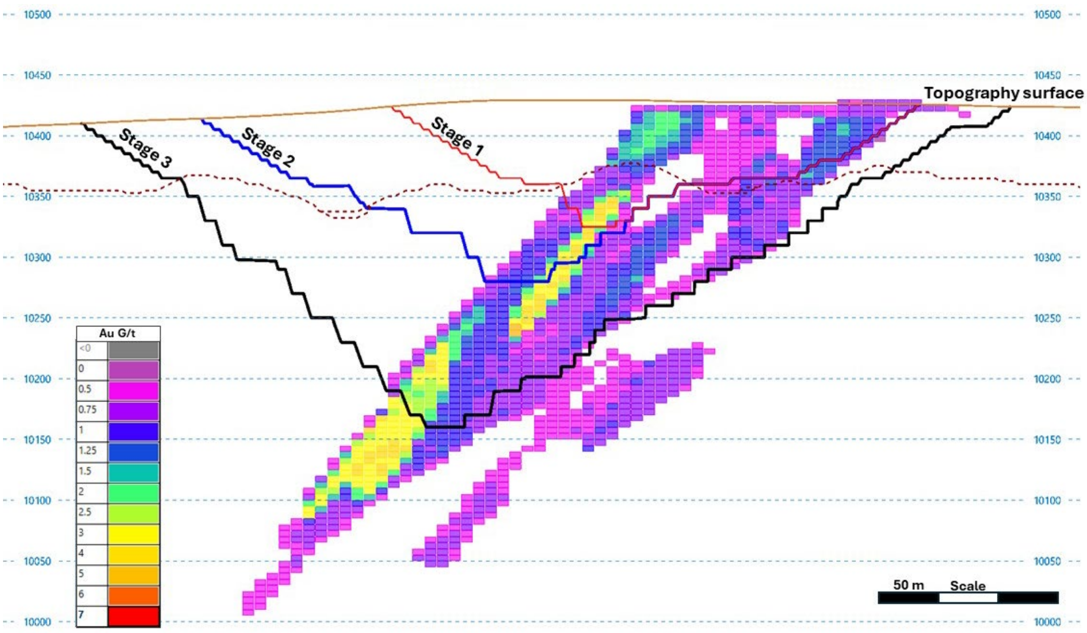

Figure 16.12: NEB Pit Cross Section

16.4.2.3 BC Pit Design

The BC pit is mined in a single stage towards the end of the LOM extending down to depth of approximately 70 m as shown in Figure 16.13 below. The BC pit targets Indicated Mineral Resources down to a depth of 70 m below surface, below which is only Inferred Mineral Resource. The BC pit is serviced by a single dual lane ramp, which converts to single lane access at the base of the pit.

The BC pit, due to its location in Bankan Creek and proximity to the Niger River, requires significant surface water management infrastructures including an upstream diversion bund and channel and a downstream flood protection bund. Further detail is provided in Section 18.8.

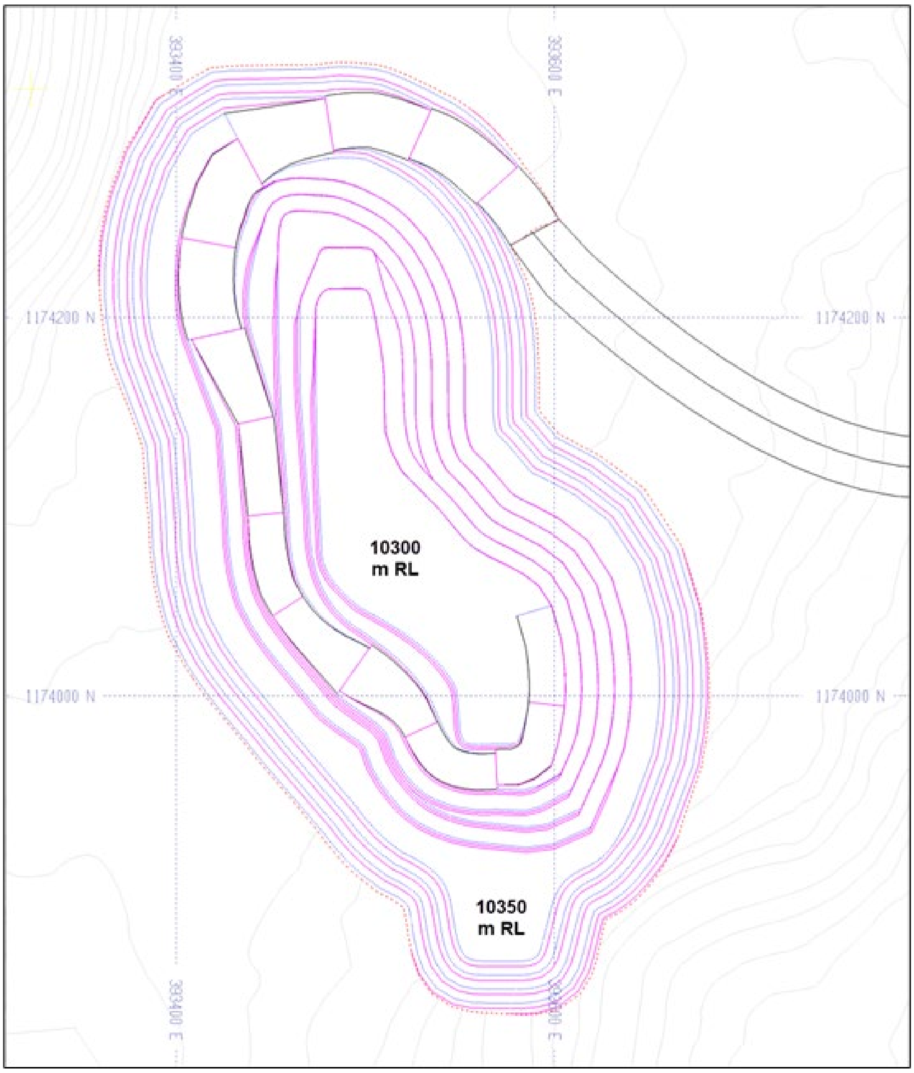

Figure 16.13: BC Pit Layout

16.4.2.4 Design to Optimisation Reconciliation

Table 16.4 provides a comparison of the pit design inventories against the open pit optimisation on which they are based. This provides a measure of how accurately the optimisation can be replicated as a practical design and therefore validation the optimisation and design outcomes. In general, the reconciliation is very close, with only GBE generating discrepancy due to its targeting of a deeper pit to facilitate the underground development.

Table 16.4: Optimisation Shell and Pit Design Comparison

Source	Area	Ore			Waste	Total	Strip Ratio
		Mt	Au (g/t)	Au (koz)	Mt	Mt	
Pit Design	NEB	39.6	1.36	1,738	75.3	114.9	1.9
	GBE	0.6	0.73	13	2.0	2.5	3.5
	BC	3.5	1.78	200	4.5	8.0	1.3
	Total	**43.7**	**1.39**	**1,951**	**81.7**	**125.4**	**1.9**
Optimisation Shell	NEB	41.2	1.38	1,827	76.3	117.5	1.9
	GBE	0.4	0.87	12	0.9	1.3	2.1
	BC	3.5	1.78	201	3.1	6.7	0.9
	Total	**45.1**	**1.41**	**2,040[1]**	**80.4**	**125.5**	**1.8**
Variation from Shell	NEB	-4%	-1%	-5%	-1%	-2%	2%
	GBE	30%	-16%	9%	120%	91%	69%
	BC	-1%	0%	-1%	43%	20%	44%
	Total	-3%	-1%	-4%	2%	0%	5%

Notes:

1. Small variation from Whittle output as inventory by deposit generated out of MineSight GMP.

As discussed previously, the dilution and ore loss vary depending on location. Table 16.5 details the average dilution and ore loss for each pit and a global average.

Table 16.5: Dilution and Ore Loss Within Pit Designs

Deposit	Ore Loss			Dilution
	Modelled	Operational	Total	Modelled
GBE	11.3%	1.0%	4.0%	8.7%
NEB	3.2%	1.0%	4.0%	3.3%
BC	4.9%	1.0%	5.9%	5.4%
Global Average	**3.4%**	**1.0%**	**4.4%**	**3.6%**

This clearly shows that the smaller pits experience higher levels of ore loss and dilution than the main NEB pit, particularly in the case of GBE. This is appropriate, as it reflects the broader and more continuous nature of the NEB orebody, which means a higher proportion of the ore blocks sit within the orebody and do not attract an "edge" dilution.

16.4.3 Waste Rock Dump Design

The waste dumps for the open pit mining provide sufficient capacity for all the as-mined pit waste inclusive of allowances for appropriate swell.

The GBE waste rock dump will be located to the north of the pit as shown in Figure 16.14. It will be approximately 500 m by 250 m and will have an average height above surface of 10 m on the south to 15 m on the north. This dump is rehandled to construct the ROM Pad at the start of production mining and the GBE waste dump location is then used for long term stockpiling of low-grade ore.

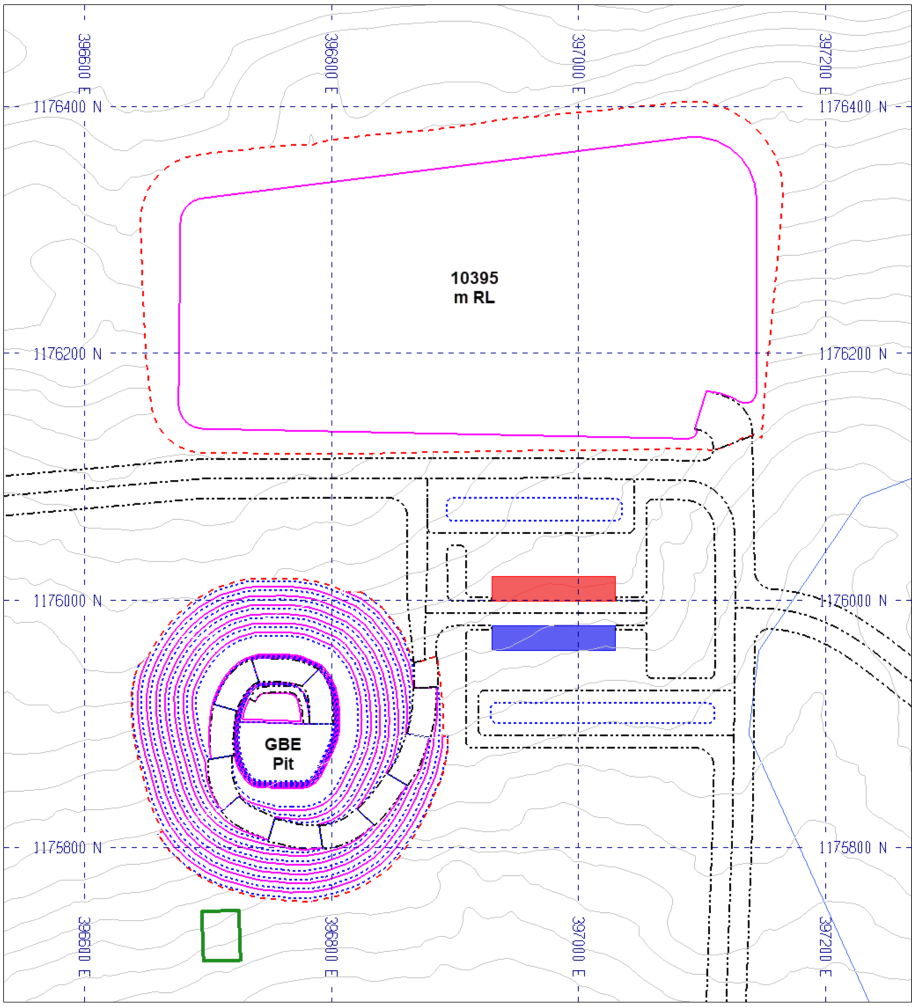

Figure 16.14: GBE Waste Rock Dump Layout

The main NEB waste dump will be located on the eastern footwall side of the pit. It is approximately 750 m by 1,750 m and has an average height above surface of 45 m on the west to 65 m on the east as shown in Figure 16.15 below. The ramp and road shown to the south services NEB Stage 2. The dump will be developed from the south to the north over the life of mine.

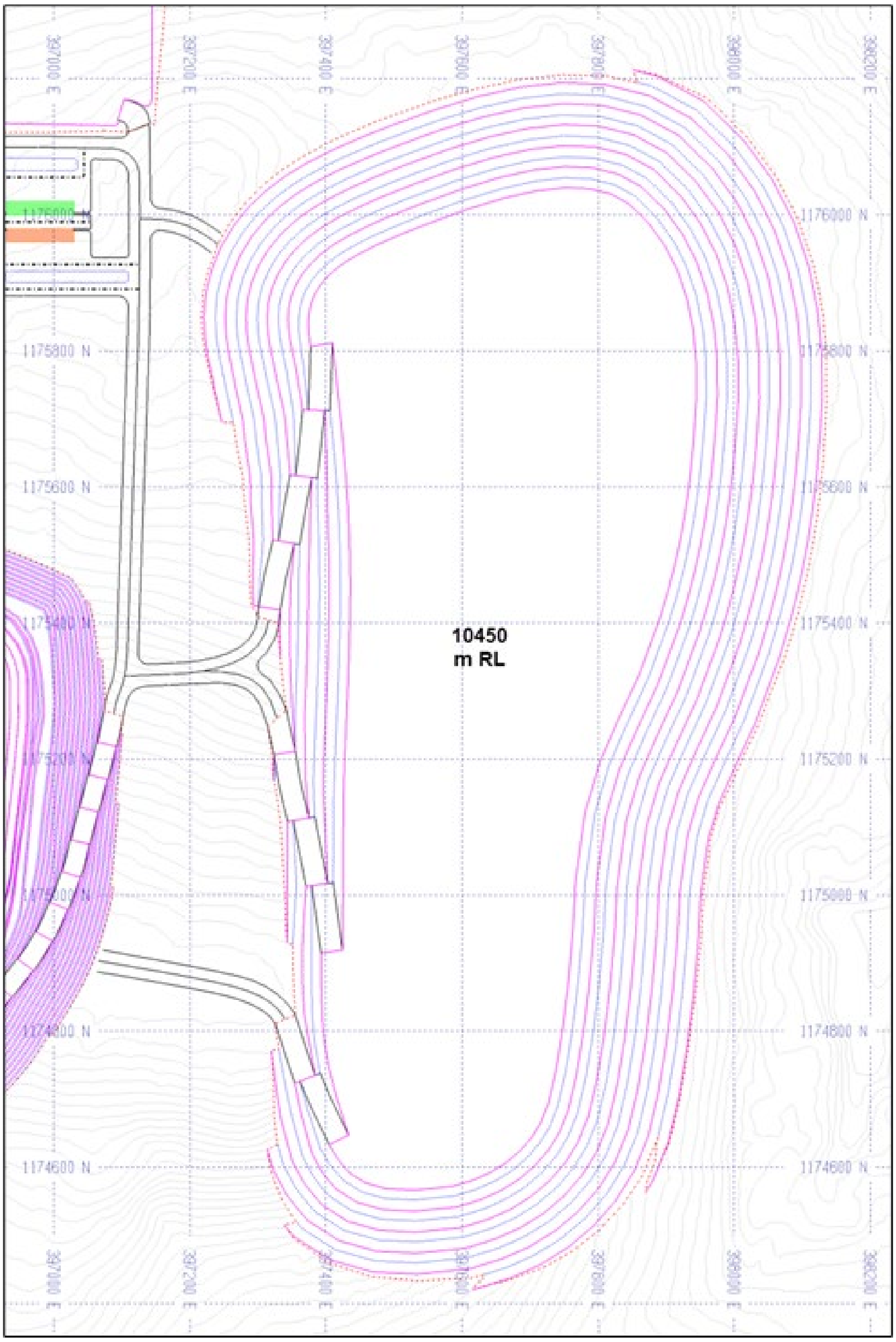

Figure 16.15: NEB Waste Rock Dump Layout

The BC waste rock dump is located northeast of the pit out of the water course as shown Figure 16.16 below. It is approximately 470 m by 570 m and will have an average height above surface of 25 m.

The waste dump volume requirements were calculated based on varying swell factors by rock type. Table 16.6 provides the calculations for required capacity and designed capacities.

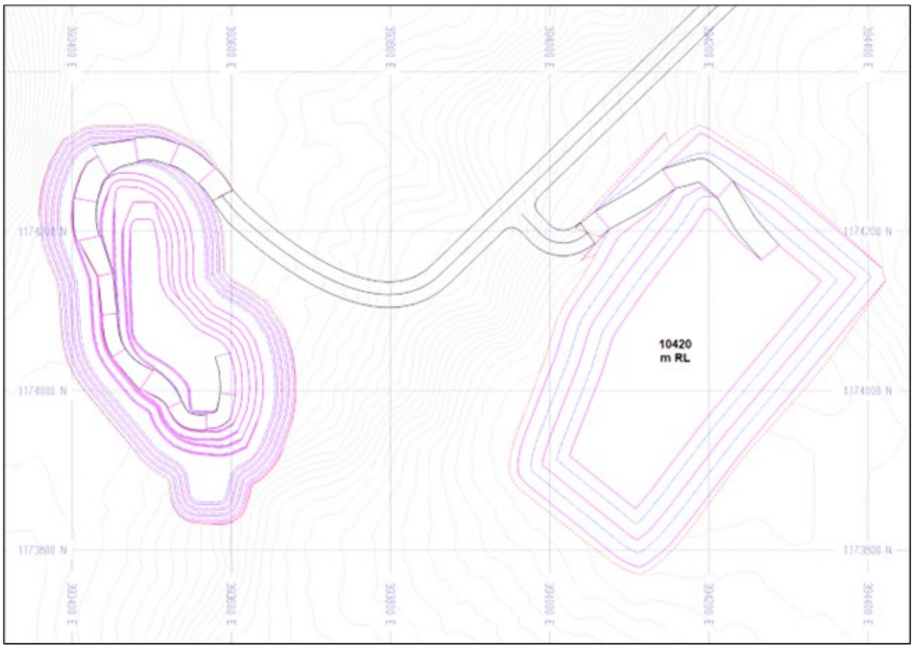

Figure 16.16: BC Waste Rock Dump Layout

Table 16.6: Waste Dump Capacities

Material Type	BC (Mbcm)	NEB/GBE (Mbcm)	Swell (%)	BC (Mlcm)	NEB/GBE (Mlcm)
Laterite	0.5	3.6	20%	0.6	4.3
Saprolite	0.5	15.6	15%	0.5	18.0
Mottled	0.1	0.9	15%	0.1	1.0
Saprock	0.2	1.1	20%	0.2	1.3
Fresh	0.8	14.4	25%	0.9	18.0
Total	**2.0**	**35.6**	**20%**	**2.4**	**42.6**
Dump Capacity				**2.5**	**44.6**
Spare Capacity				**+6%**	**+4%**

16.4.4 ROM Pad and Stockpile Design

As discussed previously, the GBE is the first pit to be mined commencing at the start of the two-year Project construction period, after which it is used as the primary access point for the underground mining operation. As such it requires the following infrastructure to be developed:

- Waste rock dump for waste produced from the GBE pit (refer to Section 16.4.3).
- ROM ore stockpile for ore produced from the GBE pit prior to process plant commissioning.
- Short term rehandling stockpiles for ore and waste material generated from the underground operation.

As discussed in Section 16.4.3, the waste rock stored in the GBE waste dump is subsequently rehandled to complete construction of the ROM Pad or the TSF. This area is then utilised as the long-term low-grade ore stockpile. This requires the area to be extended further to the east. Figure 16.17 below shows the long-term low-grade stockpile along with the outline of the initial GBE waste dump.

A short-term ore stockpile for the GBE open pit ore is designed to the east of the pit within the ultimate footprint of the NEB waste dump. This ore stockpile will be drawn down over the first years of plant feed, such that it is depleted when this area is required for the NEB waste dump. Figure 16.17 below also show this stockpile and the limit of the NEB waste dump.

The underground haulage trucks will bring both ore and waste material to the surface where it will be placed in on-ground stockpiles adjacent to the GBE ramp exit, as shown in Figure 16.17. From here the ore will be rehandled directly to the process plant, and waste rock will be rehandled to the NEB waste dump. This will be undertaken by the open pit mining contractor on an on-going basis as part of their schedule of rates. Detail of the truck movement around the GBE pit is shown in Figure 16.18.

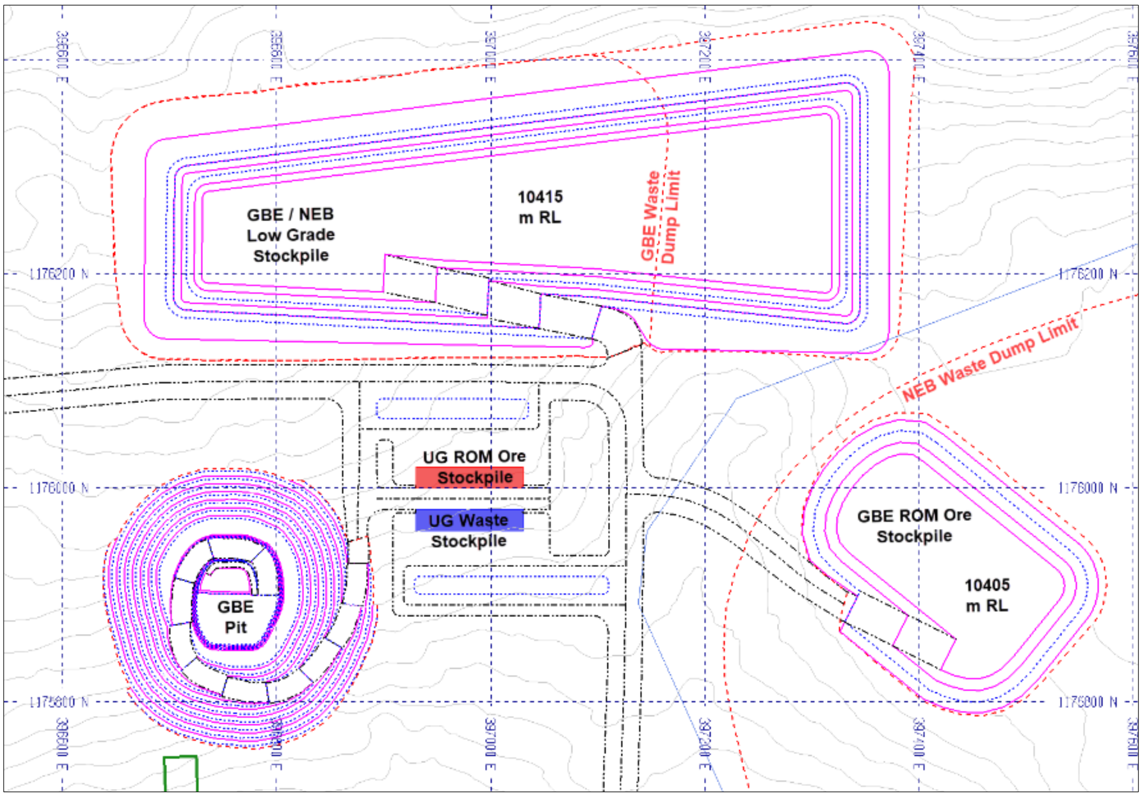

Figure 16.17: GBE Pit Infrastructure Layout

The ROM Pad has been designed with three finger stockpiles and an aerial skyway as shown in Figure 16.19. The fingers are 5 m high and average 175 m in length for a total capacity of approximately 300 kt of ore. The ROM pad base requires 1.67 Mlcm of waste fill material. Approximately 1.23 Mlcm will be rehandled from the GBE waste dump, with the remainder being sourced from waste mined from NEB pit during the pre-production period.

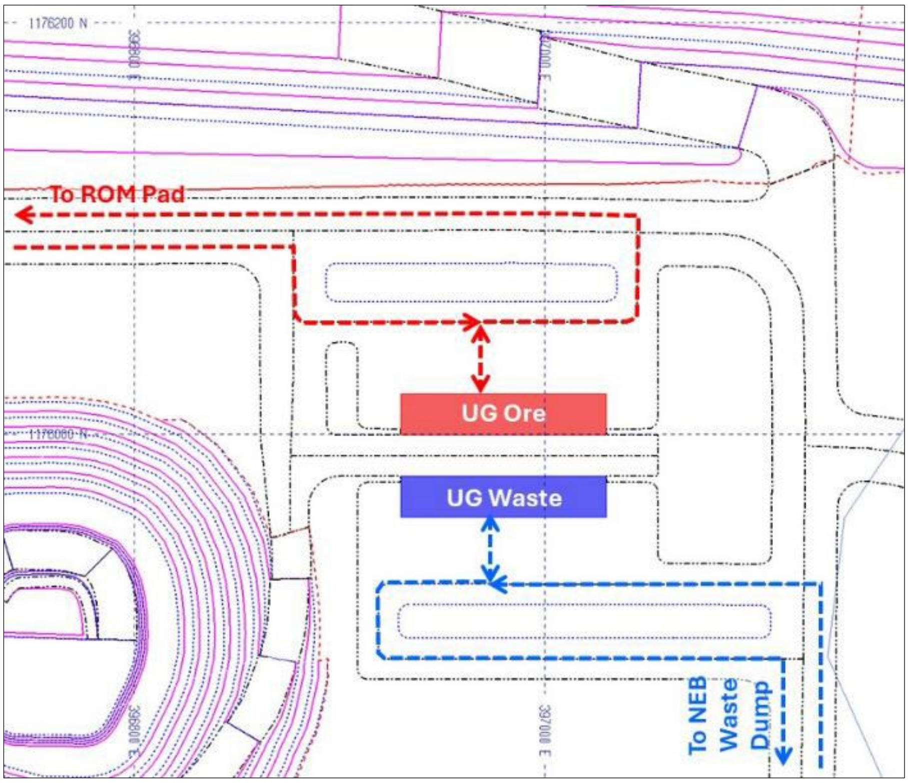

Figure 16.18: GBE Underground Stockpile Traffic Flow

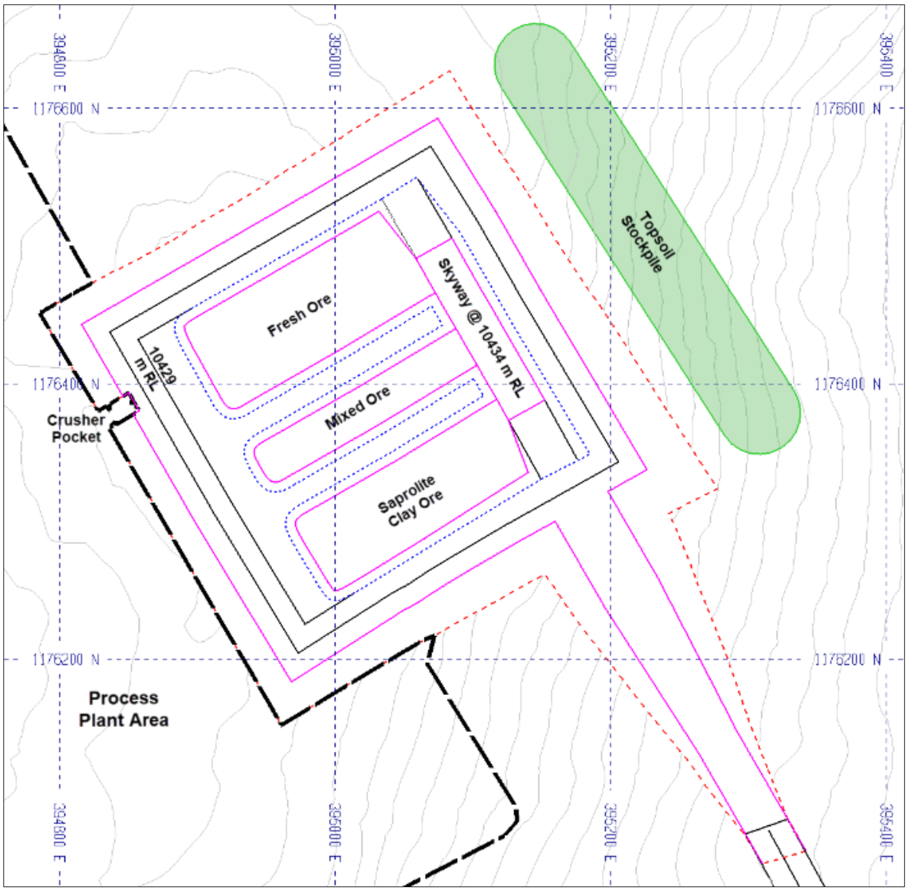

Figure 16.19: ROM Pad Layout

A separate low-grade stockpile has not been designed for the BC pit. Approximately 780 kt of BC low-grade ore is stockpiled over the final quarter of the mine life. It is assumed that most of this material can be stockpiled on or around the ROM Pad over this period, with any shortfall being stockpiled on the NEB low-grade stockpile.

16.4.5 Topsoil Stockpiles

Topsoil will be stripped from working areas and placed in stockpiles.

The topsoil stockpile areas were calculated based on removing an average of 0.3 m of topsoil and this material then being stockpiled to a height of 2.5 m. A swell of 10% was assumed. A total of 32.7 ha of topsoil stockpile volume is required and the design capacity equates to 33.6 ha.

Figure 16.20 below shows the location of topsoil stockpiles around NEB and GBE pits.

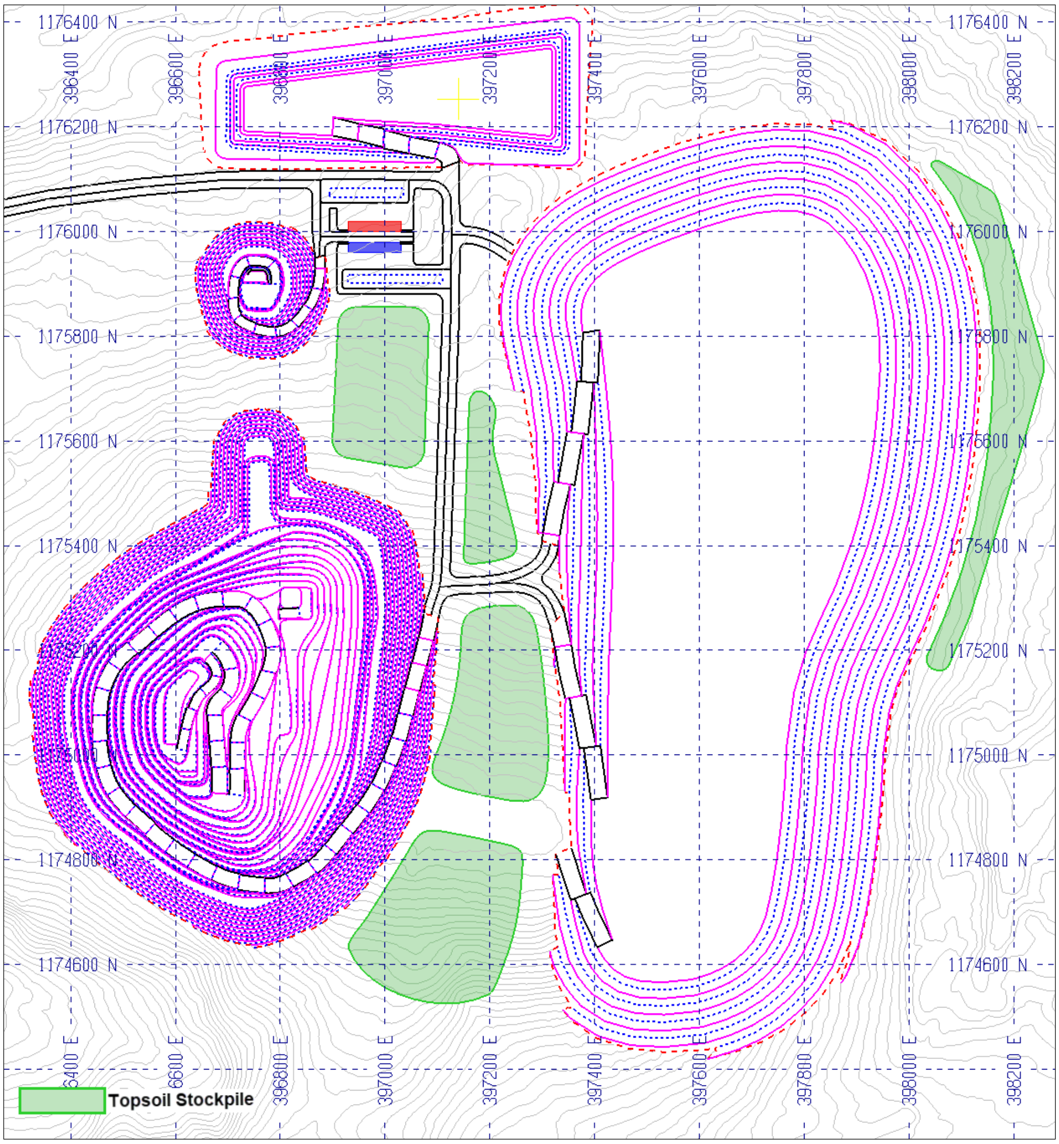

Figure 16.20: NEB/GBE Topsoil Stockpile Layout

Figure 16.21 shows the layout of topsoil stockpiles around the BC pit. The topsoil stockpile adjacent to the ROM pad is shown in Figure 16.19.

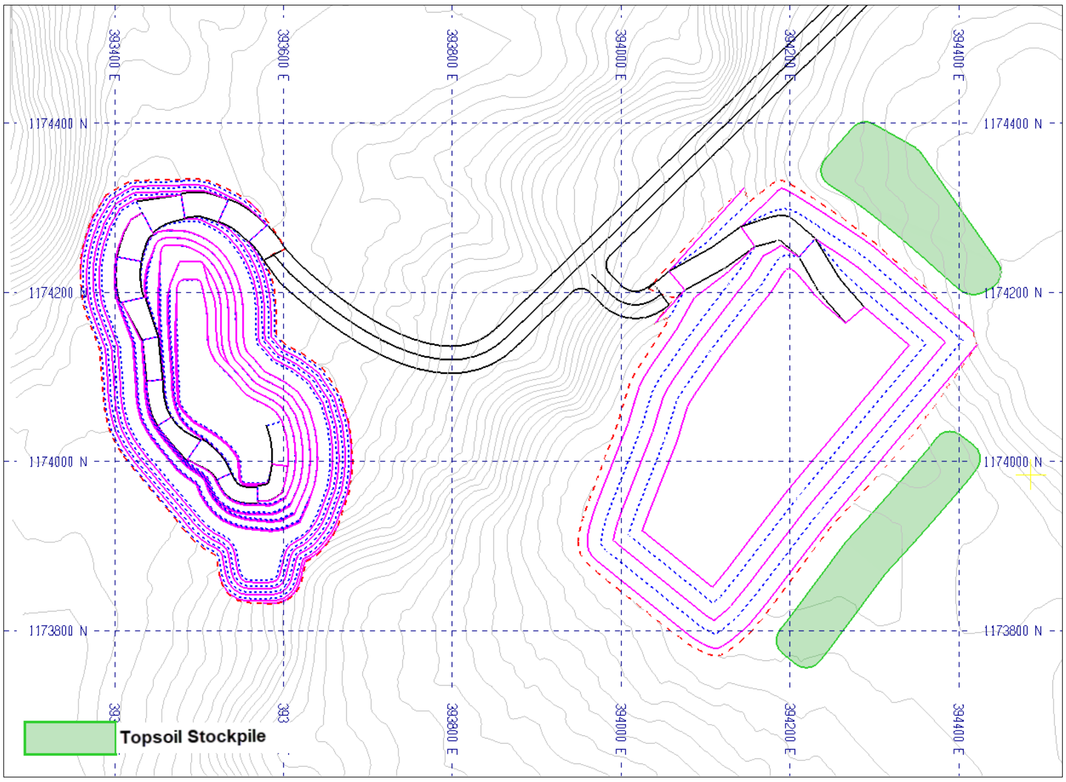

Figure 16.21: BC Topsoil Stockpile Layout

The operational approach will be to undertake progressive rehabilitation of the waste dumps wherever possible. Therefore, some topsoil stockpiles may be progressively reclaimed and then the same location used for subsequent stockpiling. Also, areas within the northern end of the large NEB WRD dump footprint may be utilised for topsoil stockpiling, particularly for material stripped for the WRD itself. However, the scheduling of the topsoil stockpiling and reclaim has not been undertaken as part of this study. Consequently, the land utilisation for topsoil stockpiles presented here can be considered a worst-case maximum.

16.5 Underground Mine Design

The underground mine design adopts a top-down approach, beginning with the optimised stope layout and subsequently defining the extraction levels, access development, and decline infrastructure. This stope-driven sequencing ensures that the overall design remained aligned with key considerations, including safety requirements, production targets, and equipment constraints. The NEB underground deposit has demonstrated favourable geotechnical conditions for development design, with no significant faults or structures identified to date. This has enabled a relatively straightforward development design. The final layout reflects these priorities and incorporates the detailed specifications presented in the following sections.

Overall mine design has level heights of 20 m, industry standard haulage and ventilation decline, fresh air raise for refrigeration, a main return air raise, sill pillar, crown pillar and exploration drill drives. Figure 16.22 and Figure 16.23 show an overall scheme of the mine design including inferred stopes which are excluded from the Mineral Reserve and LOM production schedule.

It should be noted that the optimisation and design of the underground was completed including consideration of potential future mining of the Inferred Mineral Resources, which is not included in the Mineral Reserve, for strategic planning purposes at the request of the Company. The Qualified Persons confirm that this approach has not impacted on the mining recovery of Mineral Reserves as the Inferred Mineral Resources are spatially separate and located below the Mineral Reserves.

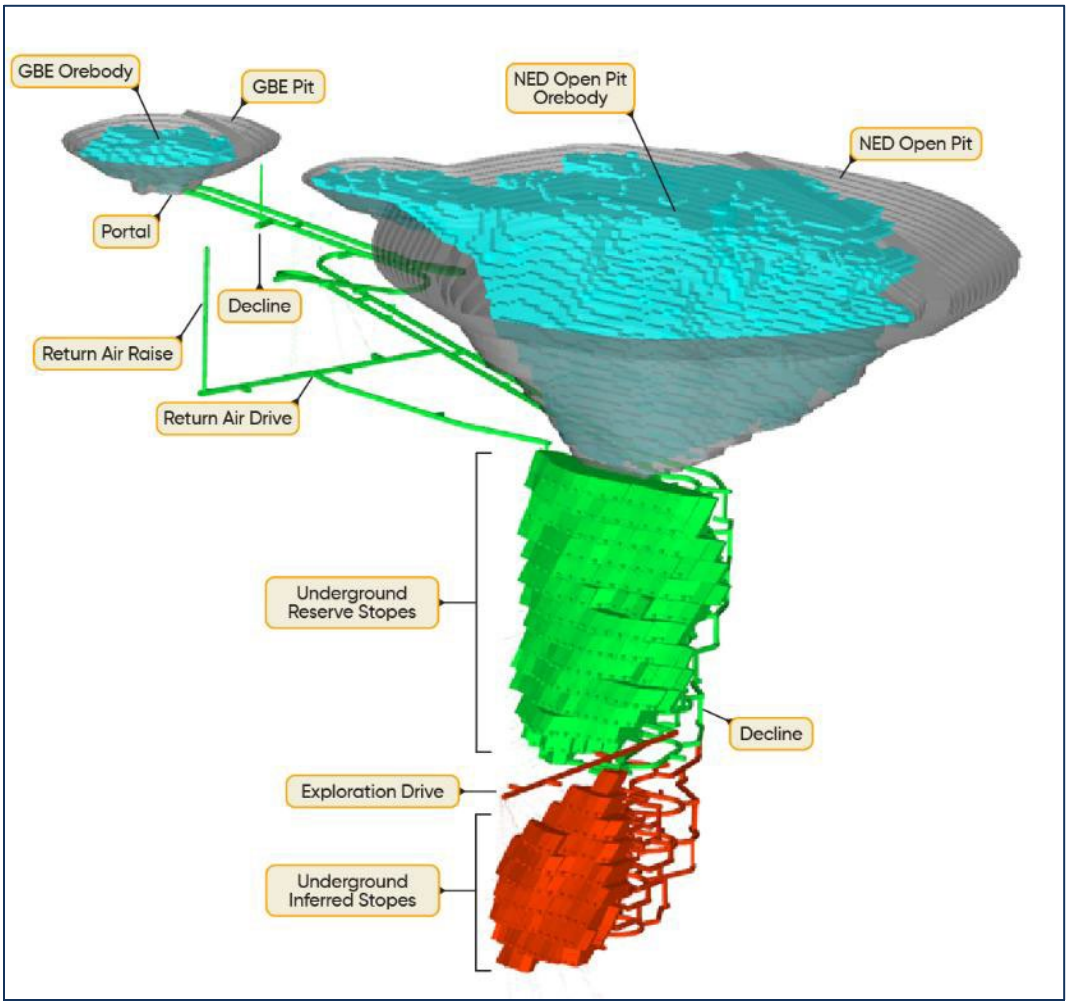

Figure 16.22: GBE Pit and Underground Mine Design – Looking East

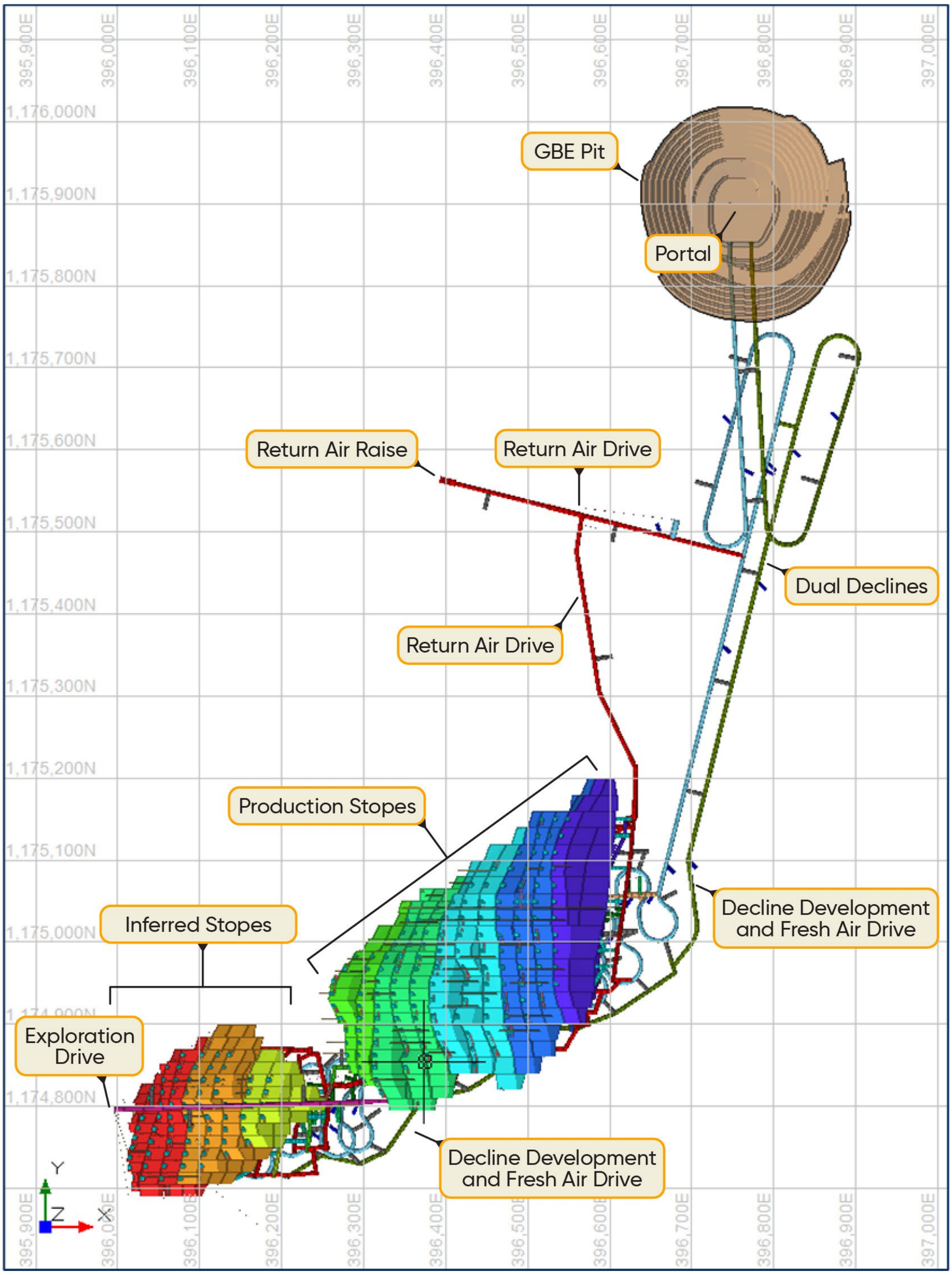

Figure 16.23: Plan view of GBE Pit and Underground Mine Design

16.5.1 Portal

In the PFS underground access was planned via a box cut from surface, followed by the development of a decline through weathered material. However, subsequent review during the DFS identified several geotechnical challenges with this approach. The original decline alignment required development through approximately 400 m of highly weathered ground, including saprolite and saprock, which significantly increased ground support requirements and reduced development productivity. In particular, the poor ground conditions could necessitate the use of concrete arches, further reducing advance rates and increasing costs. Figure 16.24 depicts the planned boxcut from the PFS, illustrating the decline trajectory relative to the saprock base, which marks the end of the problematic ground.

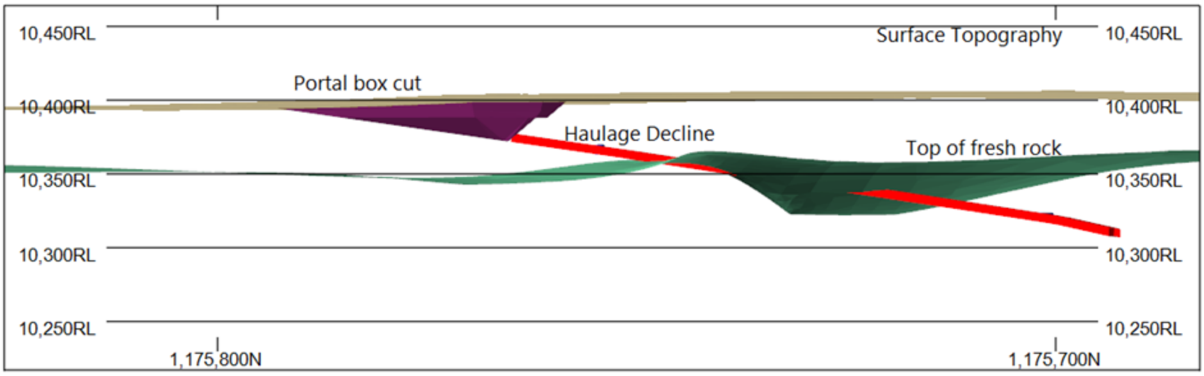

Figure 16.24: PFS BoxCut and Decline Looking North

To address this, an alternative access strategy was developed as part of the DFS which involves mining the GBE pit and establishing the underground portal at this location, where competent fresh rock is exposed. While this adjustment results in some early-stage pre-production mining expenditure, it significantly de-risks the underground access by eliminating development through poor-quality ground. The proposed GBE design and decline access is shown in Figure 16.25.

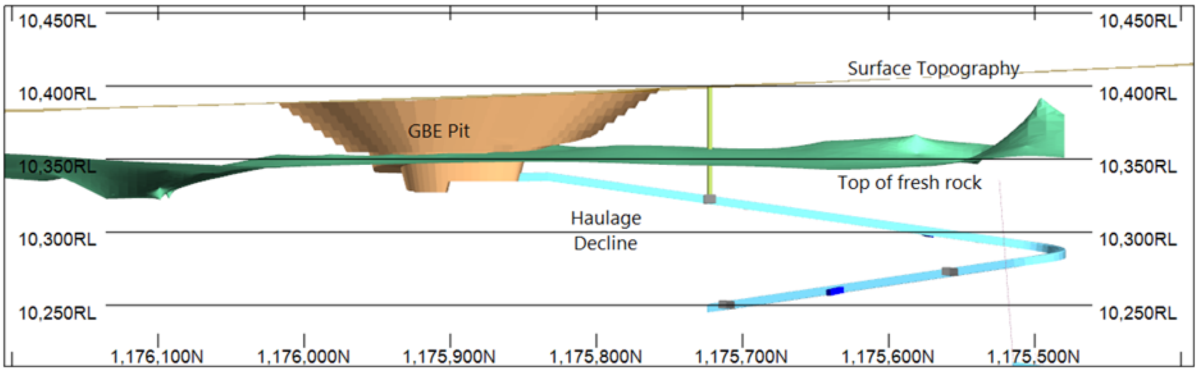

Figure 16.25: DFS GBE and Decline Access Looking East

The expected improvement in development conditions is substantial, with projected advance rates expected to increase from 50 m per month using concrete arches to up to 160 m per month using the proposed mesh and bolts, which is typical in these geotechnical conditions. This change is expected to result in both improved schedule performance and reduced overall cost for decline development. The portal is located at the 10335 mRL. Geotechnical constraints placed on the portal location in order to guarantee access stability include:

- Minimum of 20 m fresh rock above of the portal.
- Pit wall at 20 m bench and 75 degree face angle.
- Berm of 15 m on top of the portal.
- Portal located at southern end of the pit, where there is more fresh rock present.

There are two portals planned for the underground mining of NEB. The first is designed as a haulage decline, with the second being a ventilation decline that will also be considered as secondary means of egress for the entire mine.

A sump was designed in GBE the pit floor to catch water and prevent it from flowing to the underground workings. This sump has a capacity of 8,800 kL, and a 37 kW submersible pump will be located in the sump to pump the water from the pit for continuous sump dewatering.

Figure 16.26 below shows the location of the ventilation and haulage portals in the GBE pit in plan view.

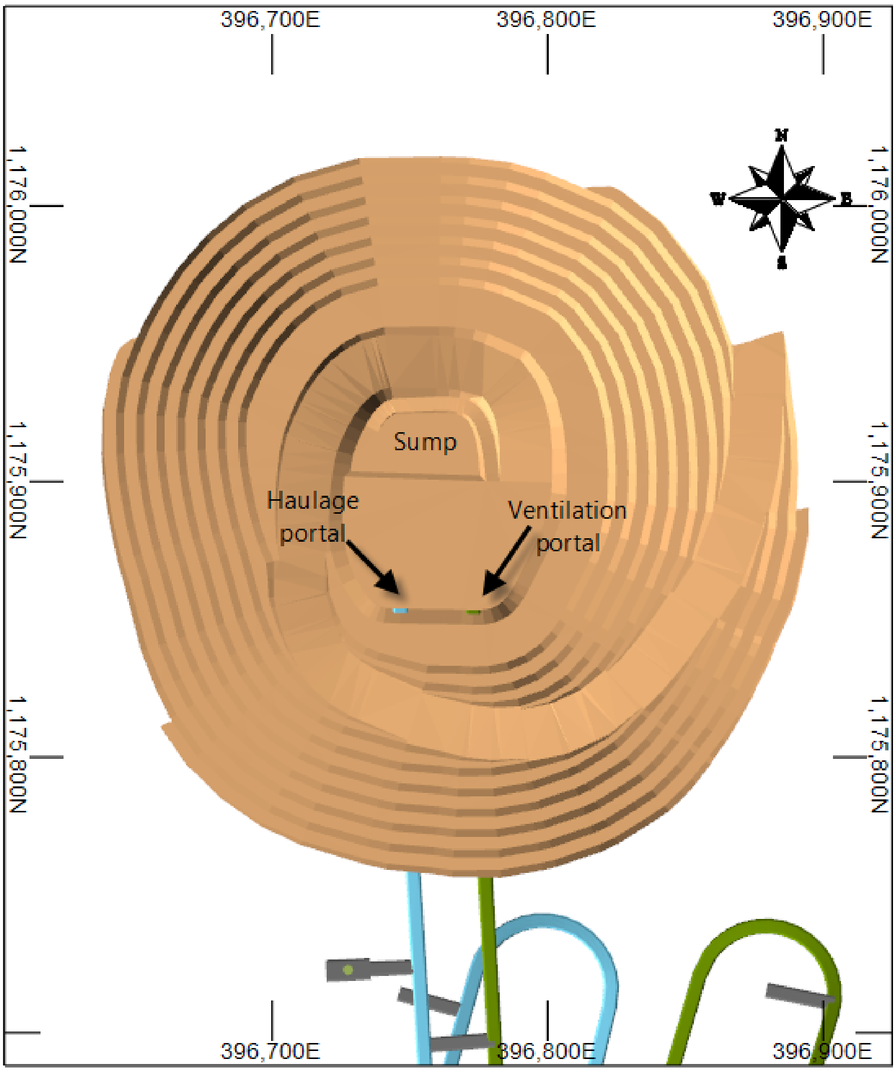

Figure 16.26: Haulage and Ventilation Portal Locations – Plan View GBE Pit

16.5.2 Development Design

A structured, top-down approach was adopted to ensure both operational efficiency and safety throughout the development layout. Longitudinal drives, hangingwall drives and ore drives were designed for all stopes, configured to align with the selected stoping method. Stockpile locations were optimised, ensuring placement within 150 m of material sources to support efficient haulage and bogging operations.

The footwall drives were divided into northern and southern sections, based on access intersections. Critical infrastructure, including escapeways and ventilation raises, were designed to extend from the backs of lower-level drives to the floors of the levels above, with sufficient clearance allocated for equipment setup at each collar.

The decline was engineered to provide seamless access across all production levels, with stockpile bays spaced at 160 m intervals and sumps installed every 40 vertical metres to manage groundwater inflows. Pump stations were positioned in accordance with infrastructure layout constraints to support dewatering and operational continuity.

At this stage of the study, ore drives, footwall drives, hangingwall drives and slot drives have been designed with zero gradient. These will be refined in later stages once higher-resolution geological and structural models are available.

The main haulage decline, and ventilation decline have been designed at a gradient of 1:7 down and with a minimum 20 m pillar separation between them. The mine design layout is shown in Figure 16.27 and the lateral and vertical development metres required are shown in Table 16.7.

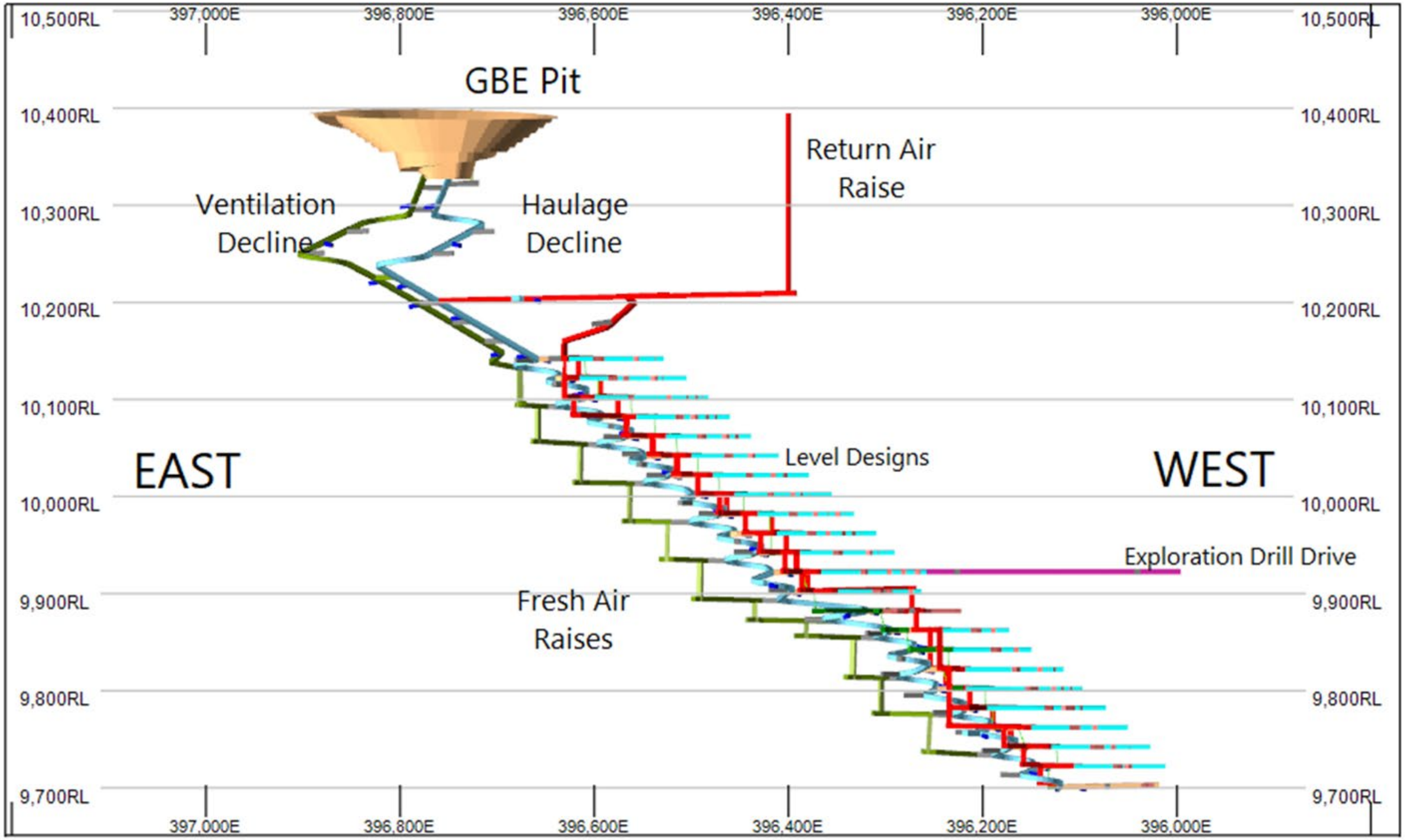

Figure 16.27: Overview of Bankan Underground Development Design – Looking South

Table 16.7: Overview of Bankan Development Design

Activity type	Profile	Type	Metres
Decline	5.5mWx5.8mH	Rectangle with arched corners	4,624.7
Access	5.5mWx5.8mH	Rectangle with arched corners	946.2
Escapeway Drive	5.5mWx5.5mH	Rectangle with arched corners	872.1
Footwall Drive	5.5mWx5.8mH	Rectangle with arched corners	3,560.0
Fresh Air Drive/Ventilation Decline	5.5mWx5.5mH	Rectangle with arched corners	2,346.8
Hanging wall Drive	5.0mWx5.0mH	Rectangle with arched corners	2,619.7
Longitudinal Drive	5.0mWx5.0mH	Rectangle with arched corners	4,501.6
Ore Drive	5.0mWx5.0mH	Rectangle with arched corners	14,414.4
Pump Station	8.0mWx6.0mH	Rectangle with arched corners	67.5
Refuge Chamber	5.5mWx5.8mH	Rectangle with arched corners	453.7
Return Air Way Drive	5.5mWx5.8mH	Rectangle with arched corners	2,679.5
Slot Drive	5.0mWx5.0mH	Rectangle with arched corners	279.9
Stockpile	5.5mWx5.5mH	Rectangle with arched corners	1857.9
Diamond Drill Drive	5.5mWx5.8mH	Rectangle with arched corners	370.1
Sump	4.5mWx4.5mH	Rectangle with arched corners	570.4
Escapeway Raise	1.2mD	Circular	347.2
Main Return Air Way Raise	5.5mD	Circular	182.3
Return Air Way Raise	4mWx6mL	Rectangular	529.7
Refrigeration Fresh Air Raise	3.5mD	Circular	74.1
Fresh Air Raise	4mWx6mL	Rectangular	383.0

16.5.2.1 Decline

The main access to the underground part of the orebody is via a twin decline coming from portals located within the GBE pit. The decline access has been designed from the GBE pit to intersect the strike of the NEB orebody to minimise the life of mine underground haulage costs and to optimise distances for secondary ventilation in the access drives. Declines are positioned on the footwall of the orebody in line with geotechnical industry guidelines which enable better ground control conditions as stope mining tend not to affect the footwall stability at distance.

Twin declines are connected through stockpiles every 140 m to 160 m where possible which will be used for ventilation set up and haulage system up to establishment of the main ventilation raise. The decline respects a 50 m pillar to the NEB pit and 65 m to the stope’s footwall. No hazardous faults or structures were identified at this stage.

16.5.2.2 Main Access and Production Levels

The main access off the decline to the orebody is positioned centrally relative to the orebody, with the decline located to the east in plan view. The location was chosen with the following considerations:

- Reduce the access development metres, and capital expenditure required to access the orebody.
- Minimise the horizontal distance requirements for secondary ventilation from the planned decline fresh air raises to the extremities of the level ore drives.

The level design was developed with a strong focus on both safety and the efficient extraction of the orebody. Each level includes a footwall drive located 25 m from the orebody, providing access to multiple mining fronts. A hangingwall drive is located on each level to facilitate stoping, creating a free mining face and allowing for the installation of hangingwall cablebolts for ground stability and control, with ore drives connecting the footwall to the hangingwall.

Longitudinal drives are positioned on the footwall of the orebody, slot drives were designed as required based on stope geometry. Each level also includes a refuge chamber drive, which may alternatively serve as substation location. Additional drives include sumps, stockpiles the north and south sides, return air drives in both ends and designated escapeway drives to ensure additional means of egress.

Figure 16.28 shows a standard level design on 9960 mRL.

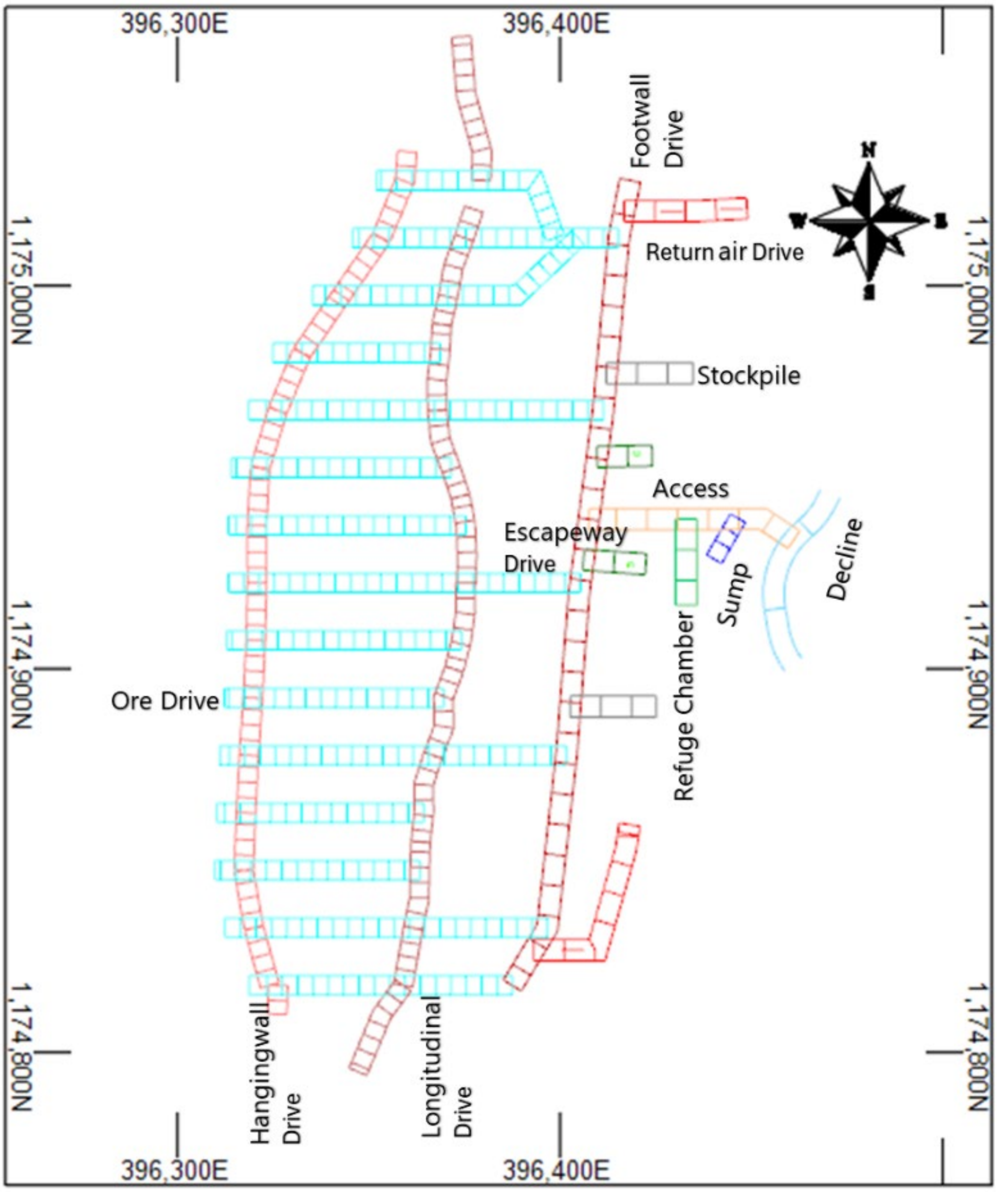

Figure 16.28: Typical Level Layout (9960 mRL) – Plan View

16.5.2.3 Vertical Development

Vertical development consists of all raises and service holes required to support underground operations. It is a fundamental component of mine infrastructure development, enabling the extension of ventilation, power distribution, emergency egress, and dewatering. These vertical connections ensure the essential services required to support development and production can advance concurrently with these operations.

The underground mine has two main air ways connected to the surface as part of the primary ventilation circuit, a 5.5 m diameter return air raise (RAR) and a 3.5 m diameter fresh air raise (FAR). The RAR will be developed first and will be located near the paste plant. It will be equipped with three primary ventilation fans to exhaust return air from the underground workings. The FAR will be developed close to the GBE pit and facilitates the delivery of fresh air, including cooled air from the connected refrigeration plant installed progressively as the underground mine moves deeper, into the underground mine.

The layout for the surface infrastructure is shown in Figure 16.29.

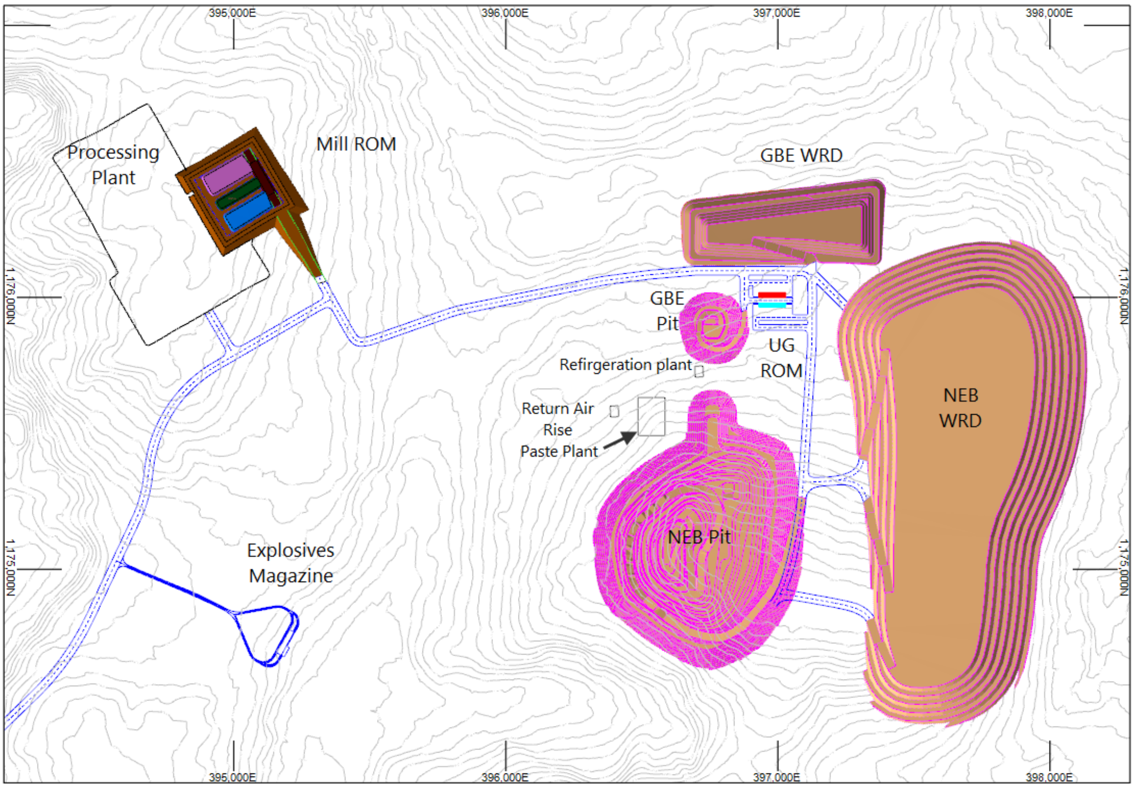

Figure 16.29: Underground Mine Surface Infrastructure

Due to the presence of weathered material near surface, shaft sinking has been planned and costed for both the RAR and FAR. Approximately 40 m of shaft sinking is anticipated for each raise, however, further geotechnical investigation is required to confirm the absolute extent of the shaft sinking requirement. Geotechnical drill holes are recommended to be undertaken for each raise location to define the depth and characteristics of the weathered zone and determine the shaft sinking requirements.

Figure 16.30 presents a cross-sectional view looking northwest, illustrating the locations of the RAR, FAR, paste fill delivery lines, rising main, and the electrical service hole.

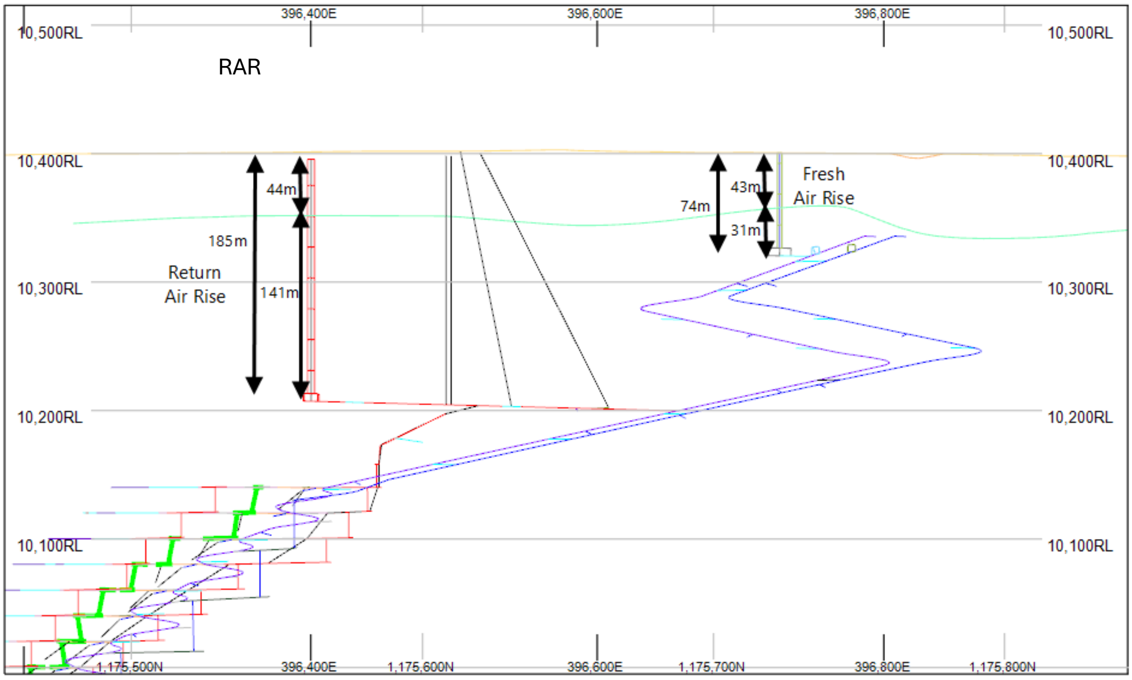

Figure 16.30: Vertical Development Elevation – Looking Northwest

Other vertical development will be undertaken using long hole drilling. The internal primary ventilation raises have been designed with a rectangular profile of 4.0 m W x 6.0 m L, allowing for adequate air to flow and ventilate the underground workings. Escapeways for emergency egress ladderways are planned at 1.2 m diameter. Service holes are planned from the surface allowing infrastructure to connect to the underground workings and these vary in size depending on designated use for paste, water pumping, and electrical connection.

16.5.2.4 Stope Layout

Stope design was developed based on optimisation results with emphasis on flexibility, safety, and productivity. The orebody, dipping between 40° and 45°, was a key factor in determining the level spacing. Two options, 20 m and 25 m, were assessed, with the 20 m interval ultimately selected. This decision reflects the limited geological information with respect to foliation and drill hole deviation, as well as the critical importance of drill hole accuracy for effective stope recovery. The 20 m spacing reduces the risks associated with hole deviation, enhances slot performance and extraction and aligns with the orebody geometry. This also reduces the internal dilution and assists to preserve the hangingwall stability throughout mining operations. The stope parameters for the two different level spacings are shown in Table 16.8 and Figure 16.31 shows a cross-sectional view of the level spacing options with the orebody dip.

Table 16.8: Stope Parameters for Different Level Spacings

Parameters/cases	20m spacing	25m spacing
Minimum hole length (m)	15	20
Maximum hole length (m)	21	28
Orebody dip (deg)	45	45

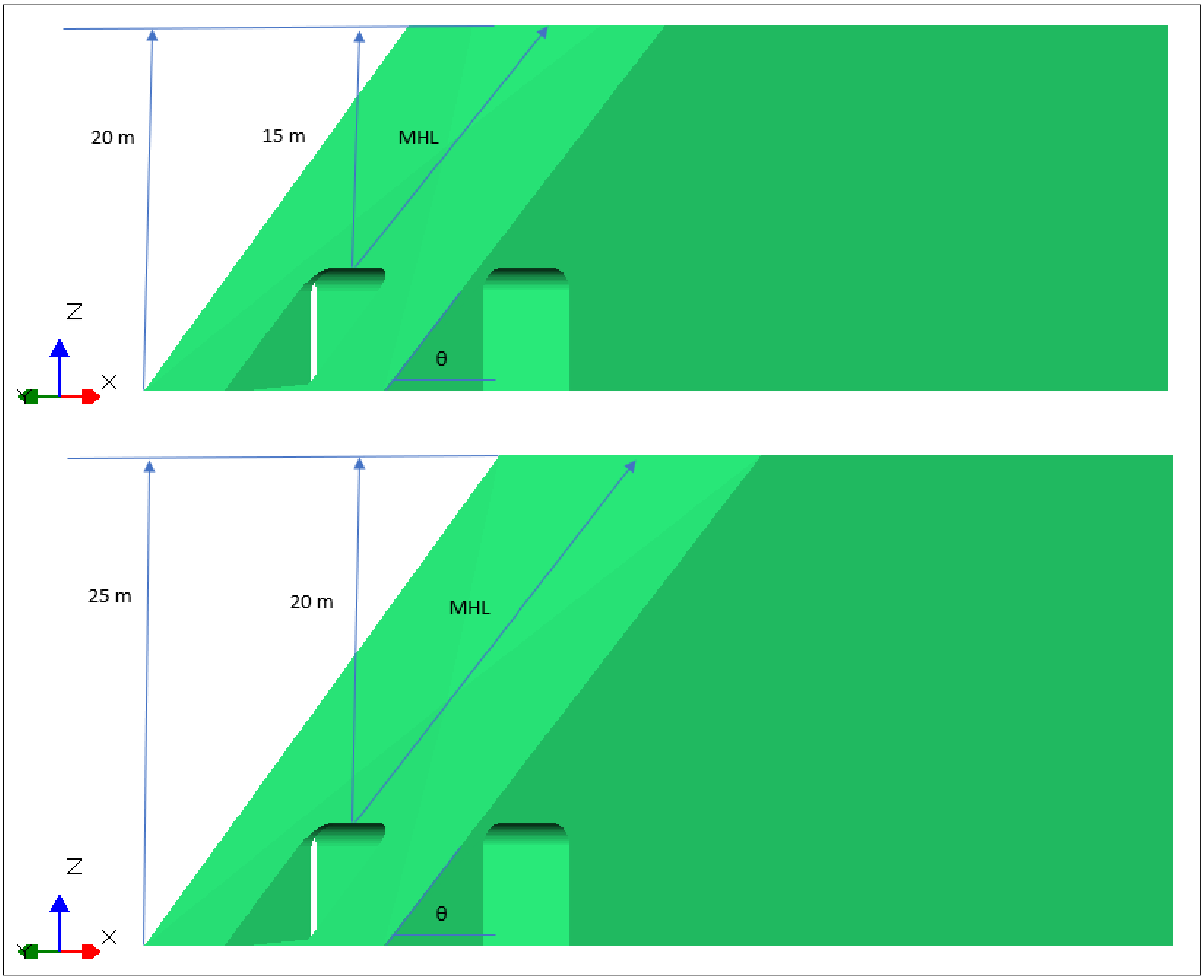

Figure 16.31: Level Spacing – Cross Section

Following the stope optimisation, the designs were assessed against geotechnical constraints, with a particular focus on hydraulic radius (HR) limits, one of the primary indicators of stope stability in underground mining. The HR is calculated by dividing the exposed surface area by its perimeter and is influenced by the quality of the surrounding rock mass. In this instance, where the orebody dips at approximately 45°, the HR was calculated for both the hangingwall (HW_HR) and the back of the stope (BKS_HR) to ensure that stope geometries remained within acceptable stability limits.

Once the stope optimisation calculations were completed, each stope was manually reviewed against the geotechnical criteria outlined in Section 15.4.3. Stopes that exceeded the acceptable HR limits were modified by subdividing them into multiple smaller shapes. Internal walls were adjusted to angles between 90° and 70°, as inclined walls contribute to paste fill dilution as paste does not sustain itself when footwall is mined. These modifications were implemented to ensure geotechnical stability and to minimise the impact of dilution on stope performance. Figure 16.32 shows the stope shapes generated using Deswik.SO, looking west and Figure 16.33 displays the shapes in plan view. These figures include inferred stopes (below 9,900RL) which are excluded from the Mineral Reserve and LOM production schedule.

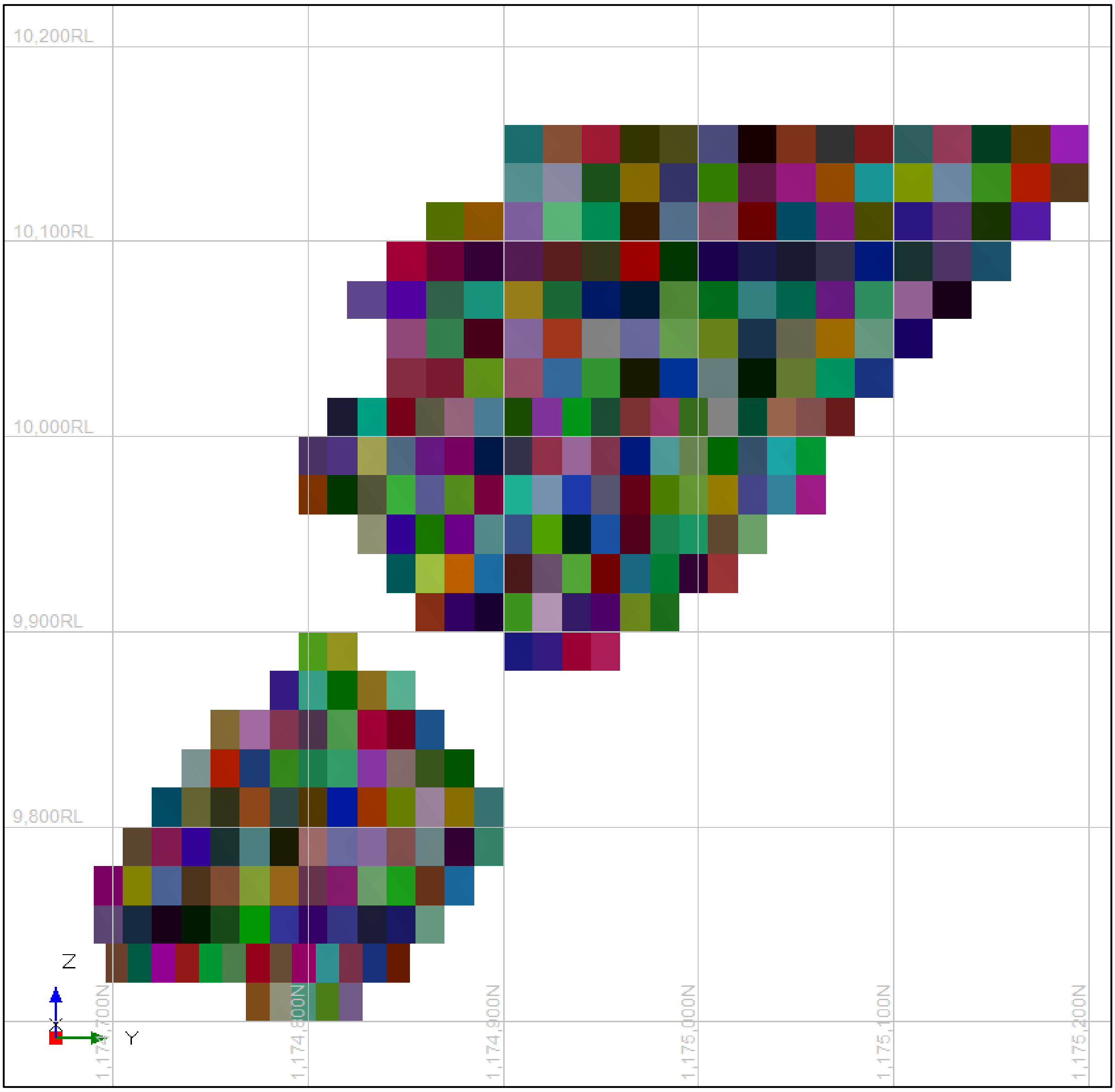

Figure 16.32: Stope Shapes Generated using Deswik.SO – Looking West

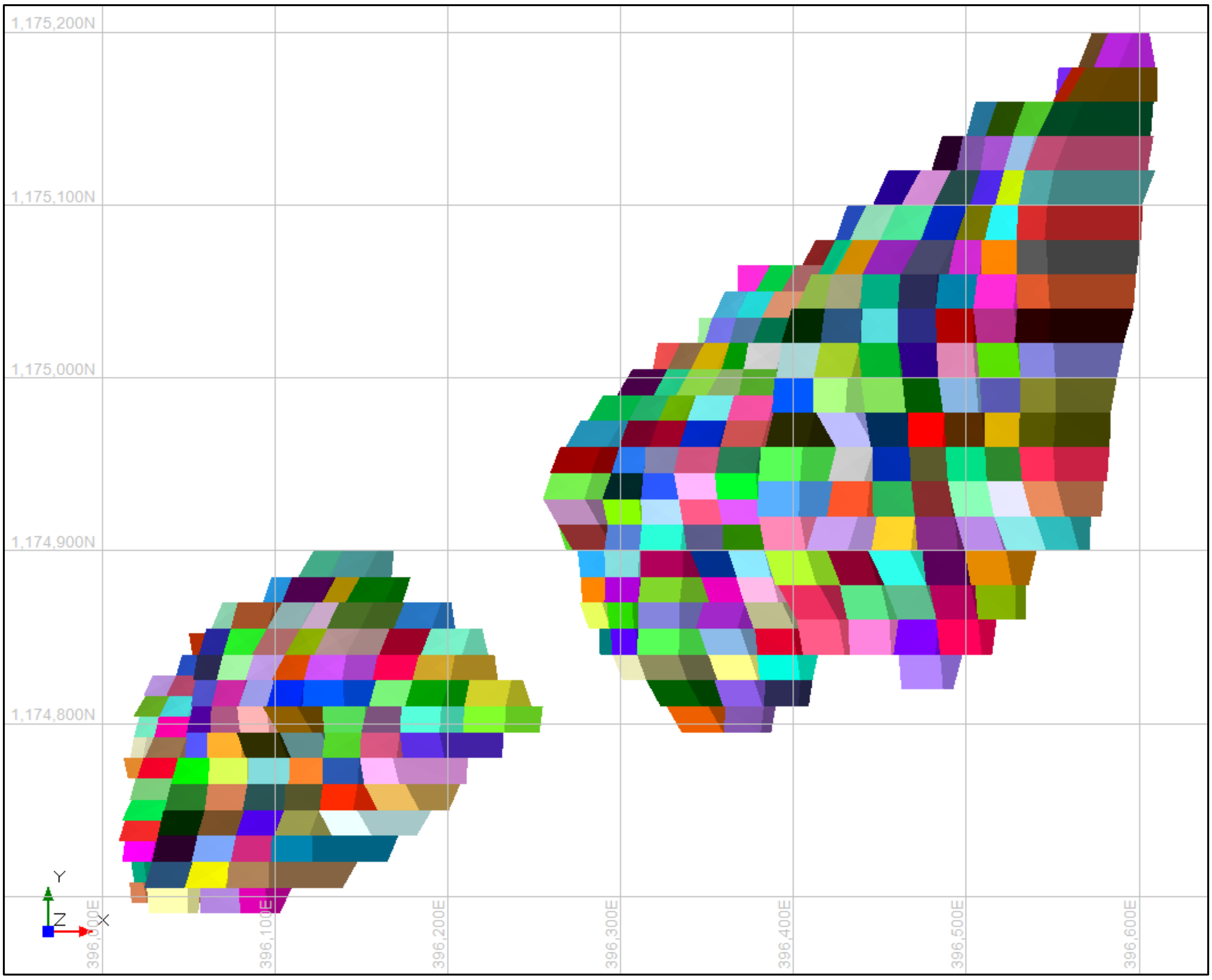

Figure 16.33: Stope Shapes Generated using Deswik.SO – Plan View

The HR was estimated by based on different level intervals of the stope shapes as shown in Table 16.9 and using three different methods, including:

- Calculation of HR based on the following formula with L being length and H being height:

$$\text{HR} = \frac{L \times H}{2 \times (L + H)}$$

- HR based on geotechnical graph analysis.
- HR determined with a 15% penalty where the graph analysis HR penalised by 15% due to limited geotechnical data available.

Table 16.9: Stope Shape Calculated Hydraulic Radius

Level mRL	Stope Dimensions (m)			Calculated HR		HR Graph Analysis		HR, 15% Penalty	
	HW Length	Level Height	BKS Length	HW	BKS	HW	BKS	HW	BKS
10160 – 10020	20	20	30	5.9	6.0	7.8	7.3	7.8	6.2
10020 – 9880	15	20	25	4.9	4.7	5.7	5.5	5.7	4.7
9880 – 9740	15	20	20	4.9	4.3	4.9	4.7	4.9	4.0
9740 – 9620	12	20	15	4.2	3.3	4.3	4.0	4.3	3.4

Figure 16.34 shows the stopes shapes displayed in the different geotechnical zones looking west. This figure includes inferred stopes (below 9,900RL) which are excluded from the Mineral Reserve and LOM production schedule.

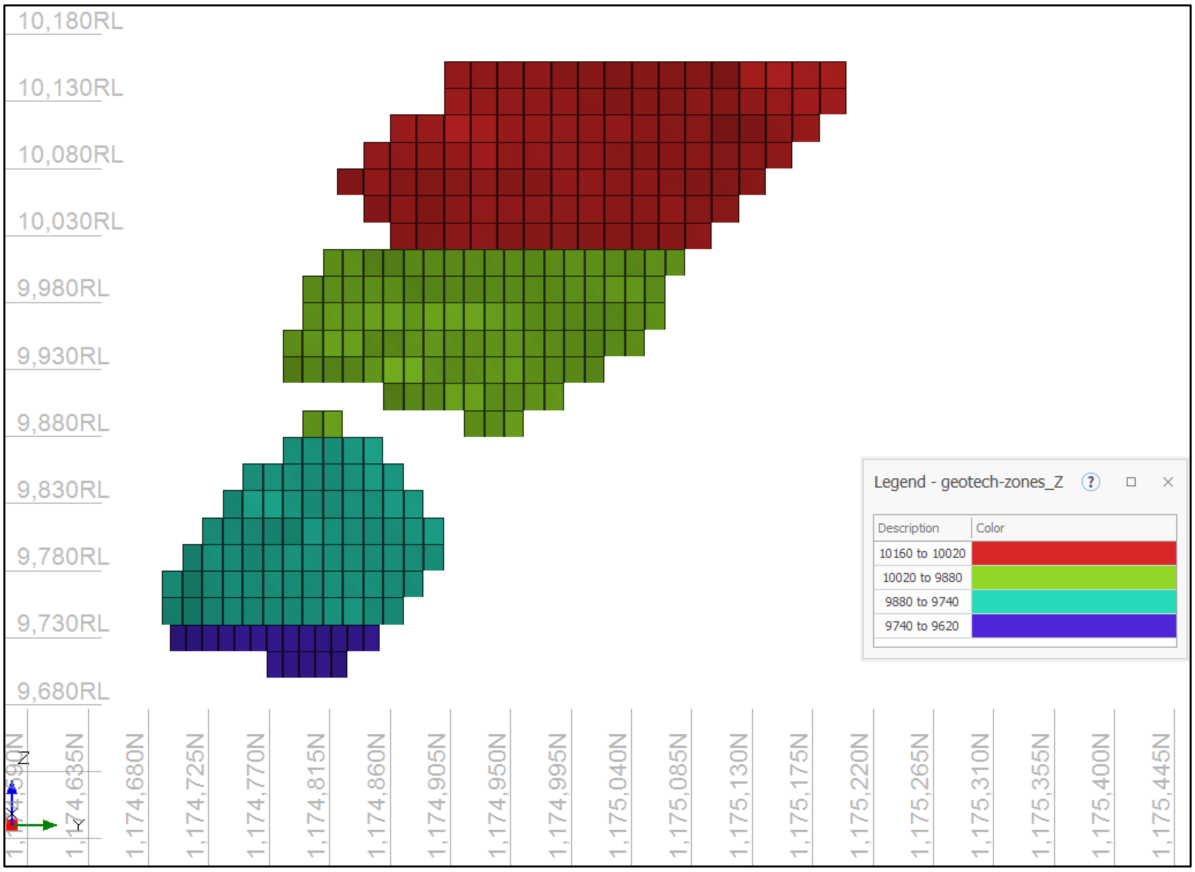

Figure 16.34: Stope Shapes Divided into the Different Geotechnical Zones – Looking West

Figure 16.35 below shows examples in plan view of the divided stopes on different level intervals.

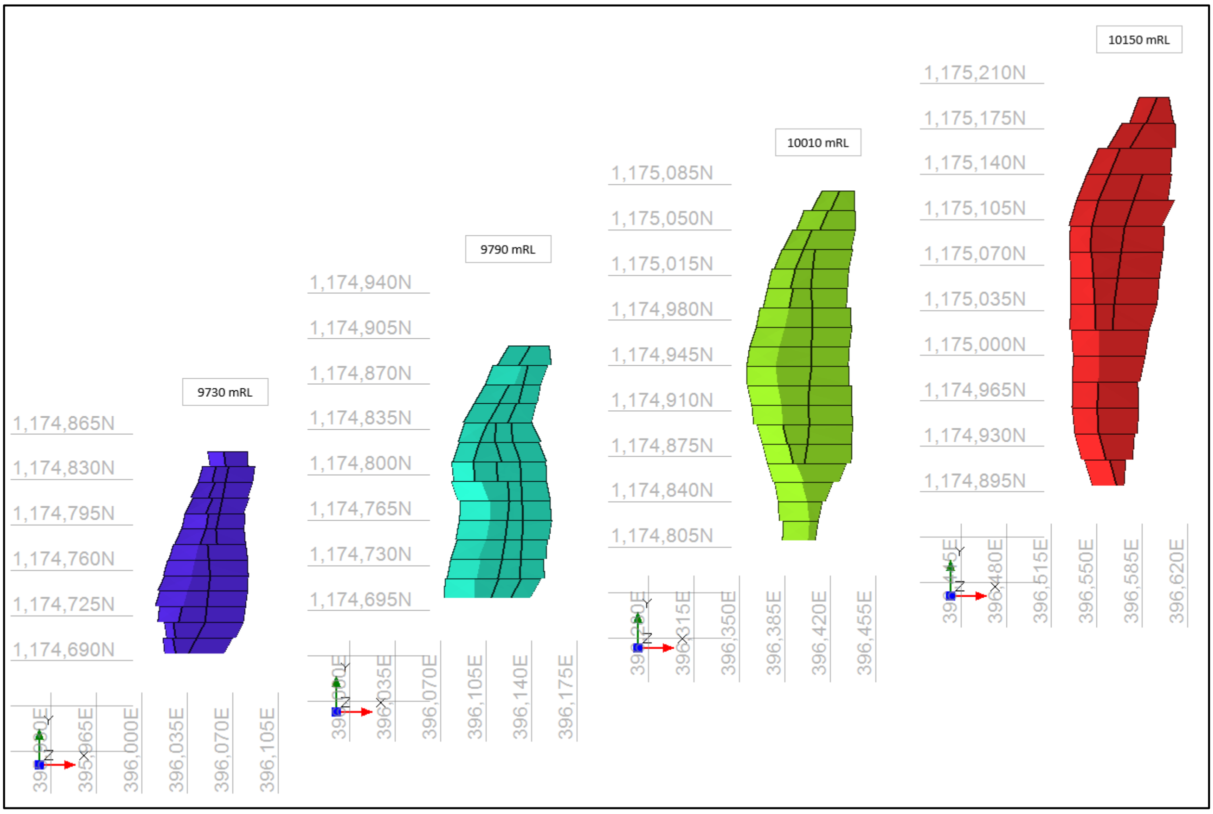

Figure 16.35: Examples of Split Stopes on Different Level Intervals – Plan View

The outcome of this process was the generation of the final stope shapes, which in turn informed the selection of the appropriate mining method based on stope geometry. Stopes with greater width in the transverse direction on the bottom level were classified for extraction using the TLHOS method. Conversely, stopes with greater length in the longitudinal direction were designated for LLHOS. This classification ensured alignment between stope geometry and the most efficient and geotechnically appropriate mining method.

Information used to inform this decision is displayed in Figure 16.36.

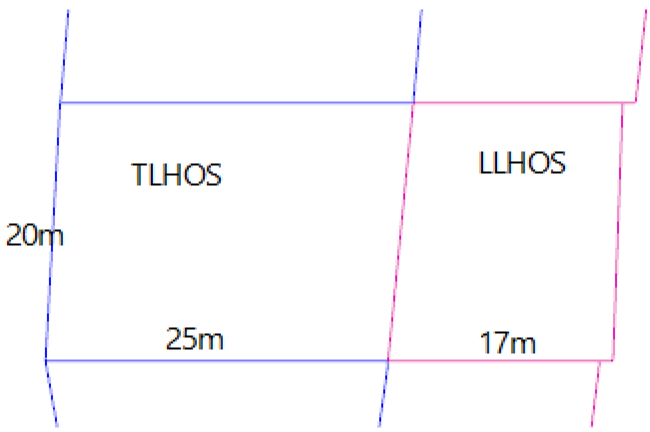

Figure 16.36: TLHOS vs LLHOS Decision Making – Plan View

Figure 16.37 displays the stope shapes by mining method after they have been divided, in plan view. This figure includes inferred stopes (below 9,900RL) which are excluded from the Mineral Reserve and LOM production schedule.

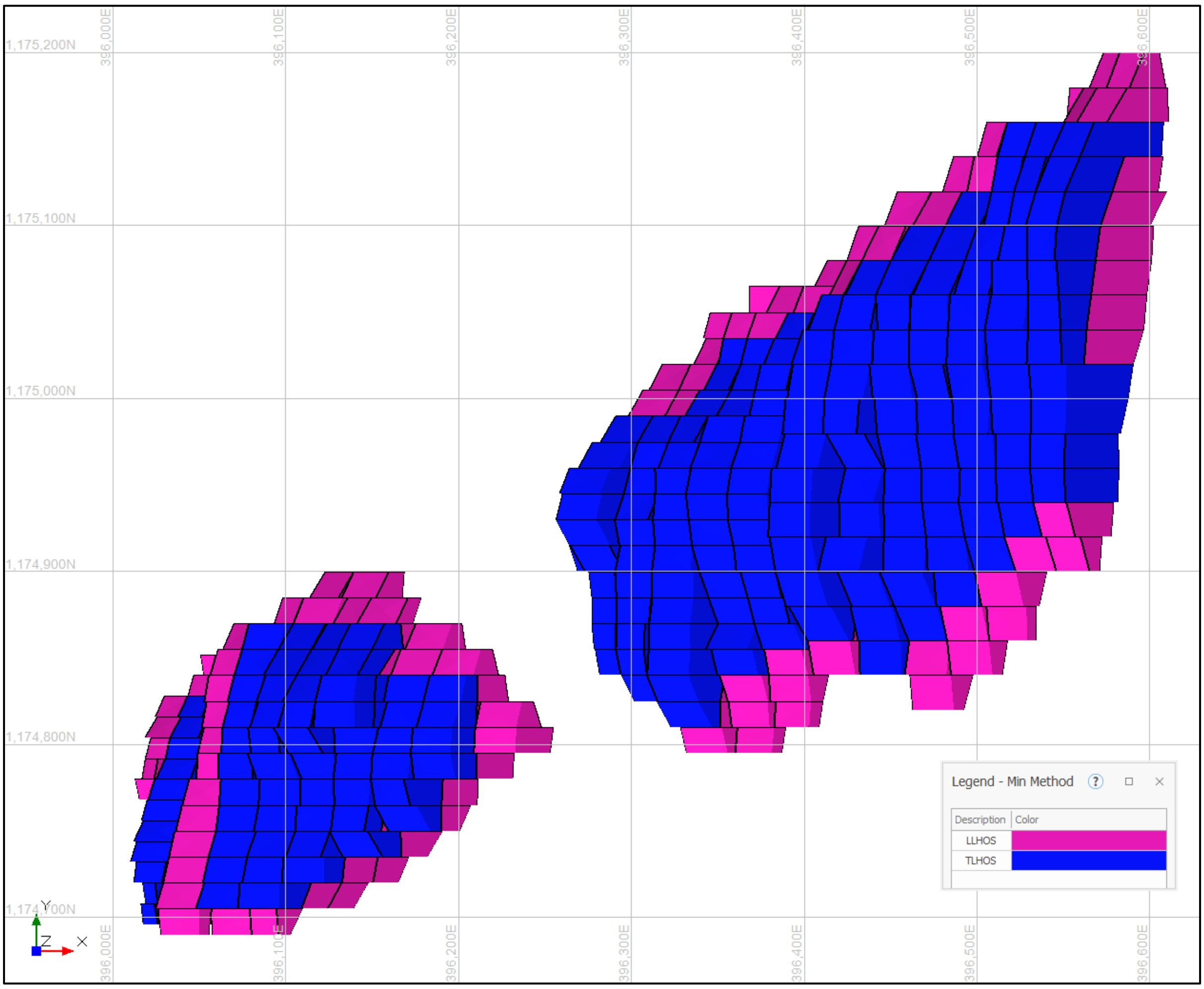

Figure 16.37: Stope Shapes Grouped Based on Mining Method After Splitting – Plan View

Figure 16.38 displays the stope shapes by mining method after they have been divided, looking east. This figure includes inferred stopes (below 9,900RL) which are excluded from the Mineral Reserve and LOM production schedule.

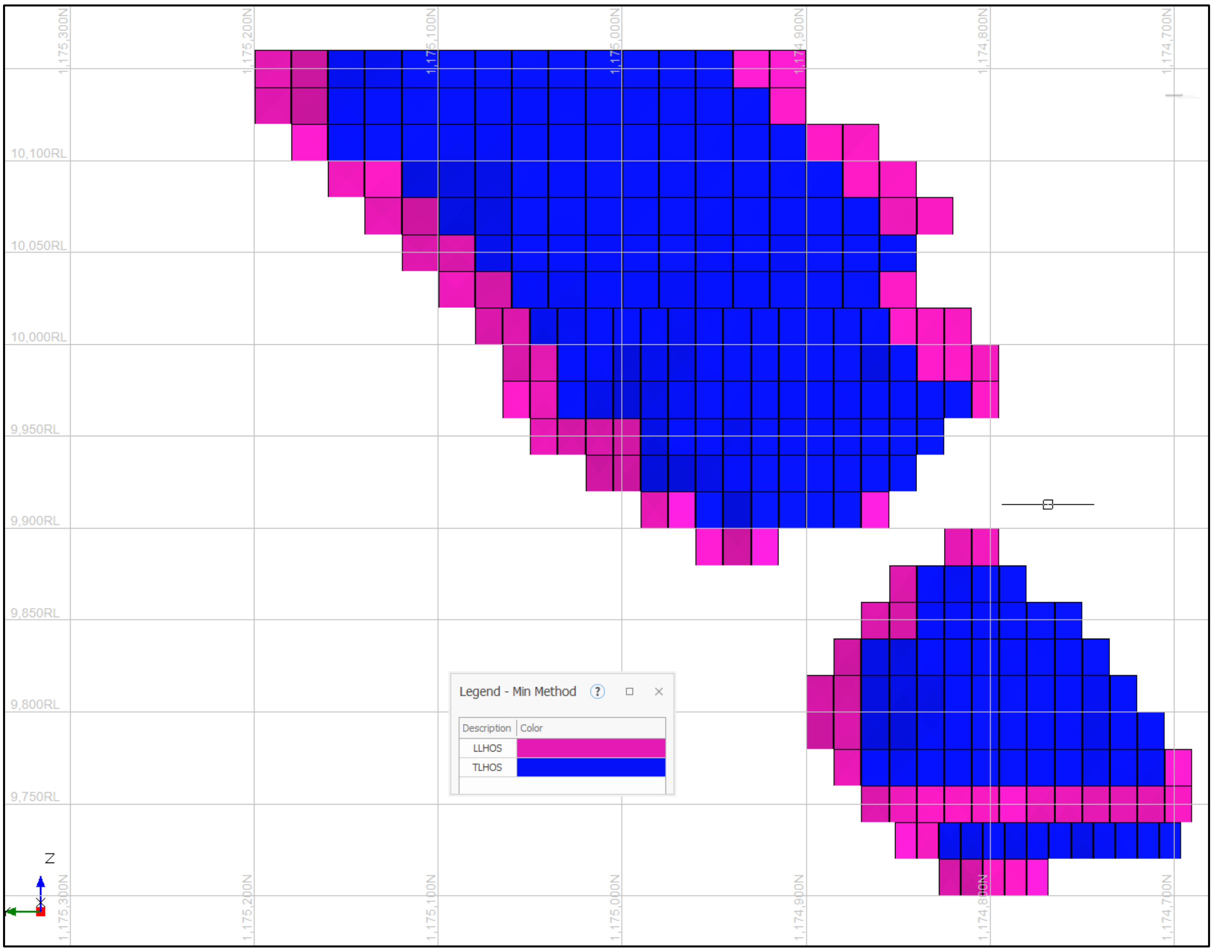

Figure 16.38: Stope Shapes Grouped by Mining Method After Splitting – Looking East

The final tonnes and grade after development drives were extracted from shapes and the number of stopes for each mining method are detailed in Table 16.10 below.

Table 16.10: Stopes Shapes Tonnes and Grade by Mining Method

Mining Method	Tonnes Mt	Au Grade g/t	Number of Stopes
Longitudinal long-hole open stoping	2.79	4.41	170
Transverse long-hole open stoping	3.85	5.12	163
Total	**6.64**	**4.82**	**333**

16.6 Mine Schedules

16.6.1 Pre-DFS Schedule Evaluation

The mining sequence and timeline for the DFS were established based on the outcomes of a post-PFS mining strategy evaluation. This evaluation was carried out prior to commencement of the DFS.

The updated modifying factors assessed as part of this evaluation included:

- Steepening pit walls as per updated geotechnical guidance following a review of the available information, but prior to completion of additional drilling and testing.
- Edge based dilution approach.
- A more efficient hybrid transverse/longitudinal underground mining method.
- Reduced dilution from underground mining.

A range of mill throughput rates from 3.5 Mtpa to 6.5 Mtpa were assessed along with increasing the underground production rate from 1.0 Mtpa to 1.4Mtpa.

Several pit optimisations were run and then shells selected to run indicative LOM schedules with and without an underground component. The key outcomes of this work were as follows:

- Maintaining the same 5.5 Mtpa throughput and 1.0 Mtpa underground production as the PFS but applying the other modifying factor changes increased NPV_{10} by approximately 35%.
- Increasing the underground production rate to 1.4 Mtpa significantly increased value by a further 10%, more importantly, it increased the cashflow over the crucial first three years by 25% (approximately US$100m) and provided a superior feed blend in the first year of processing.
- An open pit only approach generated almost the same NPV_{10} as the 5.5 Mtpa processing with 1.4 Mtpa underground scenario, however, the mill feed blend requirements were not meet through the early years of the schedule and very high material movements and vertical advance rates were required to maintain mill feed.
- Decreasing the throughput rate to 4.5 Mtpa only decreased the NPV_{10} by 3% but crucially generated the same cashflow over the first three years of production for a lower capital outlay (i.e. quicker payback). In addition, the lower production requirement derisked mine production, meaning the open pit had greater capacity to fulfil any shortfall from the underground. This was identified as the best scenario to carry forward into the DFS.

In addition to these changes, the evaluation also determined that:

- Delaying the start of the underground, and its high-grade fresh ore, immediately degrades value and makes it impossible to meet the required plant feed blend over the first two to three years.
- Three stages appeared to be the optimum for the NEB pit.

These combined outcomes informed the following scheduling approach for the DFS:

- Monthly underground schedule was developed in advance of the open pit schedule. This schedule showed that a consistent and sustainable fresh underground ore feed of more than 25% mill feed was available following 15 months of development and this was selected as the first month of processing and production. The underground schedule continues for a total duration from the commencement of underground development of 120 months (10 years).

- GBE pit is mined prior to the commencement of underground development, allowing one month to set-up the base of the pit to commence the underground portal development.
- NEB pit requires approximately three months of pre-strip.
- BC is a very profitable pit, containing 10% of the total open pit ounces for 5% of the total material. However, it is more geologically complex than the NEB pit and is located in the Bankan Creek water course relatively close to the Niger River. Therefore, due to the additional cost and complexity of mining this pit it was determined that it should be mined at the end of the mine life to mitigate risk.

16.6.2 Underground Methodology and Parameters

Several key mining parameters have been incorporated into the mine schedule to ensure a realistic representation of underground production and to support reliable Mineral Reserve estimation. The most critical of these parameters are:

- Dilution.
- Mining recovery.

These modifying factors reflect the practical challenges encountered during underground mining operations, including equipment precision, sequencing constraints, variable ground conditions, and the limitations associated with the selected mining method.

16.6.2.1 Dilution

Dilution refers to the unintended inclusion of waste or low-grade material into the ore stream during stope extraction. While it does not result in material loss, it increases the total tonnage mined without a corresponding increase in contained metal, lowering the average grade of the material processed.

Dilution is influenced by several operational and geological factors, including:

- Drill hole deviation.
- Hangingwall sloughing or failure.
- Overbreak during bogging.
- Paste fill intrusion.

In this project, a dilution factor of 15% has been applied across all stopes. This factor reflects expected overbreak and other dilution sources, with the added material assigned a zero grade in the schedule.

16.6.2.2 Mining Recovery

Mining recovery address the portion of in-situ ore that cannot be extracted due to technical or safety limitations. These parameters reflect:

- Unrecoverable material left behind due to equipment access or blasting limitations.
- Geotechnical restrictions, such as reduced stope sizes or exclusion zones.
- Pillar retention, where some stopes are not mined to maintain ground stability.

Recovery factors will vary depending on the stope type with the following adopted for the DFS:

- Regular stopes are assigned higher recovery factors, reflecting expected performance in standard operating conditions.
- Pillar stopes have lower recovery factors due to the higher rock stress conditions present during their extraction. As these stopes are mined later in the sequence, after adjacent voids have been backfilled and cured, stress redistribution can compromise ground stability, leading to reduced recovery or increased ore loss.

The recovery factors used in the underground mine scheduling are summarised in Table 16.11.

Table 16.11: Bankan Mining Recovery Factors

Design Type	Recovery (%)
Development	100
Stoping	85
Sill Pillar Stope	75
Crown Pillar	75

Figure 16.39 details the location the crown pillar, sill pillars and stopes in cross-section view. This figure includes inferred stopes (below 9,900RL) which are excluded from the Mineral Reserve and LOM production schedule.

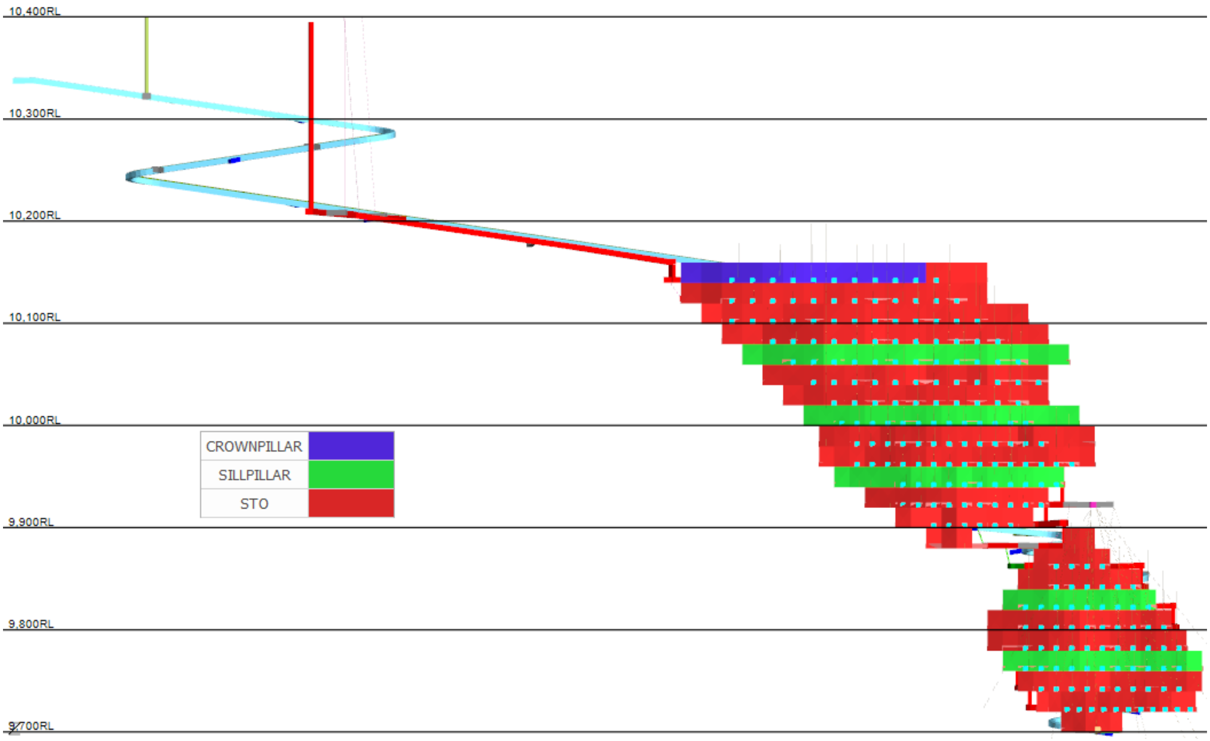

Figure 16.39: Stopes, Sill Pillars and Crown Pillars – Cross Section View

16.6.3 Underground Schedule

The underground mining schedule was developed with a clear objective: to access high-grade ore as early as possible to supplement the mill feed with fresh rock. This strategy was underpinned by both metallurgical and economic drivers. Fresh rock maintains comminution circuit performance by providing rock for the SAG mill and due to the low clay content, improves filtration performance. Concurrently, early exposure to high-grade zones maximises gold production in the initial years of operation, enhancing project economics.

Because of this approach, the process plant was designed to treat a blend comprising a minimum of 25% fresh ore, equivalent to approximately 1.4 Mtpa. Achieving this target required accelerating underground development through the establishment of two portals. This dual-portal arrangement enabled the concurrent development of the main haulage drive and the ventilation airway, thereby expediting access to the orebody without dependency on the completion of the long-term return air raise from surface.

The mine infrastructure schedule was also closely aligned with the overall production strategy. Given the significant capital requirements for underground ventilation and paste backfill systems, these components were critically assessed and strategically deferred where feasible. This approach reduced upfront capital expenditure without compromising the timely commencement of ore production or operational safety.

Stope sequencing was another key factor in delivering early gold production. Stopes were grouped into three-level blocks and scheduled in a bottom-up sequence, a configuration that supports long-term stability and efficient paste backfilling. However, this sequence can delay initial ore production, so to mitigate this, a modified top-down sequence with delayed backfilling was adopted for the first three levels. In this configuration, stopes are mined as double lifts, complying with hydraulic radius limits, but with an extended backfill cycle. Geotechnical risks associated with increased hanging wall exposure are mitigated using cablebolts, although these risks remain operational and will require reassessment as geological and geotechnical knowledge improves.

This hybrid sequencing strategy ensured that development remained ahead of production requirements and facilitated a seamless transition into the long-term bottom-up mining approach. The outcome is a balanced design that optimises geotechnical stability, fill efficiency, and early gold production.

16.6.3.1 Stope Sequencing

Stope extraction throughout the mine follows a fundamentally bottom-up approach, with sequencing adapted according to stope orientation:

- Transverse stopes are mined using a centre-out sequence, beginning from the centre of each panel and progressing outward. These stopes are classified as primary and secondary, with primary stopes mined first to provide support for adjacent secondary stopes. The primary stope is filled prior to extraction of the secondary stope.
- Longitudinal stopes follow an extremities-to-centre sequence. This approach supports geotechnical stability and facilitates efficient backfill placement and logistics.

Figure 16.40 below shows the stope panels and sequence looking east. This figure includes inferred stopes (below 9,900RL) which are excluded from the Mineral Reserve and LOM production schedule.

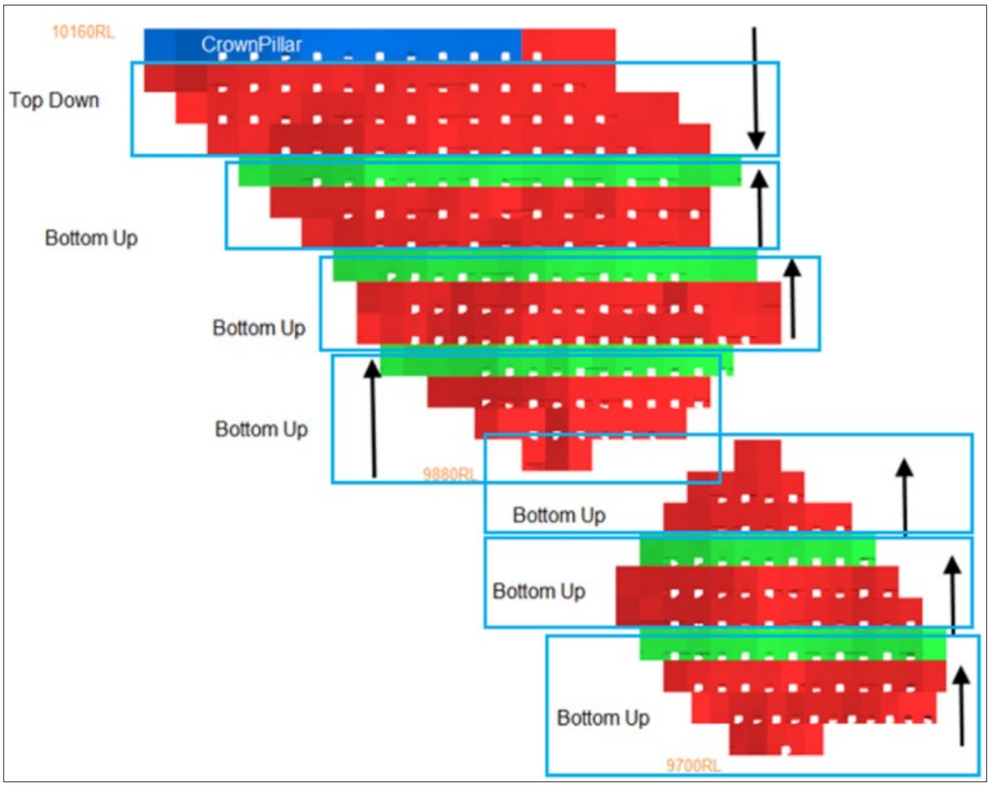

Figure 16.40: Stoping Sequence in Panels – Looking East

16.6.3.2 Development and Support Requirements

Key infrastructure elements, such as the return airways and escapeways, are established prior to the commencement of mining on each level, to ensure compliance with ventilation and safety requirements.

Sill pillar levels, which are essential for maintaining structural integrity between mining blocks, are developed with enhanced ground support measures. Extraction of these levels is deferred until all adjacent stopes have been backfilled and allowed sufficient curing time, thereby ensuring stability during extraction.

16.6.3.3 Paste Fill and Rest Time

Most stopes are backfilled with paste fill and require a 28-day curing period to achieve the specified strength before mining activities can continue in adjacent stopes. An exception is made for the final stopes mined within a sill pillar, which do not require backfilling. At the time of their extraction, all surrounding stopes will have already been mined and filled, eliminating the requirement for additional support.

Figure 16.41 below shows the stopes divided into primary and secondary panels looing east. This figure includes inferred stopes (below 9,900RL) which are excluded from the Mineral Reserve and LOM production schedule.

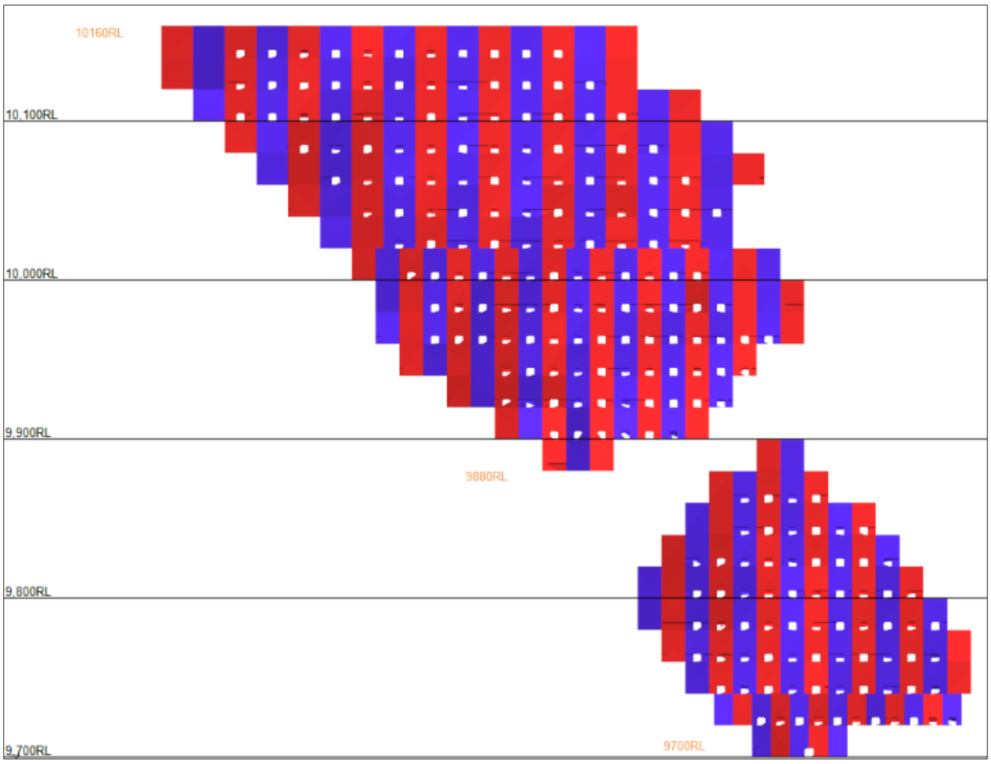

Figure 16.41: Stopes Divided by Primary and Secondary – Looking East

Within the overall scheduling strategy outline, and seeking to maintain high utilisation rates of equipment, the schedule is driven by:

- Task and resource rates as shown on Table 16.12 and Table 16.13.
- Ventilation requirements and fleet constraints.

Table 16.12: Scheduling Parameters – Resource Rates

Activity	Units	Rate
Jumbo	m/month	250
Production drill	m/day	280
Bogger	t/day	1,200
RC drill	m/day	200

Table 16.13: Scheduling Parameters – Task Rates

Activity	Units	Rate
Portal (<80m of Decline)	m/month	80
Decline/FAD (80 – 200m Decline)	m/month	120
Decline/FAD (200m – Production Level)	m/month	160
Decline after First Production Level	m/month	80
Other Waste Development	m/month	50 – 60

Activity	Units	Rate
Ore Development	m/month	50
Refuge Chamber Cuddy	m/month	20
Surface Service Holes	m/d	60
Paste Fill / Other Service Holes	m/d	200
Wall Build	wall/day	1
Paste Fill	m^3/h	75
Paste Rest	days	28
Production Bogging	t/d	1200
Production Drilling	m/d	280
RC Drill	m/d	200
Raise Borer	m/d	4

16.6.3.4 Mining Physicals

The underground mining physicals presented in this section outline the key metrics and production quantities associated with the extraction of fresh, high-grade ore from the NEB underground deposit. Physicals include development metres, production tonnes, paste fill requirements, and associated infrastructure activities. These are aligned with the bottom-up mining sequence, paste backfilling strategy, and overall mill feed constraints. The underground schedule complements open pit mining to ensure that both tonnage and grade targets are met throughout the life of mine.

Table 16.14 details the planned annual development metres by drive type and cost allocation, providing an overview of the development of the underground mine as it progresses over the life of mine.

Vertical development encompasses all raises necessary to support ongoing mine operations and represents a critical component of the LOM plan with the vertical development metres required for ventilation and escapeways detailed in

Table 16.15. Table 16.16 details the drilling metres required over the life of the mine for the different types of service holes and Table 16.17 shows the projected annual paste fill requirements over the life of mine.

Underground production mining physicals are shown in Table 16.18.

Table 16.14: Underground Development Schedule

Development Drive Type	Units	Total	Pre-Prod.	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Access & Safety															
Decline	m	3,335	1,978	747	499	111									
Access Drives	m	546	150	203	155	37									
Escape Way Drive	m	450	43	67	214	121	5								
Refuge Chamber	m	276	70	93	73	39									
Ventilation															
Fresh Air Drive	m	1,954	1,600	102	140	112									
Return air Drive	m	2,023	812	316	592	263	40								
Level Drives															
Footwall Drive	m	2,592	533	584	1,124	334	18								
Hanging wall Drive	m	1,969	2	543	331	451	642								
Longitudinal ore drive	m	2,856	67	719	807	846	418								
Ore drive	m	9,902	477	2,560	2,395	3,021	1,449								
Slot Drive	m	187				71	116								
Dewatering															
Pump Station Drive	m	45	23		23										
Sump	m	395	209	93	68	24									
Other															
Stockpile	m	1,265	504	209	346	153	52								
Exploration Drill Drive	m	0													

Development Drive Type	Units	Total	Pre-Prod.	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Total	**m**	**27,796**	**6,469**	**6,237**	**6,767**	**5,582**	**2,741**								
Capital	**m**	**12,112**	**5,839**	**2,325**	**2,971**	**879**	**98**								
Operating	**m**	**15,683**	**624**	**3,904**	**3,786**	**4,718**	**2,651**								

Table 16.15: Vertical Development Metres

Vertical Development Type	Units	Total	Pre-Prod.	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Escapeway Raises	m	199		61	93	31	14								
Return Air Raise to surface	m	182	182												
Fresh Air Raise to surface	m	74		74											
UG Fresh Air Raise	m	200	13	51	68	68									
UG Return Air Raise	m	360		80	153	97	31								

Table 16.16: Service Holes Schedule

Service Hole Type	Units	Total	Pre-Prod.	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Primary Dewater Line	m	289	226		63										
Paste Fill Service Holes	m	754	376	81	214	58	25								
Sump Drain Holes	m	242	25	75	118	24									
Electrical Service Holes	m	438	217	73	116	32									

Table 16.17: Paste Fill Requirements

Paste Fill	Units	Total	Pre-Prod.	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Paste Volume	'000 m^3	**2,300**		299	380	384	426	481	330						
Paste Tonnes	'000 t	**4,280**		556	707	714	793	896	615						
Binder	kg	**263,319**		36,363	56,910	58,314	46,385	38,745	26,602						
Tailings Tonnes	'000 t	**4,429**		576	731	739	820	927	636						
Water	'000 m^3	**15,309**		1,990	2,528	2,553	2,835	3,203	2,199						
Walls	No.	**642**		79	86	101	119	154	103						

Table 16.18: Underground Ore Production

Source	Units	Total	Pre-Prod.	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Stoping	kt	**6,447.3**		844.1	1,094.0	1,059.6	1,173.8	1,317.6	958.2						
	g/t	**4.08**		4.43	4.02	3.99	3.74	4.28	4.06						
	koz	**845.2**		120.2	141.6	136.0	141.0	181.2	125.2						
Development	kt	**1,256.2**	89.9	321.1	310.0	344.4	190.7								
	g/t	**3.15**	2.10	3.30	2.91	3.07	3.96								
	koz	**127.3**	6.1	34.0	29.0	34.0	24.3								
Total	**kt**	**7,703.5**	89.9	1,165.2	1,404.0	1,404.0	1,364.5	1,317.6	958.2						
	g/t	**3.93**	2.10	4.12	3.78	3.77	3.77	4.28	4.06						
	koz	**972.6**	6.1	154.2	170.5	170.0	165.3	181.2	125.2						

16.6.4 Open Pit Methodology and Parameters

The aim of the open pit scheduling process was to generate a practical, realistically achievable schedule that maximises project value within the given constraints and objectives and provides sufficient material to ensure the mill operates at full capacity when including the underground mill feed. Ideally, the schedule will satisfy as many of the following criteria as possible:

- Satisfy mill feed requirements.
- Incorporate a ramp-up of operations for mining and processing.
- Minimise any pre-strip and delay waste mining where possible.
- Limit vertical rate of advance to 12, 5 m benches per year (60 m vertical per year).
- Mine from a realistic number of working areas per period.
- Maintain achievable material movements per area.
- Maximise head grade wherever possible by stockpiling lower grade ore.

The open pits were scheduled using Hexagon MinePlan Schedule Optimizer™ (MPSO), an advanced mine scheduling tool that optimizes cut-off grades and phase sequencing while considering different operational constraints.

The underground mining schedule was developed prior to the open pit schedule, with the focus on reaching the 1.4 Mtpa target production rate for as long as possible. The open pit schedule was subsequently developed on the basis that the underground ore would always be the primary feed due to its high grade and the open pit would supply the process plant shortfall while maximising value and overall gold production.

16.6.4.1 Open Pit Scheduling Periods

The open pit scheduling periods comprised:

- Monthly for the 2-year pre-production period (i.e. Year -2 and Year -1), consisting of mining of GBE and pre-stripping of NEB.
- Monthly for Year 1 and 2 of processing.
- Quarterly for Year 3 to Year 5 of processing.
- Annually for Year 6 to the end of the mine life.

16.6.4.2 Processing Targets and Constraints

The processing plant design is based on the following scheduling targets and constraints:

- Ramp up in the processing to the nameplate equivalent feed rate of 4.5 Mtpa defined as:
  - Month 1, 60% of nameplate throughput.
  - Month 2, 80% of nameplate throughput.
  - Month 3, 90% of nameplate throughput.
  - Month 4, 95% of nameplate throughput.

- Month 5 onwards at 100% of nameplate throughput.

- Maintain a minimum of 25% fresh material in the plant feed, driven by comminution circuit requirements.
- Limit the maximum saprolite clay proportion in the plant feed to 50%, driven by rheology and tailings filtration limitations.
- Limit gold production to the equivalent of a maximum of 375,000 recovered oz per year, driven by the sizing of the gold recovery circuits and gold room.
- Ensure fresh proportion does not reduce by more than 5% month on month, driven by limitations on the flexibility of the comminution circuit.

16.6.5 Integrated Open Pit and Underground Schedule

The overall integrated Project LOM production schedule, based on Mineral Reserves only, is presented by source in Figure 16.42. It shows GBE pit being mined and completed in Year -2, and then underground development continuing through Year -1. Year -1 also includes mining contractor mobilisation and pre-stripping of NEB pit. Mining rates then steadily increase from approximately 7.6 Mtpa in Year 1 to a peak of 14.1 Mtpa to14.5 Mtpa through Year 4 to Year 6. Total material movement then take a stepwise reduction to 10.5 Mtpa in Year 8 and Year 9, after which it steadily decreases over the remainder of the life of mine

The process plant feed is presented by lithology in Figure 16.43, which shows that:

- The nominal mill feed limit of 4.5 Mtpa is met in all years inclusive of the ramp-up in the first year.
- The lithological blend requirements outlined above are achieved on an annual basis. These criteria are also met in the shorter period increments of the underlying schedule.
- Stockpiling successfully keeps the head grade elevated during the earlier periods with:
  - Feed grade of approximately 2.0 g/t for the first three years followed by a slight reduction to an average of 1.78g/t for years four to six.
  - Processing of lower grade feed, averaging 1.17 g/t from years seven to nine, coinciding with the loss of high-grade feed from the underground.
  - Feed grade then increases to over 2 g/t for the remainder of the LOM, years 10, 11 and 12 (partial) as the high-grade fresh material at the base of the NEB pit and material from BC pit is mined.

The mill feed by source is presented in Figure 16.44 in terms of tonnes and Figure 16.45 in terms of contained gold.

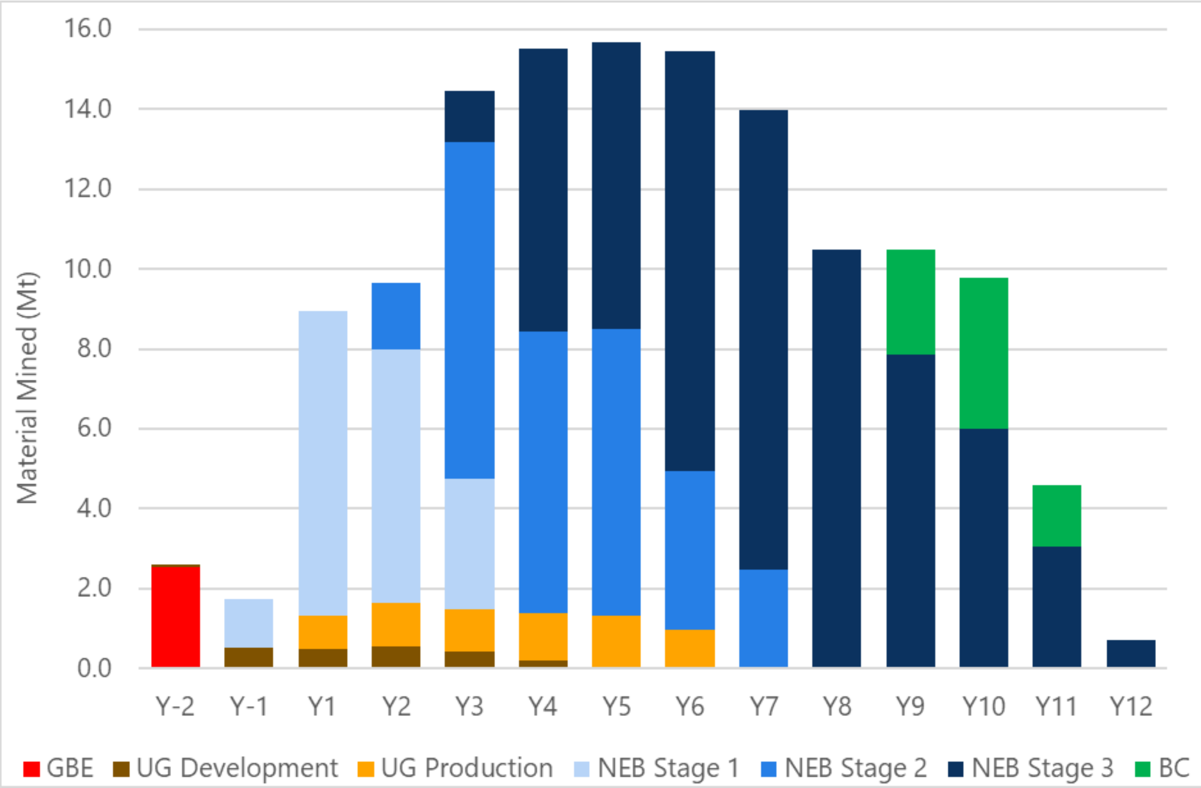

Figure 16.42: Project LOM Schedule – Total Material Mined

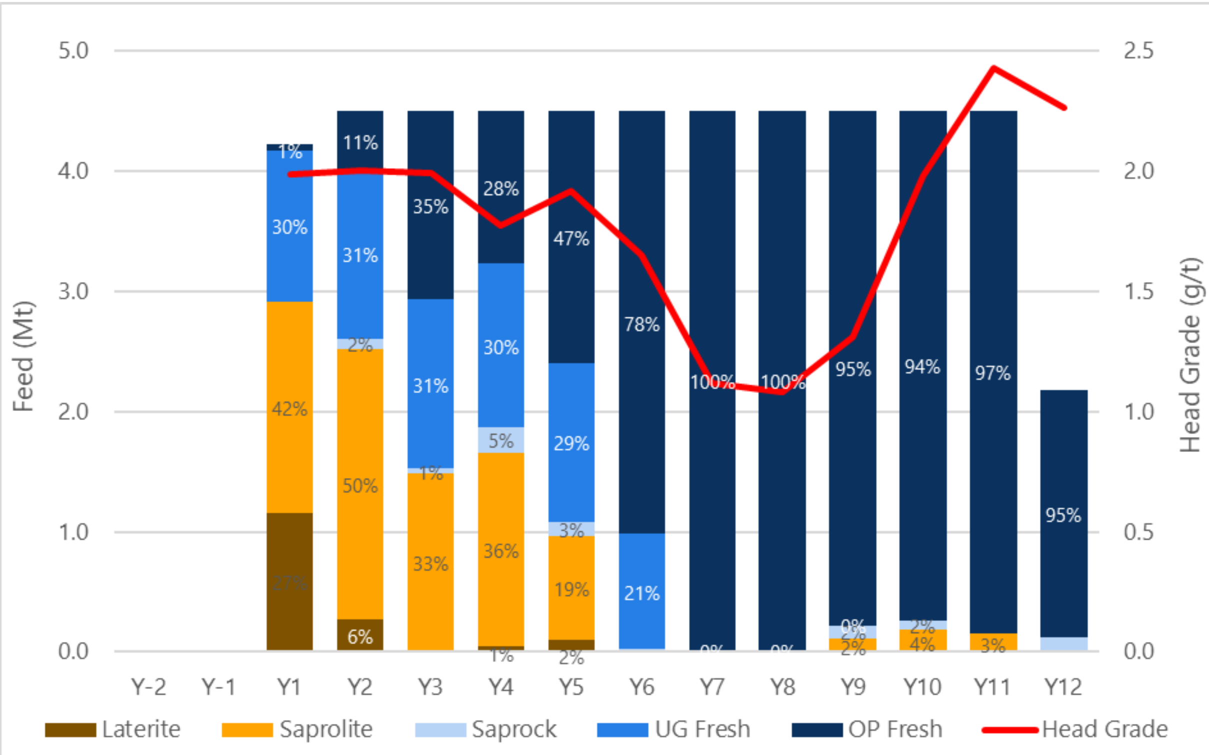

Figure 16.43: Project LOM Schedule – Mill Feed by Lithology and Grade

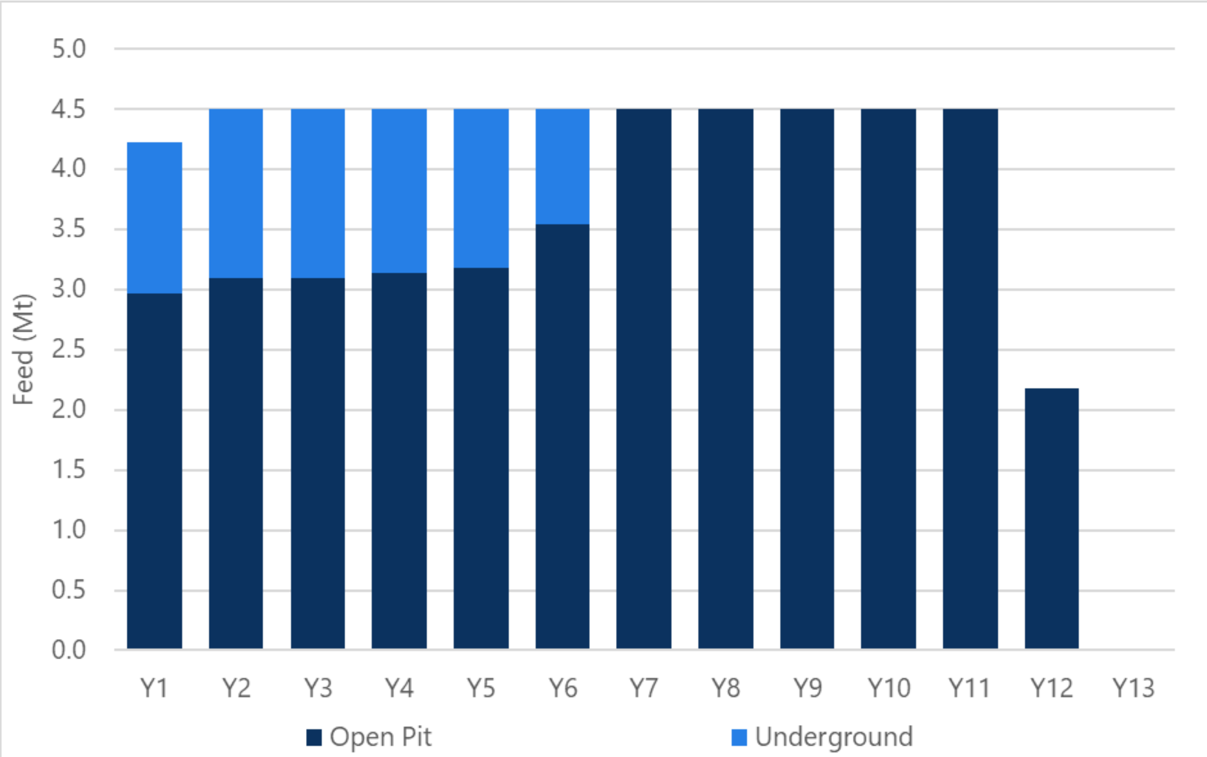

Figure 16.44: Project LOM Schedule – Mill Feed Tonnes by Source

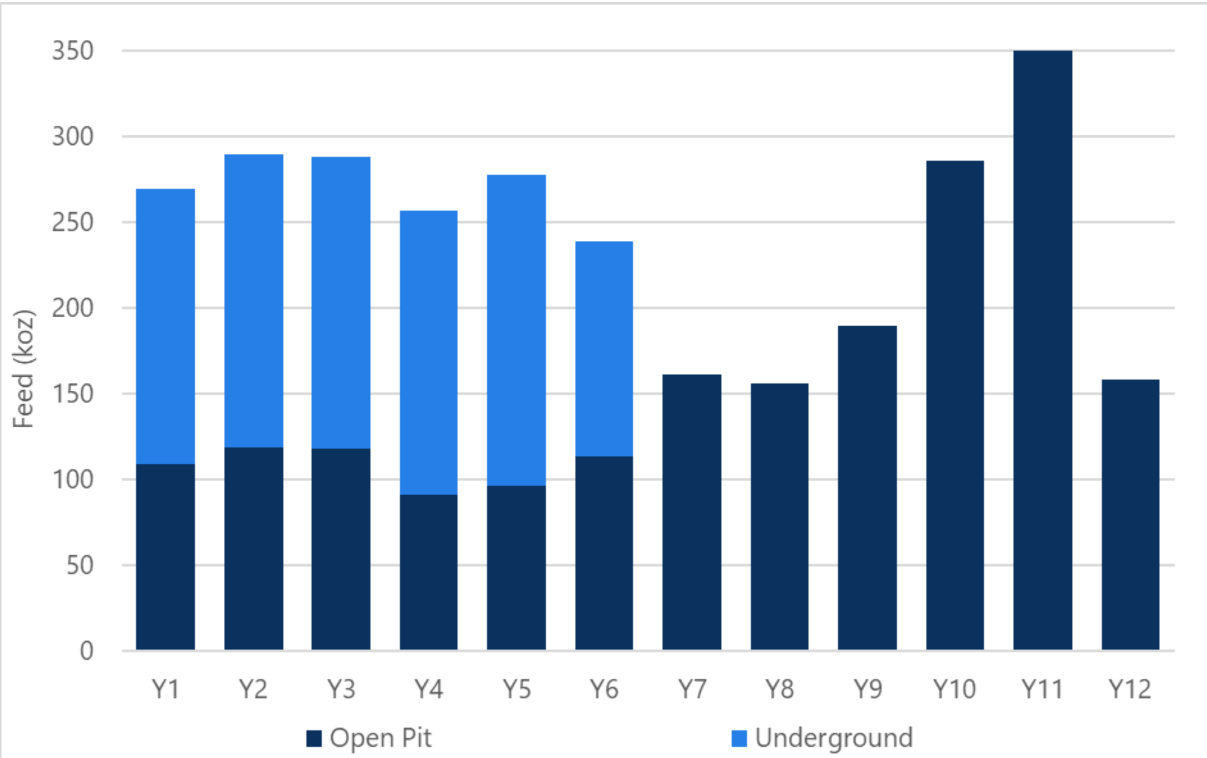

Figure 16.45: Project LOM Schedule – Mill Feed Contained Gold by Source

Figure 16.46 provides the gold production by year. Production average 251 koz/a over the first six years of production before dropping to an average of 156 koz/a for the following three years before a peaking at an average of 295 koz/a of the last two years of full production (Year 10 and 11).

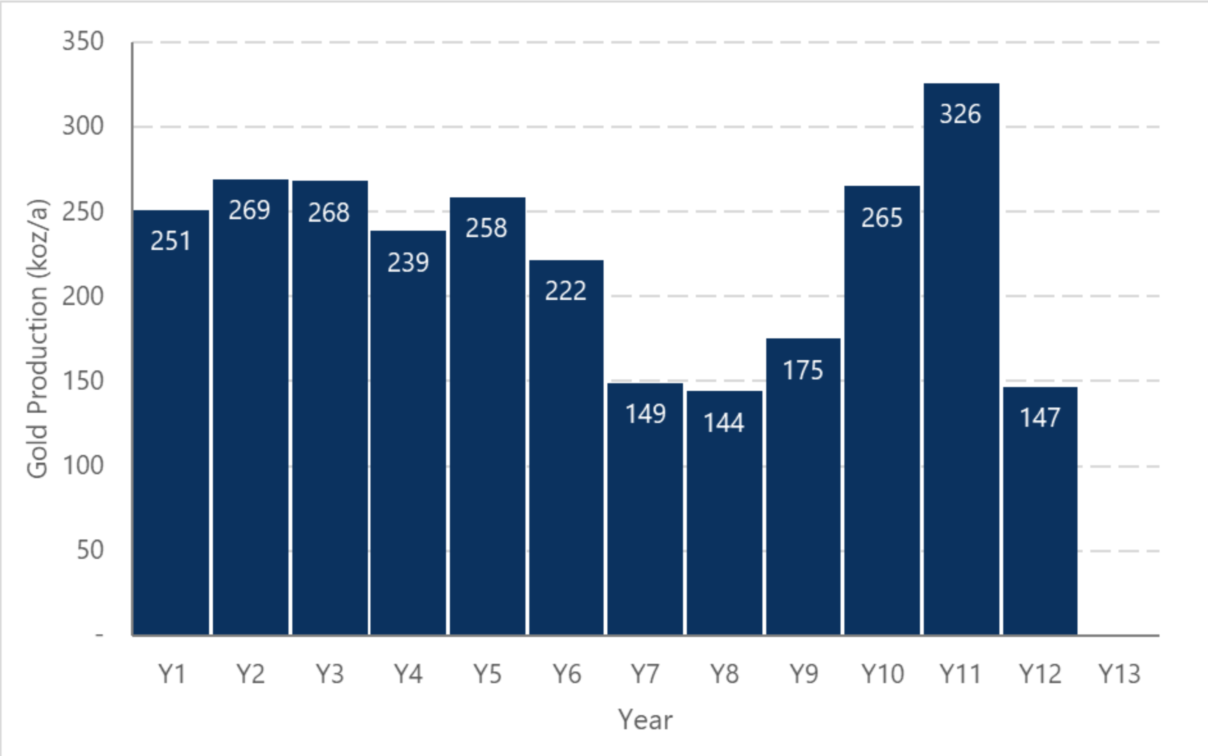

Figure 16.46: Project LOM Schedule – Mill Feed Recovered Gold

The long-term stockpile closing balances are presented in Figure 16.47, showing:

- The first third of the mine life has predominantly saprolite clay material stockpiled, both high-grade and low-grade, to maintain the lithological blend requirements. This material accounts for 50% to 80% of total stocks over this period.
- For the latter two thirds of the schedule, the stockpiles are predominantly low-grade fresh material

The Project LOM schedule is provided in Table 16.19, with a more detailed breakdown of mining by the individual pit stages provided in Table 16.20. The mill feed and production schedule is provided in Table 16.21.

Vertical rate of advance is maintained below 60 m per year for the duration of the schedule which is well within acceptable limits.

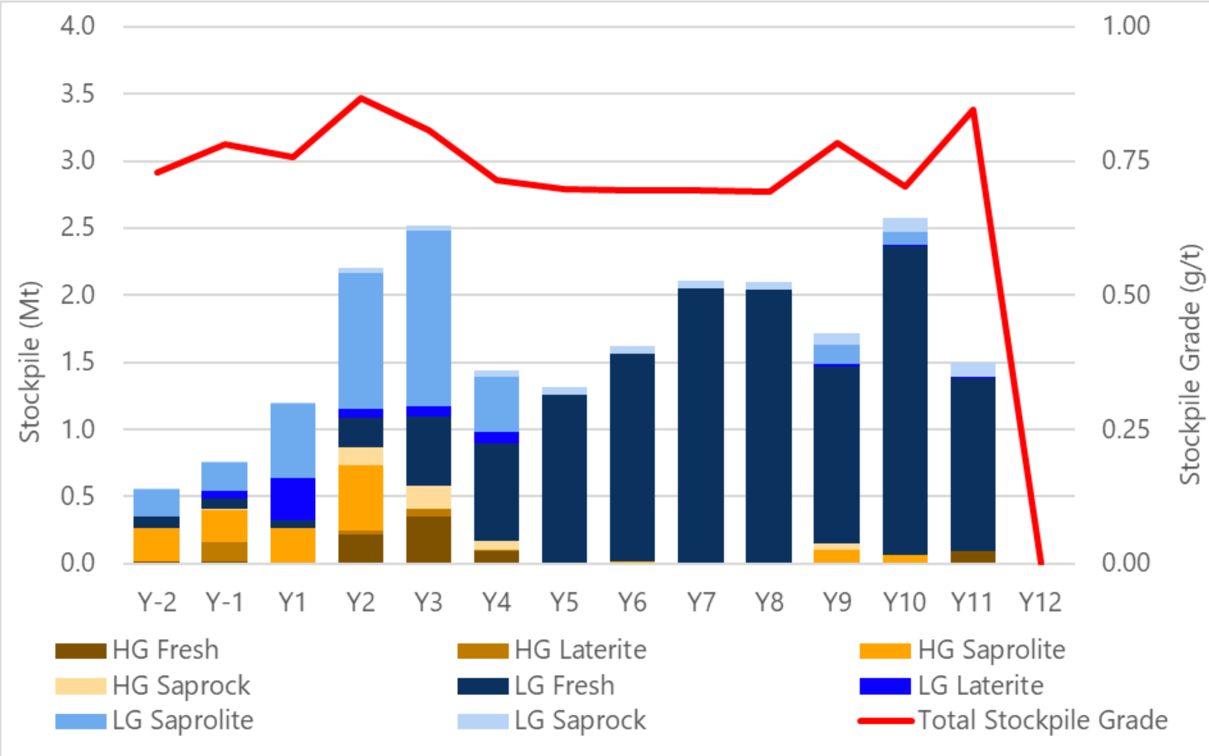

Figure 16.47: Project LOM Schedule – Stockpile Balances by Lithology

Table 16.19: Project LOM Schedule – Total Mined Tonnes

Area	Unit	Total	Pre-Prod.	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Underground Mining																
Development	Mt	2.19	0.04	0.52	0.48	0.54	0.42	0.19								
Production	Mt	6.45			0.84	1.09	1.06	1.17	1.32							
Underground Total	Mt	7.69	0.04	0.52	1.33	1.63	1.48	1.37	1.32							
Open Pit Mining																
GBE	Mt	2.55	2.55													
NEB Stage 1	Mt	18.45	1.21	7.63	6.35	3.26										
NEB Stage 2	Mt	30.83			1.66	8.45	7.07	7.18	3.99	2.47						
NEB Stage 3	Mt	65.61				1.28	7.07	7.18	10.49	11.51	10.49	7.86	5.99	3.05	0.70	
BC	Mt	7.97										2.62	3.80	1.55		
Open Pit Total	Mt	125.41	3.76	7.63	8.02	12.98	14.14	14.36	14.48	13.98	10.49	10.49	9.79	4.60	0.70	
Total Mining																
Total	**Mt**	**133.10**	**3.80**	**8.15**	**9.35**	**14.61**	**15.62**	**15.73**	**15.80**	**13.98**	**10.49**	**10.49**	**9.79**	**4.60**	**0.70**	

Table 16.20: Project LOM Schedule – Material Movement by Mining Stage

Location	Unit	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
GBE Pit																
Waste Mined	Mt	2.0	2.0													
Ore Mined	Mt	0.6	0.6													
Strip Ratio	-	3.5	3.5													

Location	Unit	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Ore Grade	g/t	0.73	0.73													
Contained Gold	koz	13.1	13.1													
NEB Pit																
Stage 1																
Waste Mined	Mt	8.7		1.0	4.2	2.3	1.1									
Ore Mined	Mt	9.7		0.2	3.4	4.0	2.1									
Strip Ratio	-	0.9		5.2	1.2	0.6	0.5									
Ore Grade	g/t	1.16		0.93	1.09	1.15	1.32									
Contained Gold	koz	363.1		5.8	119.7	147.9	89.7									
Stage 2																
Waste Mined	Mt	22.0				1.6	7.2	5.0	5.1	2.2	0.8					
Ore Mined	Mt	8.8				0.1	1.3	2.0	2.0	1.8	1.6					
Strip Ratio	-	2.5				16.3	5.6	2.5	2.5	1.3	0.5					
Ore Grade	g/t	1.01				1.03	0.79	0.89	0.93	1.09	1.36					
Contained Gold	koz	288.3				3.2	32.5	57.9	61.1	61.5	72.2					
Stage 3																
Waste Mined	Mt	44.5					1.3	7.0	6.2	8.4	8.2	6.0	4.4	2.5	0.7	
Ore Mined	Mt	21.1					0.0	0.0	1.0	2.1	3.3	4.5	3.5	3.5	2.4	0.7
Strip Ratio	-	2.1					52.8	230.2	6.0	4.0	2.5	1.3	1.3	0.7	0.3	
Ore Grade	g/t	1.60					0.70	0.97	0.96	0.88	0.94	1.08	1.40	1.79	3.40	5.39
Contained Gold	koz	1,086.8					0.5	0.9	31.4	58.8	100.3	156.0	156.8	202.6	261.9	117.6

Location	Unit	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
NEB Total																
Waste Mined	Mt	75.3		1.0	4.2	3.9	9.6	12.1	11.3	10.6	9.0	6.0	4.4	2.5	0.7	0.0
Ore Mined	Mt	39.6		0.2	3.4	4.1	3.4	2.1	3.1	3.8	5.0	4.5	3.5	3.5	2.4	0.7
Strip Ratio	-	1.9		5.2	1.2	1.0	2.8	5.9	3.7	2.8	1.8	1.3	1.3	0.7	0.3	0.0
Ore Grade	g/t	1.36		0.93	1.09	1.15	1.12	0.89	0.94	0.97	1.08	1.08	1.40	1.79	3.40	5.39
Contained Gold	koz	1,738.3		5.8	119.7	151.1	122.7	58.9	92.5	120.3	172.5	156.0	156.8	202.6	261.9	117.6
BC Pit																
Waste Mined	Mt	4.5											2.0	2.0	0.5	
Ore Mined	Mt	3.5											0.6	1.8	1.0	
Strip Ratio	-	1.3											3.1	1.1	0.5	
Ore Grade	g/t	1.78											1.44	1.67	2.18	
Contained Gold	koz	199.8											29.4	98.6	71.8	
Total Open Pit Mining																
Waste Mined	Mt	81.7	2.0	1.0	4.2	3.9	9.6	12.1	11.3	10.6	9.0	6.0	6.4	4.4	1.2	0.0
Ore Mined	Mt	43.7	0.6	0.2	3.4	4.1	3.4	2.1	3.1	3.8	5.0	4.5	4.1	5.4	3.4	0.7
Strip Ratio	-	1.9	3.5	5.2	1.2	1.0	2.8	5.9	3.7	2.8	1.8	1.3	1.5	0.8	0.3	0.0
Ore Grade	g/t	1.39	0.73	0.93	1.09	1.15	1.12	0.89	0.94	0.97	1.08	1.08	1.41	1.75	3.03	5.39
Contained Gold	koz	1,951.3	13.1	5.8	119.7	151.1	122.7	58.9	92.5	120.3	172.5	156.0	186.2	301.2	333.7	117.6

Table 16.21: Life of Mine Schedule – Mill Feed by Lithology and Production

Material	Unit	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Laterite	Mt	1.60	1.16	0.27		0.04	0.10				0.02	0.00	0.00	0.01	
	g/t	0.98	1.08	0.61		1.16	0.69				1.61	0.58	0.58	0.58	
	Recovery (%)	92.1%	92.1%	91.8%		92.1%	91.9%				92.4%	91.8%	91.8%	91.8%	
	Production (koz)	46.4	37.1	4.8		1.5	2.0				0.7	0.0	0.0	0.2	
Saprolite	Mt	8.43	1.75	2.25	1.48	1.61	0.87	0.02	0.01	0.01	0.10	0.19	0.15		
	g/t	1.03	1.20	1.24	0.99	0.70	0.80	0.68	0.84	0.84	1.49	1.02	0.93		
	Recovery (%)	92.1%	92.2%	92.2%	92.1%	91.9%	91.9%	91.9%	92.0%	92.0%	92.3%	92.1%	92.0%		
	Production (koz)	257.2	62.6	82.5	43.4	33.4	20.5	0.4	0.2	0.2	4.3	5.6	4.3		
Saprock	Mt	0.76	0.01	0.09	0.05	0.21	0.12	0.00			0.10	0.08		0.10	
	g/t	1.34	0.89	1.20	1.21	1.14	1.02	0.82			2.49	2.14		0.60	
	Recovery (%)	92.4%	92.0%	92.2%	92.2%	92.1%	92.1%	92.0%			92.8%	92.6%		91.8%	
	Production (koz)	30.0	0.2	3.1	1.7	6.9	3.6	0.1			7.7	4.9		1.8	
OP Fresh	Mt	32.9	0.05	0.49	1.57	1.27	2.10	3.52	4.49	4.49	4.28	4.24	4.35	2.06	
	g/t	1.50	0.79	1.31	1.38	1.12	1.01	1.00	1.12	1.08	1.28	2.02	2.48	2.36	
	Recovery (%)	92.4%	91.9%	92.2%	92.2%	92.1%	92.1%	92.0%	92.1%	92.1%	92.2%	92.6%	92.8%	92.7%	
	Production (koz)	1,468.5	1.1	19.1	64.2	42.1	62.5	104.0	148.8	143.8	162.1	254.5	321.6	144.7	
UG Fresh	Mt	7.70	1.26	1.40	1.40	1.36	1.32	0.96							
	g/t	3.93	3.97	3.78	3.77	3.77	4.28	4.06							
	Recovery (%)	93.6%	93.6%	93.5%	93.5%	93.5%	93.8%	93.6%							
	Production (koz)	910.1	150.	159.5	158.9	154.5	169.9	117.2							

Material	Unit	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13
Total	**Mt**	**51.39**	4.22	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	2.17	
	g/t	**1.77**	1.99	2.00	1.99	1.77	1.92	1.65	1.12	1.08	1.31	1.98	2.43	2.26	
	Recovery (%)	**92.8%**	93.0%	93.0%	92.9%	93.0%	93.2%	92.9%	92.1%	92.1%	92.2%	92.6%	92.8%	92.7%	
	Production (koz)	**2,712.2**	251.0	269.0	268.1	238.5	258.4	221.7	149.0	143.9	174.9	265.0	325.8	146.8	

16.7 Open Pit Mining Operations

16.7.1 Open Pit Mining Approach

The open pit mine operating strategy has been based on contract mining with PDI responsible for the overall mining operation and assuming any statutory requirements. This is the preferred operating strategy on the basis that the contractor provides:

- Mining equipment and infrastructure, significantly reducing initial capital.
- Operational personnel (locally sourced wherever possible) along with established training and operating procedures and systems, which will reduce startup time frame and risk.

These benefits will generate the best project value with the greatest likelihood of success for a greenfields project start-up.

PDI will be responsible for the following:

- Geology and the associated grade control process.
- Geotechnical overview and any associated monitoring and review.
- Mine planning, design and scheduling.

The mining contractor will be responsible for all ongoing mining activities including, but not necessarily limited to:

- Mobilisation of equipment to site and establishing site infrastructure
- Clearing and topsoil stripping,
- Ex-pit haul road construction and maintenance.
- Placement of bulk fill for ROM pad.
- Drilling and blasting.
- Dozer ripping of ore, as required by material properties.
- Load and haul of ore and waste.
- Rehandle of underground ore and waste.
- Crusher Feed and stockpile rehandle.
- Waste dump rehabilitation.
- Emergency response and preparedness.
- Pit dewatering.
- Dust Suppression.

16.7.2 Clearing & Topsoil Removal and Storage

Clearing of the moderate tree cover and light underbrush will be necessary before removal of topsoil to stockpiles for later use in rehabilitation of the site. Topsoil ranges in depth from insignificant on the top of the hills to up to 1 m in the valley floors. Therefore, an average depth of 0.3 m has been assumed for the mining areas given they are predominantly located on the higher ground. Topsoil will be removed and placed in stockpiles located adjacent to the areas of disturbance to limit the distances required for haulage to stockpile, and subsequent rehandle off stockpile for rehabilitation purposes. Topsoil stockpile height will be capped at 2.5 m. To preserve the soil viability and reduce rehandle costs, it is planned that rehabilitation is undertaken progressively, primarily of waste rock dumps as final faces are reprofiled. After the initial phase of topsoil stripping, any subsequent topsoil stripped from new areas will be placed directly onto reprofiled waste dump slopes whenever possible.

16.7.3 Grade Control

Grade control drilling will be campaigned using RC methods to ensure sample quality. A geological assessment for the drilling density has recommended a pattern of 10 m by 7 m at 20 m vertical intervals. Samples will be collected for 1 m composites using a 3-stage splitter to reduce sample size. Each sample will be bagged and labelled at the drill hole before being collected and sent to an on-site laboratory for assaying.

Ore presents from with 5 m of surface in all the mining areas. Consequently, ore areas will be defined based on the resource model and quarantined for the grade control drilling campaign after which the ore will be marked out and mined selectively from the waste. Ore will be hauled to either the ROM pad located adjacent to the process plant, or to a long-term stockpile for reclaim later in the mine life. Stockpiles will be separated based on grade and weathering type (i.e. saprolite, fresh rock and other weathered materials) to facilitate suitable blending to the mill. The excavator operator will have access to the digital grade control plan and GPS location during mining.

The entire grade control program for the GBE pit will be undertaken in advance of mining to ensure rapid bench turnovers can be achieved with no delays from grade control.

16.7.4 Drilling and Blasting

All material will require a degree of blasting. For this study it has been assumed blasting will be carried out on a combination of 5 m and 10 m benches. The drill and blast design parameters and assumptions are shown in Table 16.22.

Wall control will be managed through a combination of buffer blasts and pre-shear/pre-split incorporated into the drill and blast processes to minimise damage to pit walls. It is assumed that buffer blasts will be free faced where possible, with lower powder factor and denser drill spacing than the production blasts.

Explosives and associated blasting products will be sourced in-country from a government approved supplier. The mining contractor will undertake production and pre-split drilling, loading and firing of the blasts. They will also have suitably qualified personnel to fulfil any statutory or regulatory requirements for blast supervision. Regardless of in-country requirements, blast control and safety will be undertaken to world industry standards as a minimum.

Table 16.22: Drill and Blast Design Parameters

Description	Bench Height (m)	Hole Dia. (mm)	Burden (m)	Spacing (m)	Powder Factor (kg/m^3)	% Free Dig	% Blasted
Saprolite Clay	5	152	5.3	8.0	0.26	75%	50%
Mottled Clay						50%	50%
Laterite			4.6	6.0	0.41	0%	50%
Saprock						0%	50%
Fresh Rock			3.8	4.6	0.64	0%	25%
Saprolite Clay	10	178	6.2	9.3	0.36	75%	50%
Mottled Clays						50%	50%
Laterite			5.3	6.9	0.57	0%	50%
Saprock						0%	50%
Fresh Rock			4.5	5.4	0.86	0%	75%

16.7.5 Load and Haul

The mine plan has been developed assuming a combination of 150-tonne class excavators for bulk mining and with 100-tonne class excavators for selectively mining ore and associated waste to minimise dilution and ore loss.

The ore generally dips to the west at 40° to 50°. Wherever possible the ore will be faced up from the east hanging wall side to allow the waste to be stripped off with minimal ore loss. The bucket width of approximately 2.5 m for the 100-tonne class excavator will permit adequate selectivity when digging to the ore contact.

The both the 100-tonne and 150-tonne class excavators are matched to 90-tonne class rear dump trucks nominated by the selected mining contractor.

16.7.6 Open Pit Mine Production Fleet

The following equipment will be supplied by the mining contractors for the activities detailed below:

- Dozers, floor control and waste dump tip-head management.
- Graders, maintenance of roads in-pit, ex-pit and on stockpiles and dumps.
- Front end loaders, ongoing ROM re-handle, stockpile reclaim and general site maintenance.
- Water truck, dust suppression.
- Trailer mounted diesel pumps, dewatering sumps within the open pits.
- Lighting towers, night operations.

The primary production fleet proposed by the selected mining contractor is provided in Table 16.23 with the numbers shown for steady state mining operations.

Table 16.23: Proposed Open Pit Mining Fleet

Category	Type	Model	Specification		Number
			Unit	Value	
Loading	Excavator	Caterpillar 6015	operating weight (t)	140	3
	Excavator	Caterpillar 395	operating weight (t)	94	1
	Excavator	Caterpillar 374	operating weight (t)	74	1
	Excavator	Caterpillar 335 (w/ hammer)	operating weight (t)	35	1
	FEL	Caterpillar 988	bucket capacity (m^3)	6.5	2
Drilling	Drill	EPIROC D65	hole dia.(mm)	110-229 mm	3
Hauling	Dump Truck	Caterpillar 777	capacity (m^3)	60	13
Support	Track Dozer	Caterpillar D9	power (kW)	357	6
	Wheel Dozer	Caterpillar 834K	power (kW)	419	1
	Motor Grader	Caterpillar 16	blade length (m)	4.9	2
	Water Truck	Caterpillar 777	tank size (kl)	75	2
	Water Truck	MAN 6x6	tank size (kl)	20	2
	Roller/Compactor	Caterpillar CS78	weight (t)	18	1
	Rockbreaker	Furukawa FXJ375/ Caterpillar 335	weight (t)	2.6	1
U/G Rehandle	FEL	Caterpillar 992	bucket capacity (m^3)	105	1
	Truck	Caterpillar 777	capacity (m^3)	90	3
Total					**43**

16.7.7 Dewatering and Surface Water Management

Dewatering of the open pits will be achieved through a combination of:

- Dewatering bores around the periphery of the workings.
- In-pit dewatering from sumps via diesel pumps.

The site water management infrastructure is discussed in Section 18.9, including the operation of the dewatering bores.

Surface water management is discussed in Section 18.8, however in general surface water flows away from the open pits, except for the BC pit which is located in a seasonal water course. As detailed in Section 18.8, this pit will require an engineered stream diversion channel and embankment on the upstream side of the pit and a flood protection bund on the downstream side closest to the Niger River.

For the BC waste dump and the NEB/GBE pits and dumps, surface water management infrastructure has been developed which:

- Limits ingress into the pits from surface run-off (e.g. diversion berms or drains etc.).

- Manages discharge of any silted or contaminated water from the site.

Water inflows into the pits from either ground water and/or surface run-off will be the responsibility of the mining contractor. In-pit water management will primarily consist of runoff control and in-pit sumps. This water will be pumped via portable diesel-powered pumps to dams constructed by the mining contractor at the pit edge. From there the water will either be utilised for dust suppressions or pumped to the site water storage dams.

16.7.8 ROM Management

The ROM operating strategy is based on stockpiling ore at a central location adjacent to the processing plant. Blending for processing efficiency, including feeding of "stick clay ore" to the dedicated mineral sizer crushing circuit, requires management of both the saprolite clay and fresh rock components of the plant feed within the following constraints:

- Saprolite clay, maximum of 50% of mill feed.
- Fresh rock, minimum of 25% of mill feed.

Consequently, the ROM stockpile will comprise a minimum three fingers for:

- Fresh ore.
- Saprolite ore.
- Other materials (laterite, saprock and mottled).

Ore will be fed to the primary crushers from these fingers utilising a front-end loader.

The crusher bins will allow direct tipping for a Cat 777 sized truck as selected by the mining contractor. For the DFS, it has been assumed that 70% of ore sent to the ROM Pad will be able to be directly tipped into the crusher: This is on the basis that:

- Material will require on-going blending as detailed above.
- When the ore is being loaded by the larger excavator, ex-pit ore production will exceed the crusher throughput rate.
- Ore reclaimed from stockpiles can generally be schedule to match crusher feed requirements.

16.7.9 Ore Stockpiling

Over the life of the project some low-grade ore will be directed to a long-term stockpile located north of the GBE pit (refer to Section 16.4.4). This will primarily be lower grade plant feed that will allow higher grade material to be preferentially fed to the process plant early in the mine life. Saprolite material will also be stockpiled to ensure an acceptable feed blend is maintained, and this material may be of higher grade.

Ore from the underground operation will be hauled to surface and placed in a stockpiling area adjacent to the GBE pit ramp. From there it will be reclaimed by a front-end loader loading the open pit mining trucks for transport to the ROM pad. It is assumed this material will be rehandled on an on-going basis and 100% direct tipped due to its consistently high grade. Therefore, these stockpiles do not have to be large to accommodate discrepancies between mining and crushing rates.

16.7.10 Waste Rock Dump Management

The WRDs will be constructed in 10 m lifts as per the design criteria described in Section 16.4.1.4. The tip head will be maintained on an-ongoing basis by bulldozer, whereby waste rock will be dumped short of the active face and then dozer pushed over the face to ensure safe operating practices. The upper surface of the dump will be maintained with a gradient and appropriate drainage to prevent ponding of water.

Progressive rehabilitation of the waste dumps will be carried out over the life of mine as areas of the waste dumps are completed. This will involve:

- Reprofiling of the outer dump face to the final landform.
- Reclaiming topsoil material from stockpiles and spreading on dump face and top surfaces in preparation for revegetation.

16.7.11 Mine Infrastructure

The mining infrastructure areas is located immediately to the southwest of the process plant. This designated area will be provided to the mining contractor by PDI, inclusive of power and potable water off-take points. The contractor will be responsible for the construction of all open pit mining facilities, including but not necessarily limited to:

- Mining offices for the Contractor's personnel and some members of the Company's mining team.
- Ablutions facilities for all its mining personnel.
- Messing facilities for all its mining personnel.
- Heavy vehicle and light vehicle workshop including all required fit out.
- Lubrication and fluids storage, distribution and waste collection systems in the heavy and light vehicle workshops.
- Tyre change equipment and facilities.
- Mine parts and consumables warehouse.

PDI will provide fuel storage and distribution facilities along with a heavy vehicle washdown system for the use of both the open pit and underground contractors. The layout and facilities of this area is discussed in Sections 18.10 and 18.11.

16.7.12 Explosives Storage and Management

Supply of explosives and associated blasting products will be sourced in-country from a government approved supplier. Their product will be stored in a magazine constructed by PDI between the NEB/GBE pit and the TSF. It has been estimated that no more than 300 tonnes of bulk explosives storage is required on site at any given time.

If the appointed mining contractor has the appropriately qualified and approved personnel to manage the magazine facility then they will undertake this responsibility, otherwise the supply contractor will be contracted to fulfil this requirement.

Security of the magazine will fall under PDI's overall site security mandate.

16.7.13 Open Pit Management and Supervision

The open pit mining operation will be undertaken by a suitably qualified and experienced mining contractor who will undertake ongoing daily control and supervision of the mining operation. As such they will employ their supervisory team to ensure safe and effective operations, and suitable quality control of mill feed blend.

PDI will have an owner's team that manages the mining contractors and takes ultimate responsibility for the open pit mining operation.

The structure and number of personnel within this team are detailed in Section 24.2. The owner's team will be responsible for all the technical aspects on the open pit operation, including but not limited to:

- Geology and grade control.
- Mine planning and scheduling (short, medium and long term).
- Blast design.
- Geotechnical monitoring and evaluation.
- Hydrogeological monitoring and evaluation.

16.8 Underground Mining Operations

16.8.1 Underground Mining Philosophy

The underground mining philosophy for the Project is designed to maximise resource recovery, optimise operational efficiency, and align with processing constraints, while effectively managing geotechnical risks. Given the high-grade nature of the orebody, the selected method combines transverse and longitudinal long hole open stoping with engineered paste fill to enable large-scale stope extraction without leaving behind stabilising pillars. A bottom-up mining sequence has been adopted to support this approach, as it allows for larger stope designs and reduces the number of required stope cycles, thereby improving productivity and lowering unit costs. The use of paste fill plays a critical role in achieving this strategy, serving as structural support to enable the maximum stope recovery while minimising ground stability risks. Careful attention has been given to sequencing to limit dilution from exposure to paste-filled voids.

The underground operation will adopt a conventional mechanised mining approach, in line with industry best practices. Lateral development will be carried out using twin-boom jumbo rigs, which will also install in cycle ground support. In areas requiring increased ground control, such as sill pillar drives, the application of fibrecrete will provide the required additional reinforcement.

Vertical development will be executed using a combination of raise boring equipment for larger diameter infrastructure (e.g., return air raises) and long hole drilling techniques for smaller openings such as slot raises, secondary ventilation, escapeways and service holes.

Once footwall access drives are completed, grade control drilling using RC rigs will be undertaken to define ore boundaries and support detailed stope design. Stopes will be planned to maximise recovery while controlling dilution and will be drilled using long hole production rigs. Blasting activities will utilise emulsion explosives and electronic detonators to ensure precise rock fragmentation and controlled breakage. Broken ore will be bogged by LHD units and stored in level stockpiles before being transferred to underground trucks and hauled to the surface stockpile. From there, surface haulage trucks will deliver the material to the ROM pad for processing.

It should be noted that the optimisation and design of the underground was completed including consideration of potential future mining of the Inferred Mineral Resources, which is not included in the Mineral Reserve, for strategic planning purposes at the request of the Company. The Qualified Persons confirm that this approach has not impacted on the costs of mining or the mining recovery of Mineral Reserves as the Inferred Mineral Resources are spatially separate and located below the Mineral Reserves.

16.8.2 Grade Control

Effective grade control is essential for ensuring the economic viability and metallurgical performance of the underground operation. Given the high-grade nature of the orebody and the importance of minimising dilution while maximising recovery, a detailed grade control strategy has been developed and integrated into the mine planning process.

Grade control drilling will be conducted from footwall drives prior to stope design finalisation and blasting. Due to the orebody's geometry, dipping at approximately 45°, some stopes will require grade control to be conducted from levels above or below, depending on drill coverage and access constraints.

A total of approximately 48 km of grade control drilling is planned throughout the life of the underground operation. These metres have been strategically scheduled to align with development progress and stope sequencing, ensuring that every production stope is fully informed by current, high-resolution grade data before any material is extracted.

16.8.3 Drill and Blast

Drill and blast activities will be conducted in line with standard industry practices, utilising long hole drill rigs and emulsion explosives to ensure effective and controlled stope extraction. Upon completion of development activities and the installation of hangingwall cable bolts, long hole drilling of the full stope will be undertaken.

While the use of boxhole drilling methods was evaluated, the dip of the orebody lies outside the optimal operational range for such equipment. Their use could potentially result in increased dilution and reduced ore recovery. As a result, a conventional uphole drilling method was selected as the most appropriate and efficient approach for the orebody geometry.

Following completion of stope drilling, brow cables will be installed to reinforce the stope brow and ensure ground stability. Charging operations will generally be conducted in three stages:

- Slot raise.
- Free-face rings.
- Remainder of the stope rings.

This staged approach supports operational safety and productivity by facilitating controlled breakage and effective bogging. Standard brow management procedures will be followed, including the installation of safety bunds between firings, to further reduce operational risk and ensure compliance with best practice safety protocols.

Production drill holes will be 89 mm in diameter and drilled as upholes, while slot raises will use developed using 152 mm diameter reamed holes. A typical ring layout design uses a burden of 2.5 m and a spacing of 3.0 m, resulting in a drill density as detailed in Table 16.24. The firing sequence described in the steps above can be seen in Figure 16.48.

Table 16.24: Drill Density per Mining Method

Mining Method	Hangingwall Length (m)	Drill Density (t/dm)
Longitudinal	20	12
	15	9
	10	6
Transverse	20	8
	15	6
	10	4

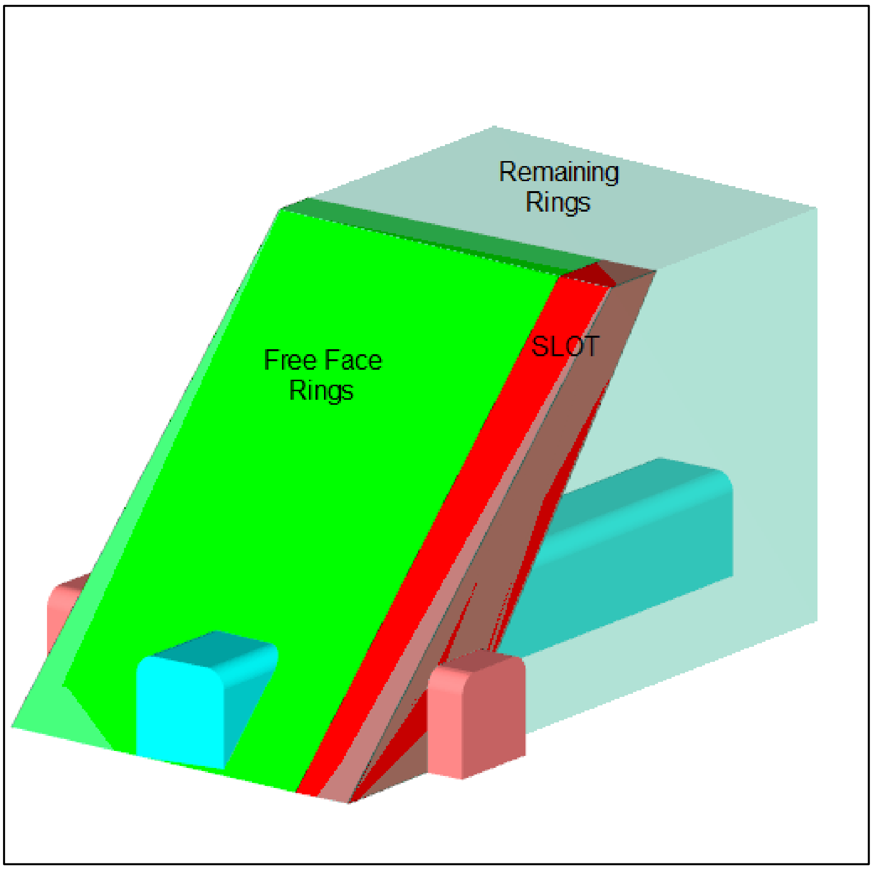

Figure 16.48: Firing Sequence (Orelogy 2025)

Figure 16.49 shows a cross-sectional view of the typical drill pattern for a production ring in a transverse long hole open stope while Figure 16.50 shows a longitudinal long hole open stope.

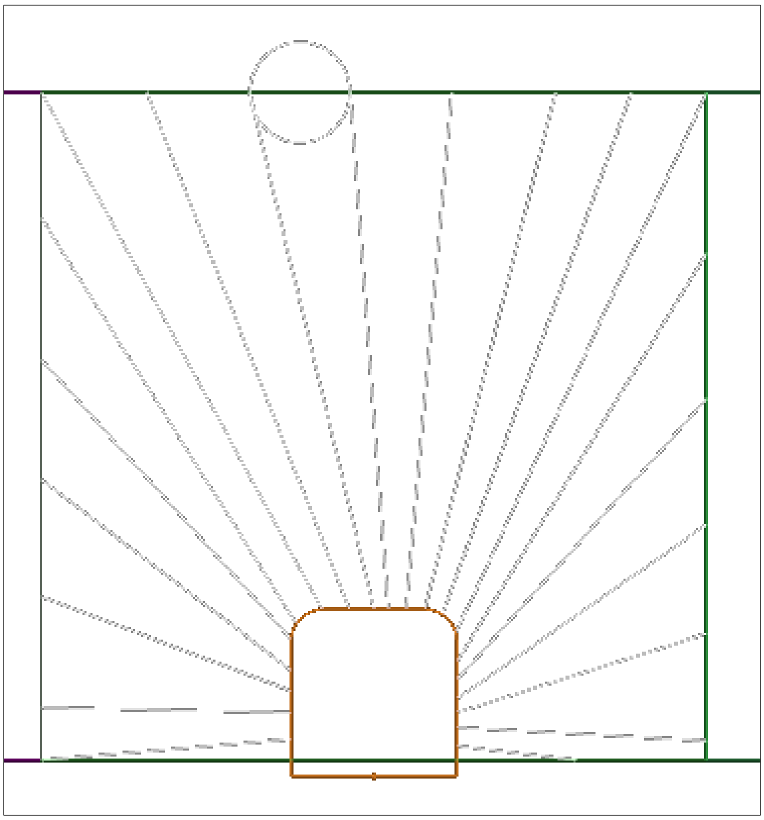

Figure 16.49: Typical Production Ring Configuration for Transverse Stopes (Orelogy 2025)

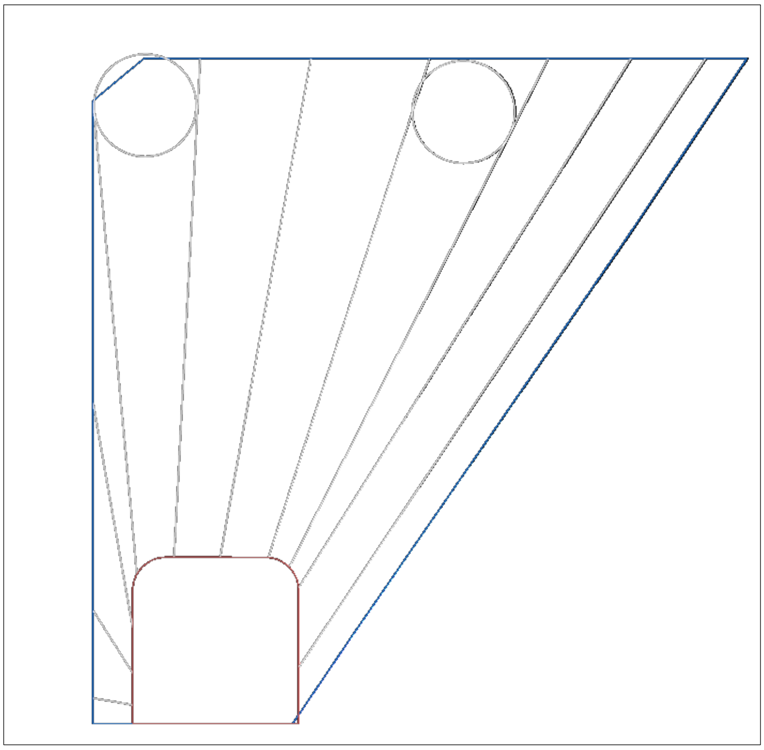

Figure 16.50: Typical Production Ring Configuration for Longitudinal Stopes (Orelogy 2025)

16.8.4 Material Transport System

The materials handling transport system will be conventional underground dump trucks and LHD units, or "boggers". The dump truck selected by the underground contractor was the Toro TH663i with a nominal payload of 63 t, paired with a Toro 21t LHD for development (LH621i) and a Toro 17t LHD (LH517i) for production stoping.

Ore and waste will be trucked to the underground ROM via the haulage decline. Once dumped on the underground ROM, material will be rehandled by a surface contractor using larger trucks and FEL delivering ore to the mill ROM pad 2.1 km away for processing and waste to the designated waste dump.

Figure 16.51 shows the surface layout for the underground and mill ROM pads and waste dumps.

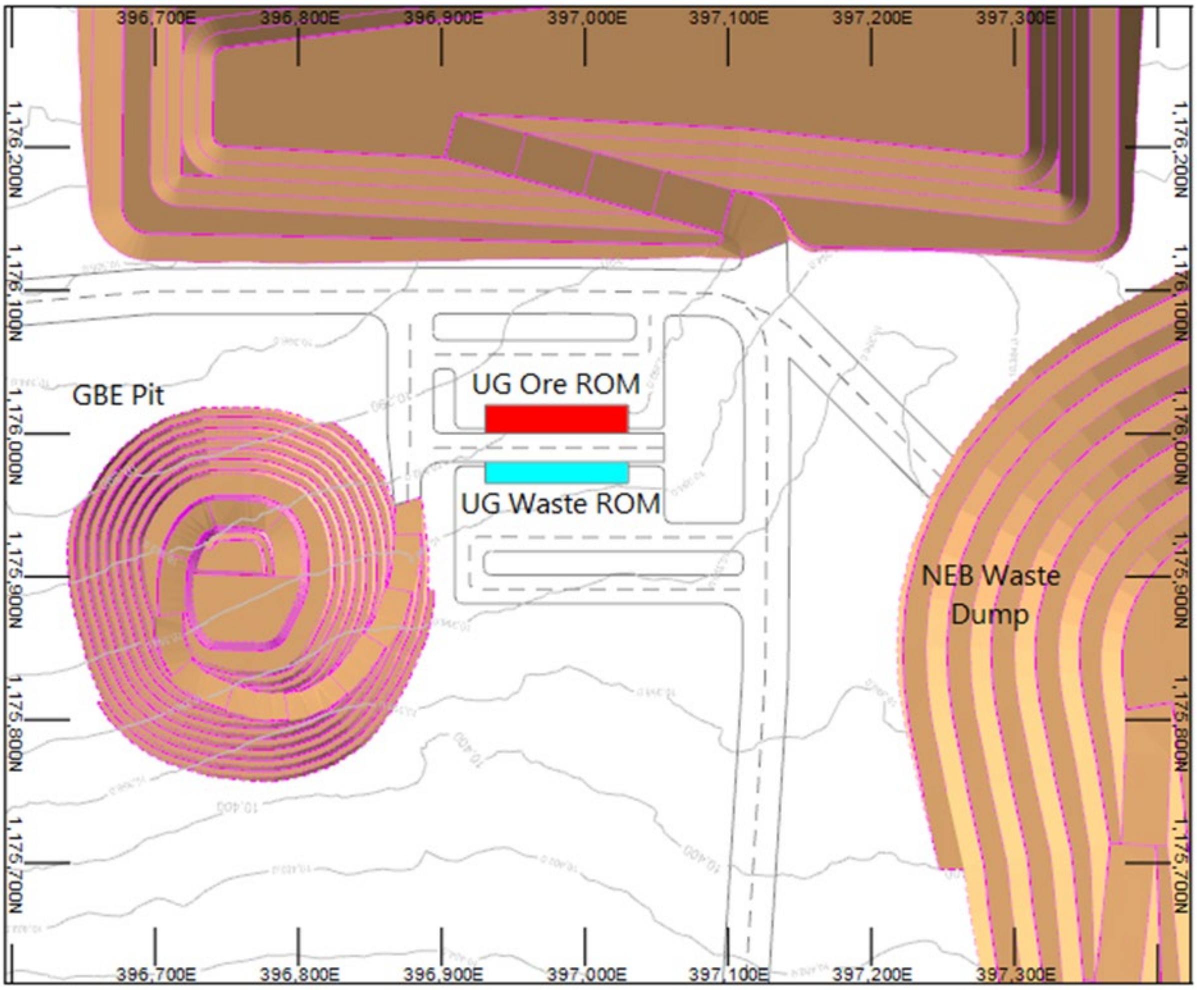

Figure 16.51: Surface Layout

16.8.4.1 Options

Underground mining operations around the world utilise a variety of haulage systems, including shaft hoisting, ore passes, conveyor belts, and combinations thereof, with each selected based on production rates, mining method, site-specific constraints and long-term operational objectives.

For the Project, a truck and bogger haulage system has been selected due to its operational flexibility, lower capital cost, and reduced implementation complexity. Unlike shaft hoisting and conveyor systems, which require high upfront investment and are typically justified only in operations with extensive mine life and high production rates, the truck-and-bogger method enables rapid deployment and adaptation to variable production demands.

The use of ore passes was also evaluated as a potential method to reduce level drive dimensions. While ore passes offer benefits in reducing lateral development for materials handling, the moderate dip of the orebody (~45°) would necessitate increased vertical and lateral development to implement this system effectively. This added development would offset the anticipated efficiencies, particularly in the early years of operation.

Therefore, the selected truck-and-bogger system represents the most practical and cost-effective solution for Project's current mine life and production profile.

16.8.4.2 Bogging

Stope and development ore will be extracted using LHD equipment, which will transfer material to designated stockpile locations within nearby drives. From these stockpiles, the ore will be rehandled into trucks for haulage to surface.

Stope ore will be bogged using a combination of manual and remote-control operation to ensure safe working conditions, particularly in areas with elevated ground control risks. A 40/60 manual-to-remote operating split has been applied to cost and productivity assumptions. Given that stockpiles are planned within 150 m of the stope source, high productivity rates are anticipated for these operations.

Figure 16.52 shows the loading points and stockpile locations on the 9,980 level, which is a typical example of the layout of a level in plan view.

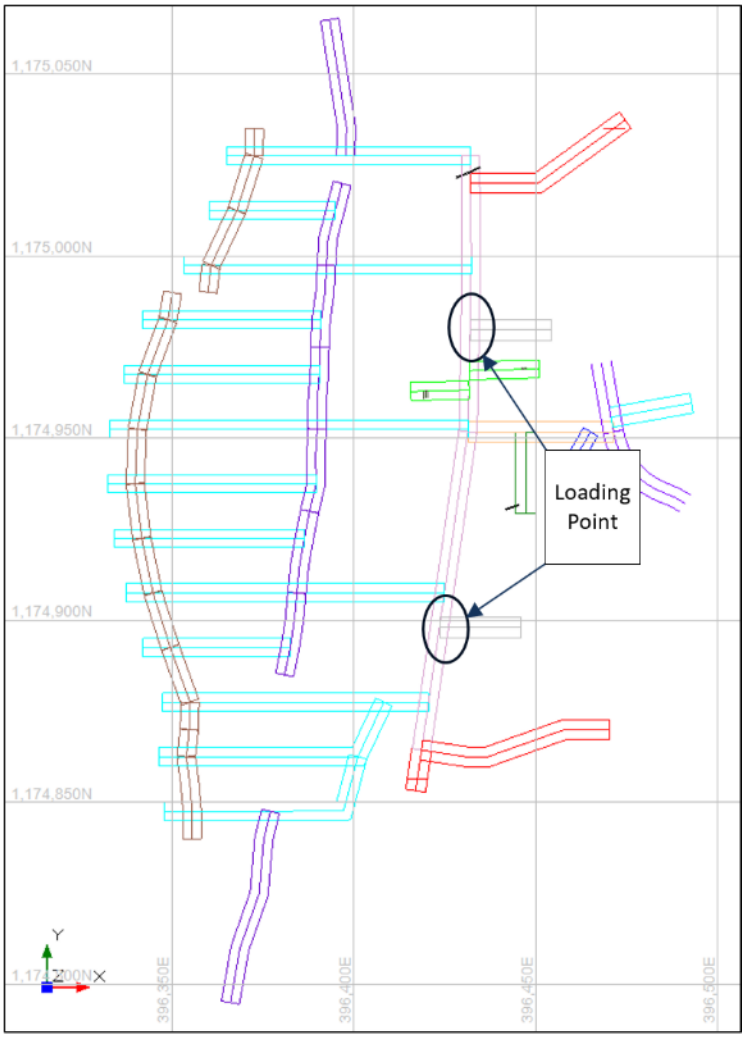

Figure 16.52: Level Layout Showing Loading Points – 9980 mRL

16.8.4.3 Tele-Remote System

A tele-remote bogging system is planned to be installed on every level of the mine to enhance safety and operational efficiency. Remote operating huts will be strategically positioned throughout the mine, including at decline stockpiles and escapeway drives on each level. These huts will be connected to the active production areas via communication cables, enabling safe remote bogging of stopes without exposing personnel to high-risk zones.

The system will be supported using laser barriers and clear signage to ensure complete isolation of remote bogging areas from the rest of the mine during operation. This setup is designed to maintain a safe working environment while maximising productivity.

16.8.5 Underground Mine Production Fleet

The underground mine production fleet will be supplied, maintained and operated by the underground mining contractor. Based on the steady state mining production requirements the estimated underground mining primary production fleet is outlined in

Table 16.25: Underground Mine Production Fleet

Category	Type	Model	Specification		Number
			Units	Value	
Mine Development	Loader	Sandvik LH621i	operating weight (t)	21	1
	Twin Boom Jumbo	Axera DD422i	power (kW)	120	2
	Charge Wagon	Charmec 605	tank size (kl)	0.72	1
Ground Support	Spraymec	Normet	concrete output (m^3/h)	27	1
	Agi Truck	Normet	tank size (m^3)	7	1
	Cabolter	DS422i	power (kW)	110	1
Production & Haulage	Sandvik Loader	LH517 (17t)	operating weight (t)	17	4
	Dump Truck	Sandvik TH663i	operating weight (t)	63	7
	Charge Wagon	Charmec 600	emulsion capacity (t)	4	2
	Production Drill	Sandvik DL432i	power (kW)	120	2
	ITH Drill	DU411	power (kW)	110	1
Mine Support Equipment	Grader	Cat 12H	blade length (m)	2.5	1
	Integrated Tool Carrier	Volvo 120	horsepower (hp)	276	6
	RC Rig	varied	power (kW)	110	1
	Diamond Drill Rig	varied	power (kW)	110	1
	Water Truck	varied	tank size (kl)	12.5	1
	Stores Truck	varied			1
	Service Truck	varied			1
Total					**35**

16.8.6 Ventilation

16.8.6.1 General

This ventilation study details the airflow requirements and strategy based on the proposed mine layout, peak diesel-powered equipment fleet, and other ancillary airflow needs. It considers the project's peak primary airflow demands and the necessary fan duties to meet these requirements.

The design criteria for the ventilation used in the study is summarised in Table 16.26.

Table 16.26: Summary of Ventilation Design Criteria

Parameters	Unit	Value	Source/Rationale
Diesel Dilution Criteria	m^3/s/kW	0.05	Western Australia, Work Health and Safety (Mines) Regulations 2022, Regulation 656C
Velocities			
Airway Velocity	m/s	0.3 if WB $<$ 25.0°C	Minimum velocities temperatures
		0.5 if WB $>$ 25.0°C	
Intake Drives Velocity		6.0 – 7.0	To minimise dust mobilisation in travel ways. Industry norm usually specifies maximum of 6m/s.
Return Raise Velocity		13.0 – 22.0	Critical velocity range for water build-up is 7-12m/s. Up cast velocity should be designed outside this range. Economic evaluation to be considered.
Friction Factors			
Raise Bored Holes	Ns^2/m^4	0.005	The MVSSA Data Books & McPherson (Subsurface Vent and Environment Engineering)
Long-hole Raises		0.015	
Declines and Level Drives		0.012	
Flexible Duct		0.004	

The diesel exhaust emission (DEE) dilution airflow provisions specified under the Western Australia Work Health and Safety (Mines) Regulations 2022 (Government of Western Australia, 2022) will not sufficiently dilute particulate emissions from older engines, such as Tier III and below. It is recommended that all diesel-powered mining equipment be specified with Tier IV engines. Machines with lower ratings will require additional exhaust filtration, such as particulate filters, to adequately reduce exhaust particulate concentrations or an increase in airflow provisions.

16.8.6.2 Legislative Requirements

Any ventilation design must comply with the applicable legislative requirements. The work undertaken in this DFS was specifically designed using the Australian Work Health and Safety (Mines) Regulations 2022 (Government of Western Australia, 2022) for the ventilation requirements. Where these regulations offer no guidance or have been surpassed by more advanced practices, preference was given to relevant guidelines and best practices from Western Australia and other contemporary mining jurisdictions.

16.8.6.3 Air Velocity Constraints

To protect personnel from health and safety hazards in underground workplaces, travelway air velocities must be maintained within specific ranges.

High air velocities can lead to:

- Dust entrapment, where visible dust may become airborne due to air movement, which can be exacerbated by the movement of machinery and personnel.

- Equipment damage such as vehicle doors, may be damaged if they are pulled beyond their normal range by passing airflow.
- Mechanical injuries where personnel may suffer injuries from equipment or ventilation doors being forcibly opened or closed by strong air currents.

Conversely, low air velocities can create different hazards, including:

- Contaminant accumulation where ineffective removal of contaminants from the workplace, leading to prolonged exposure.
- Inadequate dilution where insufficient dilution of DEE and strata gas, where applicable.
- Insufficient cooling where reduced cooling capacity leads to increasing the risk of heat stress among personnel.

16.8.6.4 Project Ventilation Demand

Determining airflow requirements must consider all relevant factors, including but not limited to:

- Ensuring adequate airflow to working spaces.
- Effectively diluting exhaust emissions from diesel engines.
- Controlling dust to minimise airborne particulates.
- Diluting flammable or toxic atmospheric contaminants, such as blasting fumes, toxic and asphyxiant gases, and radioactive dust and gas.
- Lowering temperatures to prevent heat stress and maintain a comfortable working environment.

16.8.6.5 Diesel Engine Exhaust Emission Dilution

In most cases, the requirements for diluting DEE are the primary factor determining the ventilation flow demand in mechanised underground metalliferous mines. DEE contains toxic and asphyxiant gases, as well as particulate matter, which must be diluted and removed from the mine to prevent personnel exposure to harmful contaminant concentrations. According to the Western Australian Work Health and Safety (Mines) Regulations 2022 (Government of Western Australia, 2022), a key requirement is to provide 0.05 m^3 per second per kilowatt of the maximum rated engine output specified by the manufacturer for the engine's fuelling and timing configuration (Part 10.2, Division 4, r. 656C - Additional ventilation requirements for diesel units).

Table 16.27 outlines the expected diesel equipment fleet and the required airflow for DEE dilution. It is critical that the number of diesel units operating underground are managed to ensure the total airflow demand required for the diesel equipment does not exceed the primary airflow capacity of 460 m^3/s. Maintaining this limit is essential to provide sufficient ventilation for the effective dilution of DEE, thereby safeguarding the health and safety of underground workers.

Table 16.27: Machine DEE Dilution Airflow Requirements

Description	Rated Power (kW)	Airflow Required (m3/s)
Dump Truck	565	28.3
Light Vehicle	151	7.6
Underground Loader	310/352	15.5/17.6
Integrated Tool Carrier	180	9.0
Charge Wagon	120	6.0
Grader	170	8.5
Twin Boom Jumbo	119	6.0
Cablebolter	119	6.0
Production Drill	110	5.5
Agi Truck	185	9.3
ITH Drill	119	6.0
Spraymec	100	5.0
RC Rig	110	5.5
Diamond Drill Rig	110	5.5

16.8.6.6 Infrastructure Airflow Requirements

No major underground infrastructure is planned to be located underground for the Project. All support facilities, including workshops, fuel bays, and magazines, will be situated on the surface. Therefore, no additional airflow has been allocated for fixed facilities underground.

16.8.6.7 Working Area Airflow Requirements

Adequate ventilation must be ensured in all working areas, considering:

- Equipment being used.
- Personnel present and the nature of their tasks.
- Additional ventilation needs (Auxiliary ventilation).
- Climatic conditions.

During the development phase, the minimum airflow required at the point of use is either that of a loader or 7.5 m^3/s. For declines, the required airflow is 9.6 m^3/s for a width of 5.5 m and a height of 5.8 m. In cases where dead-end headings occur during decline development or in ore drives, auxiliary ventilation systems must be employed to provide the necessary airflow to the working face(s).

In a working level with concurrent activities in two ore drives, it is typically considered as two distinct work areas. This requires a level be provided with sufficient air flow for the equipment active on the area as per Table 16.27 i.e. production loader and dump truck require 43.8 m^3/s. Additional airflow should be factored in to compensate for leakage losses and to maintain minimum air velocities in travelways.

When specifying auxiliary fans, additional airflow must be provided beyond the open-circuit capacity to prevent recirculation. This additional airflow must be sufficient to achieve a minimum air velocity of 0.3 m/s to 0.5 m/s in the tunnel, past the fan. The required volume depends on the cross-sectional area of the drive or decline. Note that flow-past allowance can be reused in other locations, so it does not contribute to the overall airflow demand.

Airflow demand will vary across mining areas depending on development and production schedules. Therefore, it is crucial to specify each fan accordingly.

16.8.6.8 Leakage

A leakage allowance of 20% has been included, which is considered realistic for a mining operation with reasonable controls and plans. Leakage depends on various factors, ranging from as low as 10% to over 50% of total mine airflows. Regular maintenance of ventilation control devices throughout the LOM is essential to ensure they operate as designed and to prevent additional leakage. With the leakage allowance, the maximum airflow required is 462.4 m^3/s.

16.8.6.9 Project Airflow Demand

Airflow demand will vary throughout the life of the project, dependent on many variables, the more important being:

- Scheduled activities.
- Equipment in use (number of units) and utilisation of those units.

Peak airflow requirements reach 435 m^3/s in year 6 of the project schedule as shown in Figure 16.53.

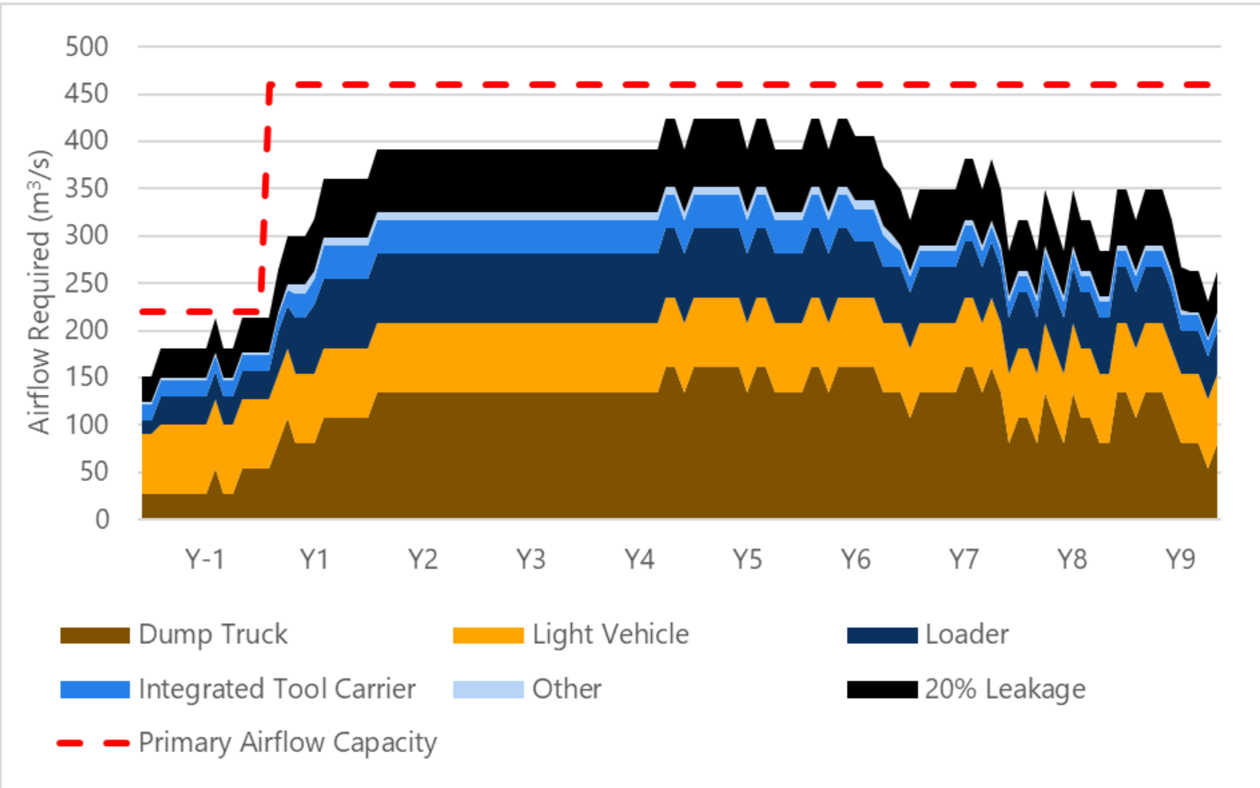

Figure 16.53: Project Airflow Demand

16.8.6.10 Primary Ventilation

The planned ventilation system will comprise four surface connections, including a main portal, a ventilation drive (or fresh air drive/return air drive (FAD/RAD)), a FAR dedicated to mine cooling, and a main RAR, which will serve as the primary exhaust pathway for the underground mine. Both the main portal and ventilation drive are accessible from within the GBE open pit. Sufficient separation between the intake portal and ventilation drive (FAD/RAD) is required to prevent recirculation of exhaust air when the ventilation drive is used as a temporary return airway (RAD) until the permanent primary fans are commissioned at the main RAR.

The proposed underground mine layout will support a negative pressure ventilation circuit. Three primary fans will be installed on surface in parallel and connected to the main return raise via a shaft-top bend. They will be mounted on concrete foundations and fitted with self-closing dampers to prevent air recirculation when one or more fans are offline. These fans will generate negative pressure, drawing fresh air through the main portal, ventilation drive, and FAR, and delivering it to the working levels. Return air from the working levels will pass through drop board regulators (DBRs) before being exhausted to the main RAR.

Each on-level primary exhaust connection must be fitted with a regulator, or similar device to allow control of airflow. These can take the form of:

- Conventional DBRs.
- Variable vane/louvre regulators.
- Slide regulators.

The primary ventilation system layout is shown in Figure 16.54 and Figure 16.55. These figures include inferred stopes (below 9,900RL) which are excluded from the Mineral Reserve and LOM production schedule.

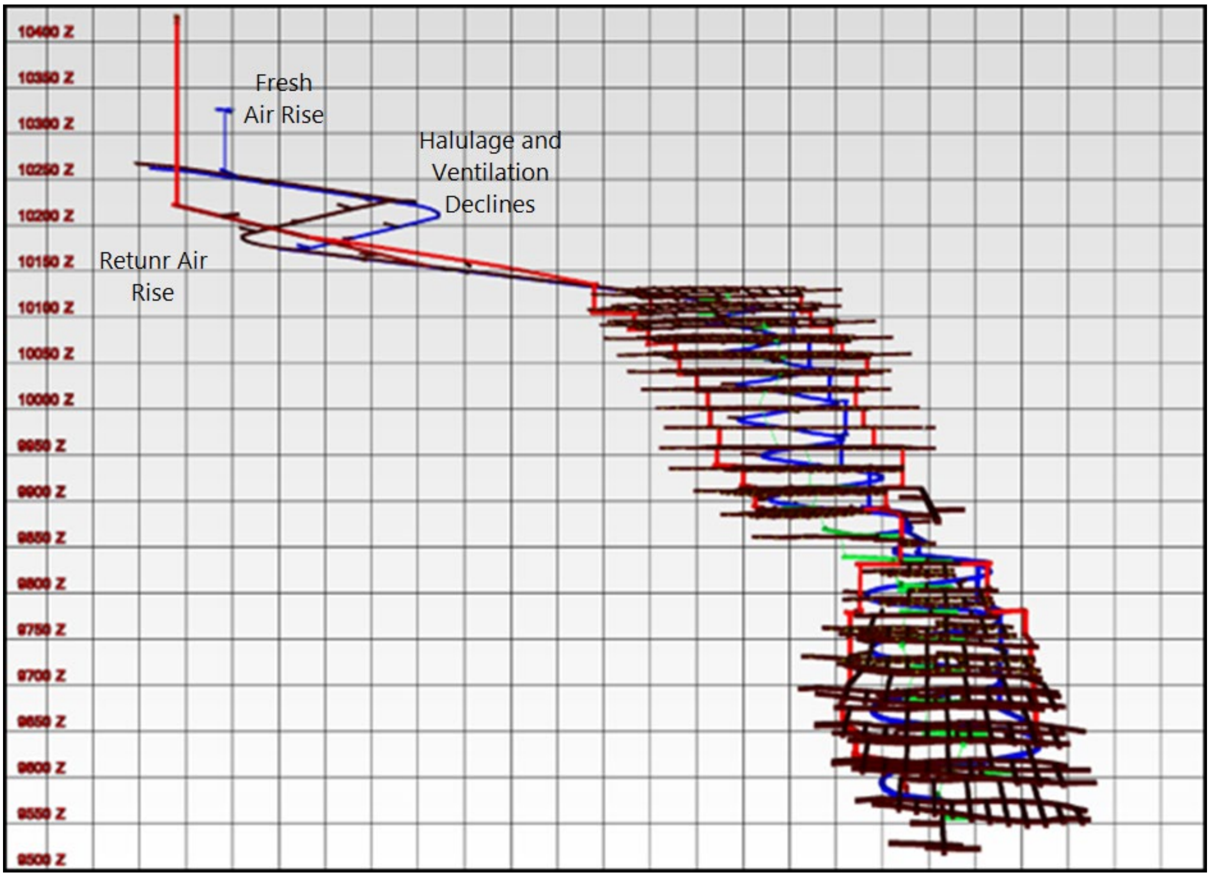

Figure 16.54: Primary Ventilation System – Section View Looking East

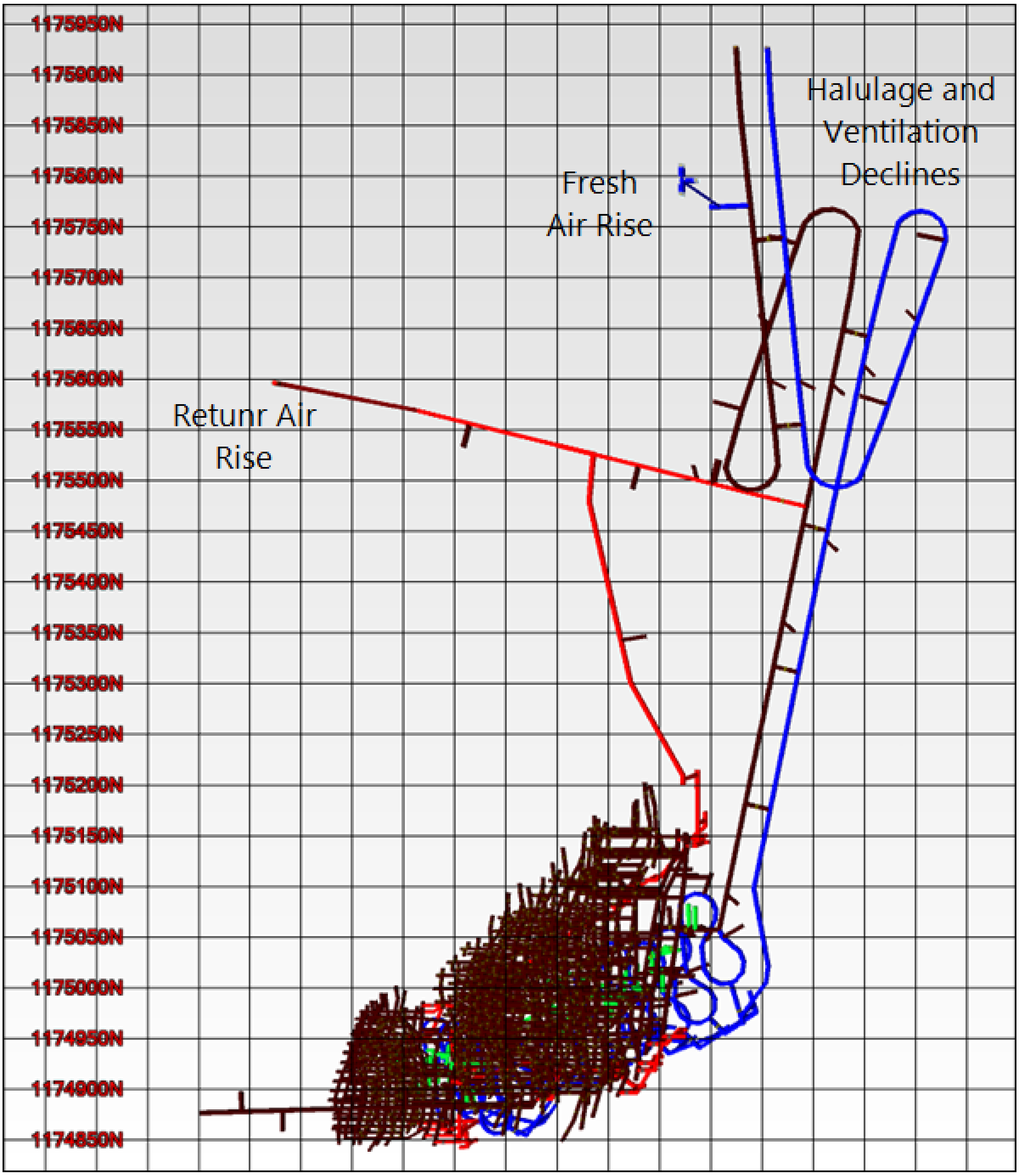

Figure 16.55: Primary Ventilation System – Plan View

16.8.6.11 Auxiliary (Secondary) Ventilation

Declines and associated excavations, being dead-end headings, must be ventilated using auxiliary (secondary) ventilation systems. Auxiliary (secondary) fans should be installed in known locations of fresh air, upstream of the furthest extremities of the primary ventilation circuit. To minimise the risk of recirculation, it is common practice to maintain a minimum distance of 20 to 30 metres between the fan intake location and the primary exhaust connection or the intersection of returning air.

Fan locations should be strategically planned, and the height of the drive/excavation should be increased to ensure that fans can be installed outside the operating envelope of mobile equipment, reducing the risk of damage.

During the early development of the decline, a fan must be located at least 50 metres outside the portal to limit the possibility of recirculation. These systems will be necessary until the primary ventilation circuit is established, which will require advancing approximately 700 metres.

A typical arrangement for such installations involves mounting fans on a purpose-built frame, elevating the fan and duct to the desired height relative to the tunnel. This setup reduces shock losses in the system due to direction changes in the duct and provides protection from mechanical damage. Figure 16.56 illustrates a typical layout of such a system.

Figure 16.56: Photo of an Auxiliary Fan mounted on a Support Structure (Mintek Australia Pty Ltd)

16.8.6.12 Level Ventilation

Auxiliary fans, located on the declines, will facilitate the re-use of air. Fans installed at the FADs will draw uncontaminated intake air directly from the ventilation drive. Fresh air will be delivered to the ore drives via 1,400 mm round ventilation ducting from these auxiliary fans. Vitiated air will be returned through in-level regulators (DBRs) to the main RAR.

Typical secondary ventilation system layouts for development and production levels are shown in Figure 16.57 and Figure 16.58.

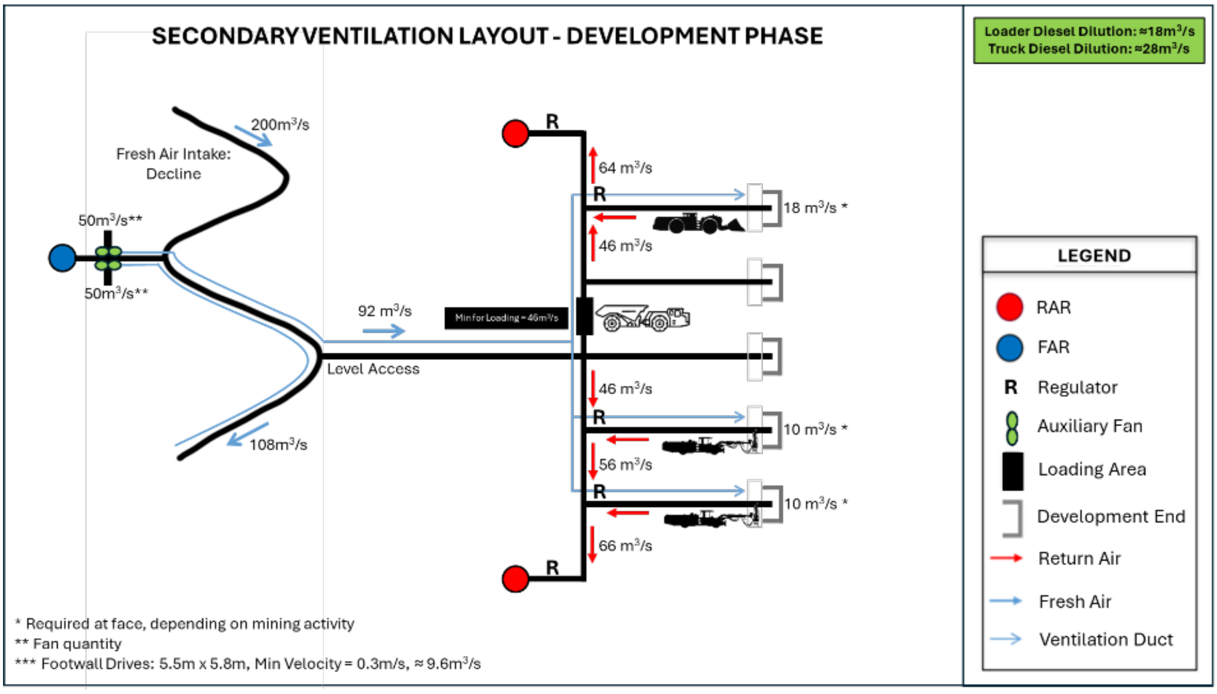

Figure 16.57: Secondary Ventilation Layout – Development Phase (Orelogy 2025)

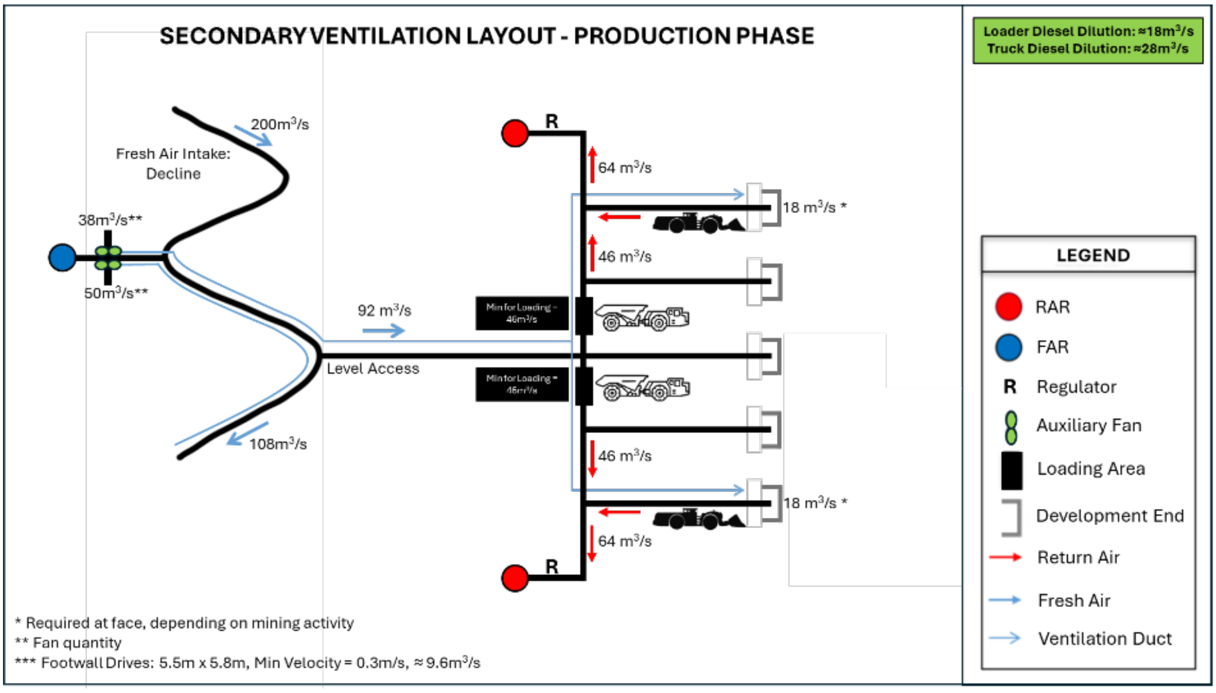

Figure 16.58: Secondary Ventilation Layout – Production Phase (Orelogy 2025)

For the development of working areas, both 1,200 mm and 1,400 mm round ventilation ducts will be utilised. Specifically, the 1,400 mm ventilation bags will be deployed during the initial decline and access development phases, providing ample airflow to support ongoing activities. As development transitions into the ore drives, the use of 1,200 mm round ducting will be implemented. This 1,200 mm ducting not only ensures adequate ventilation but also allows for sufficient clearance between the duct and a loader, as shown in Figure 16.59.

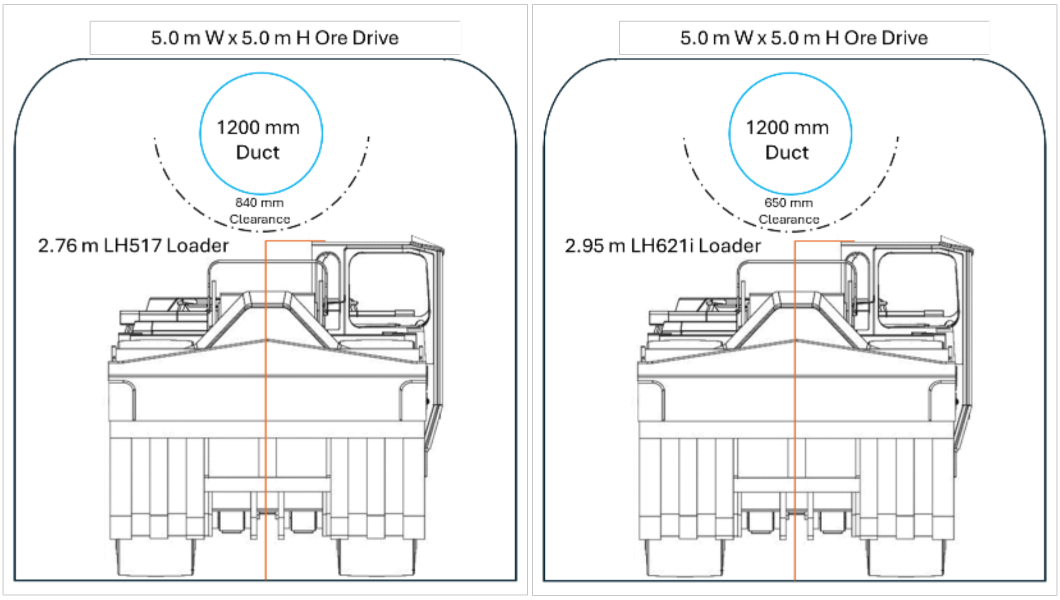

Figure 16.59: Ventilation Duct Clearance (Orelogy 2025)

16.8.6.13 Ventilation modelling

The ventilation circuit was modelled using Ventsim® software to determine operability and estimate the required primary fan duties. Key considerations in the modelling included:

- Airway size, based on proposed design dimensions.
- Friction factors and resistances, derived from excavation methodology and accepted industry design values.
- Ventilation controls, assumed to be of high quality (e.g., doors, walls).
- Emergency egresses, with no dedicated flow allocated.

An airflow of approximately 462 m^3/s was distributed throughout the mine as shown in Figure 16.60. This figure includes inferred stopes (below 9,900RL) which are excluded from the Mineral Reserve and LOM production schedule.

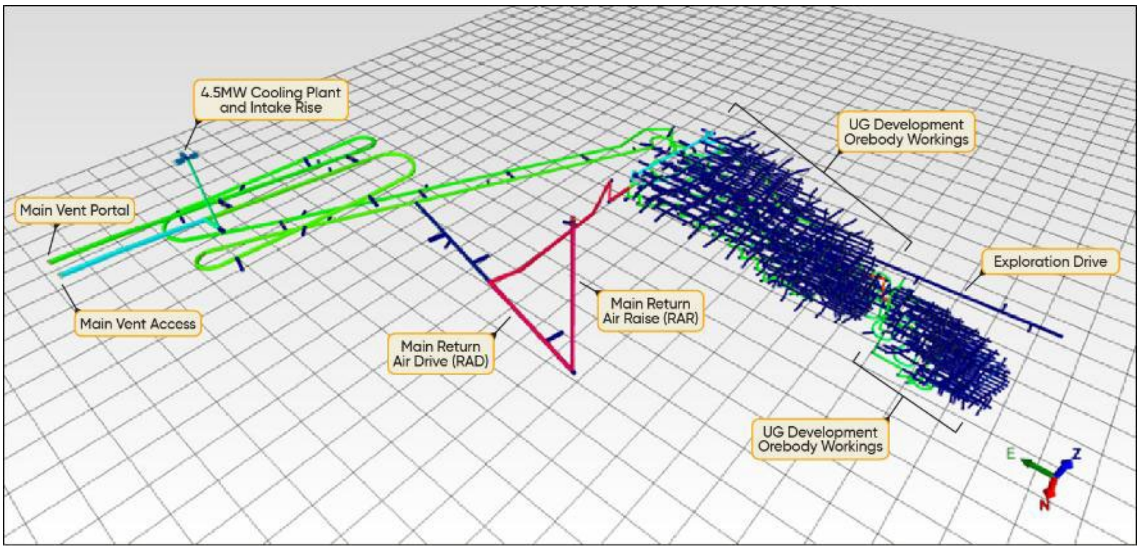

Figure 16.60: Airflow Simulation using Ventsim® Software

16.8.6.14 Primary Fan Duties

The primary fan duties will change throughout the mine's life, increasing as mining progresses. The peak primary airflow is capped at 460 m^3/s.

The initial ventilation strategy involves installing four single 110 kW fans within a bulkhead at the ventilation decline portal. This temporary installation will provide airflow for the main decline and main RAR development. Once the main RAR reaches the surface and the permanent installation is commissioned, the primary fans at the ventilation portal will be decommissioned. At this stage, the ventilation portal/drive will convert to a FAD and will no longer function as a RAD.

The performance specifications for both the temporary and permanent primary fan installations were estimated and simulated using Ventsim, with the results summarised in Table 16.28. This table details the specifications for the primary fan options selected for the temporary and permanent installations. The fan curves for the two options are shown in Figure 16.61 and Figure 16.62.

Table 16.28: Primary Fan Specification

Parameter	Specification
Temporary Installation	
System	(4) Parallel Fan Systems
Pressure Requirement	1.5 KPa
Airflow Requirement	180 m^3/s
Installation Power	440 kW
Permanent Installation	
System	(3) Parallel Fan Systems
Pressure Requirement	5.2 KPa
Airflow Requirement	460m^3/s
Installation Power	3750 kW

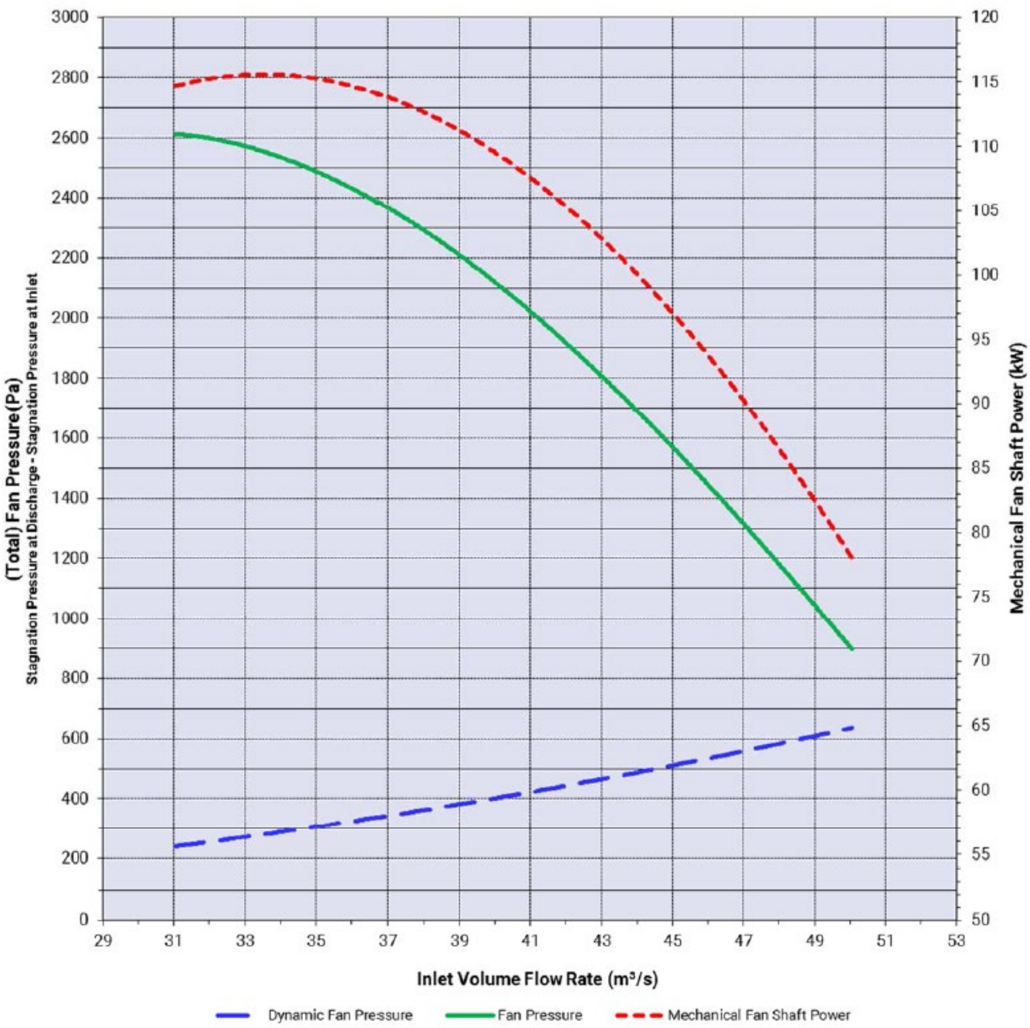

Figure 16.61: Temporary Primary Fan Installation – Example Fan Curve (ClemCorp Australia Pty Ltd)

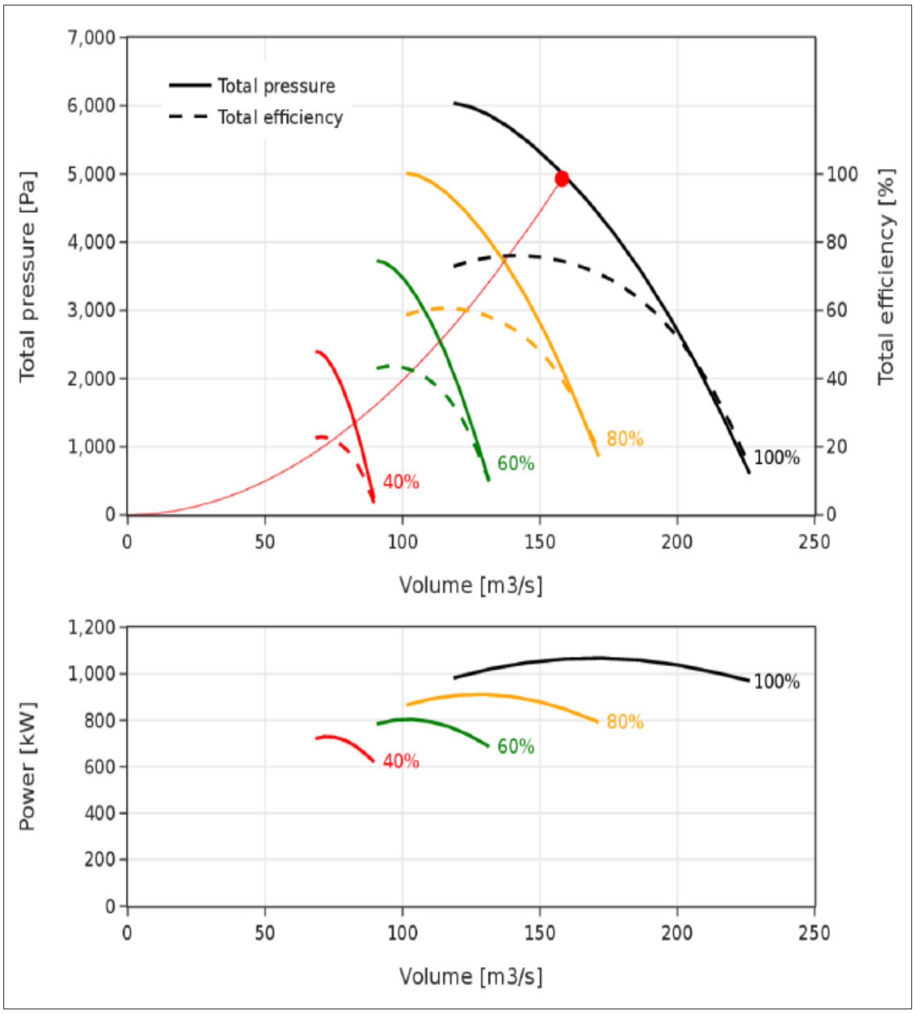

Figure 16.62: Permanent Primary Fan Installation – Example Fan Curve (Mintek Australia Pty Ltd)

To ensure adequate ventilation during the initial main decline and ventilation drive development, two twin 110 kW secondary fans will be installed at each Portal. These fans will provide sufficient ventilation until the temporary primary exhaust system is established.

16.8.6.15 Auxiliary Fans

Auxiliary fan duties vary based on the application and system requirements. Key considerations include:

- Airflow requirements of the working area to be ventilated.
- The number of work areas to be ventilated.
- Distance from the primary fresh air source.
- Type and condition of the duct to be used.

16.8.6.16 Decline Development Systems

Decline development activities impose the highest auxiliary ventilation requirements due to the concurrent operation of both loaders and haul trucks. Peak demand at the face is approximately

46 m^3/s. In the Australian mining industry, it is common practice to use either single or twin stage 1,400 mm diameter fans, each stage fitted with 110 kW motors, combined with 1,400 mm diameter flexible duct as the standard for decline development ventilation. However, increasing equipment engine power and a better understanding of the health effects posed by diesel-engine exhaust emissions have shown this specification to be inadequate for duct lengths beyond 300 m due to insufficient airflow delivery at the working face. As a result, twin fans and ducts are typically installed in parallel for lengths exceeding this distance. Additionally, high-quality, low-leakage duct should be used to minimise leakage and supply more air to the working face.

16.8.6.17 Production Level Auxiliary Ventilation Systems

Each production level will have a dedicated secondary ventilation installation to supply fresh air to the ore drives. Ventilation ducting will run from the FAR bulkhead at the main decline along the level access, then branch out at the ore drive junction to serve each drive.

To ensure a clean, adequate air supply at the working faces, auxiliary fans will be installed in the FAR bulkheads accessed via the main decline. These fans will supply fresh air directly to the work areas while preventing dust and other contaminants from being recirculated.

The pressure requirements of auxiliary fans vary based on the length of the ore drives, which can range from less than 60 m to 300 m from the level access. For concurrent operation in a pair of ore drives, one 200 m long and the other 150 m long, a twin 110 kW fan (1 stage used), combined with 1,200 mm and 1,400 mm diameter lay-flat or flexible ducting, will be sufficient. Given the ventilation demands of this project, twin 110 kW fans are recommended during the development phase to maintain consistent and effective airflow, particularly when multiple headings are active on a working level. If additional headings need to be ventilated beyond the capacity of the twin 110 kW fan setup, an additional fan must be installed in parallel to supply the extra volume of air.

16.8.6.18 Ancillary Activity Auxiliary Ventilation Systems

Minimum airflow requirements in areas with ancillary works, such as exploration activities, are lower than those in active mining areas due to reduced diesel equipment usage. In these cases, airflow demand is determined by minimum standards and climatic conditions, and the supplied volume must be sufficient to generate an average air velocity of at least 0.3 m/s.

16.8.6.19 Auxiliary Fan Fleet Requirements

Auxiliary fan requirements will vary throughout the project lifecycle depending on the stage of the mine and the specific activities being conducted. The calculation involves summing the number of active loaders, jumbos, long hole drills, and development charge equipment in operation during a given period, while subtracting one to account for the assumption that two machines will typically be working within the same active area. Operational auxiliary fan requirements for the Project are shown in Figure 16.63.

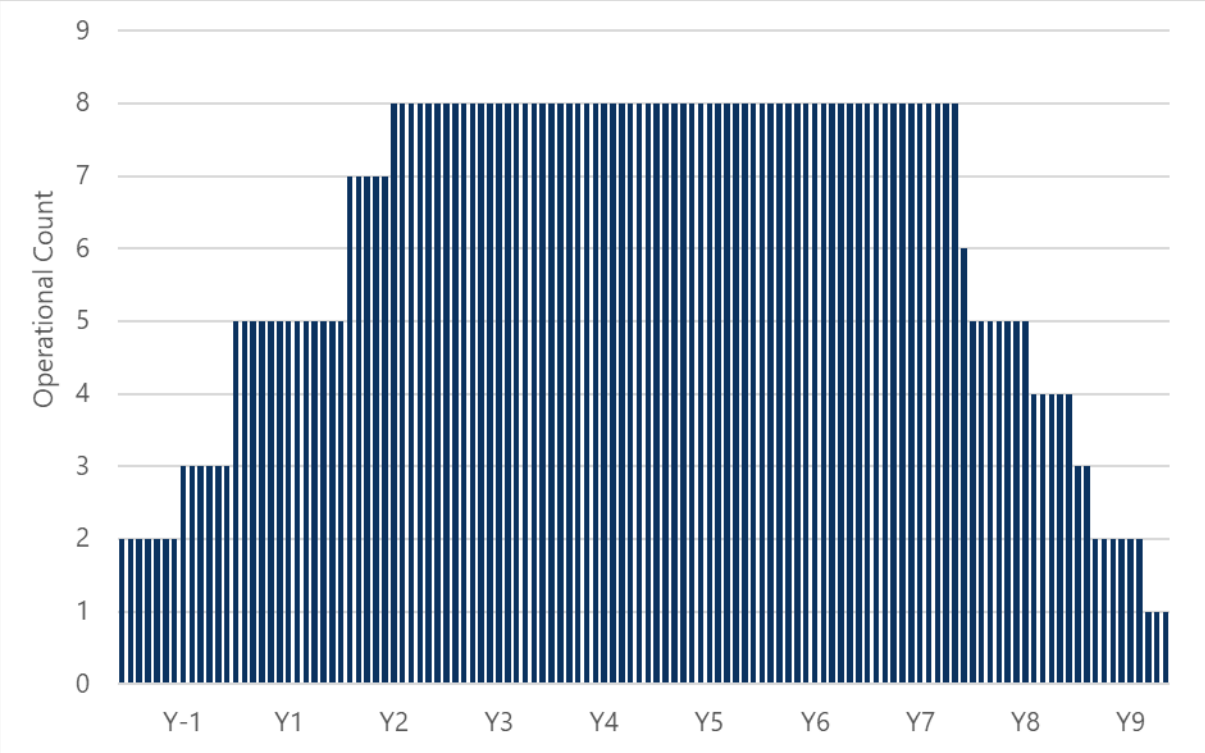

Figure 16.63: Operational Auxiliary Fan Requirements

Natural airflow must be maintained between each level access and the emergency egress system. If natural airflow cannot be sufficiently maintained, a small auxiliary fan will be required to ensure proper ventilation to the emergency egress system. Additionally, consideration should be given to potential changes in equipment or mining conditions that may necessitate adjustments to fan capacity and placement to maintain adequate ventilation and safety standards throughout the mine's life.

16.8.6.20 Heat Loads & Cooling Requirements

The overall heat load for an underground mechanised mine can be broken down into the following main components:

- Mobile equipment converts input energy into mechanical power, with any unused output and system losses dissipated as heat. For diesel-powered units operating on level ground, the heat released is equal to the product of the fuel mass flowrate and its calorific value.
- Surrounding rock contributes heat to the ventilation system through conduction from the virgin rock to the exposed surface and convection between the rock surfaces and the airflow. This heat load increases with depth due to higher virgin rock temperatures (VRT) and more exposed surface area. It also depends on the excavation age, with newly exposed rock transmitting heat more rapidly, which gradually reduces as the rock cools. The thermal conductivity of the host rock is a key governing factor.
- Broken rock from mining activities also imposes a heat load, as it enters the mine at the local VRT. The material presents a large surface area and releases more heat with each handling event as additional surfaces are exposed. Although the rock cools close to ambient

temperature by the time it exits the mine, the initial load increases with depth due to higher VRT.

As intake air descends through the mine, it undergoes auto-compression, where potential energy is converted into internal energy (enthalpy), resulting in a rise in air temperature. This effect becomes more significant with increasing depth and contributes a substantial heat load to the intake ventilation system.

Secondary fans, which do not perform useful thermodynamic work, convert all supplied electrical energy into heat. The magnitude of this load depends on the number of operating fans and the ventilation control strategy in use.

Analysis of the mining schedule, equipment fleet and weather data reveal that the underground mine will require a maximum of 4.5 MWBAC of air cooling over the LOM. To verify the cooling needs, heat simulations were performed using Ventsim™ Design. The heat loads/sinks considered for the study were:

- Auto-compression (load).
- Mobile equipment (load).
- Broken rock (load).
- Surrounding rock (load).
- Secondary fans (load).
- Natural air-cooling capacity (generally a sink).

16.8.6.21 Heat Design Criteria

The heat design criteria used for heat is summarised in Table 16.29.

Table 16.29: Heat Design Criteria

Parameters	Unit	Value	Comment
Summer Surface Design Temperature	(°C WBT / °C DBT)	23.3 / 30.1	Value represents the 95th percentile, derived from historical data sourced from multiple online resources, in which monthly temperatures were averaged across each year.
Barometric Pressure	kPa	101.3	Value derived from multiple online sources specific to the Kouroussa region."
Design UG Reject Temperature	°C WBT	28.5	Typical industry value
Surface Rock Temperature	°C	27	SRT calculated using the average annual temperature of 26.7 °C at a depth of 15 m below the surface."
Geothermal Gradient	°C/100m	1.85	Value provided by PDI
Conductivity	W/mK	4	Hartman
Specific Heat Capacity	J/kgK	837	Value provided by PDI
Rock Density	kg/m3	2800	Value provided by PDI

16.8.6.22 Monthly Design Temperatures

Mining and development activities were simulated for selected snapshot years, with heat loads from major operating equipment integrated in accordance with the mining schedule. The design objective is to ensure that the reject wet-bulb temperature remains below 28.5°C as air leaves the working face and returns to the main exhaust.

To ensure modelling accuracy, representative ambient conditions were used to reflect seasonal temperature variations. The average temperatures, shown in Table 16.30, were calculated using data from various online weather sources for the Kouroussa region.

Table 16.30: Monthly Ambient Temperatures

Description	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Design
Surface WBT(°C)	14	16	19	22	23	23	23	23	23	23	20	15	**23.3**
Surface DBT (°C)	25	27	30	30	29	27	25	25	25	26	26	25	**30.1**

16.8.6.23 Monthly Cooling Estimates

The cooling requirements for the underground mine are shown in Figure 16.64.

Initially, only 1.5 MWBAC is required, with demand increasing to 4.5 MWBAC as the project progresses and the underground mine becomes deeper. A shortfall in cooling capacity is expected during certain months due to increased production demand and equipment usage during that period. During this time, heat management strategies must be implemented to manage the heat risk and maintain safe working conditions. It is expected and confirmed through Ventsim modelling that wet bulb temperatures should not exceed 29°C during these periods. Figure 16.64 also illustrates the heat load at reject temperatures of 28.5°C and 29.0°C respectively, highlighting the sensitivity of cooling demand to small changes in temperature.

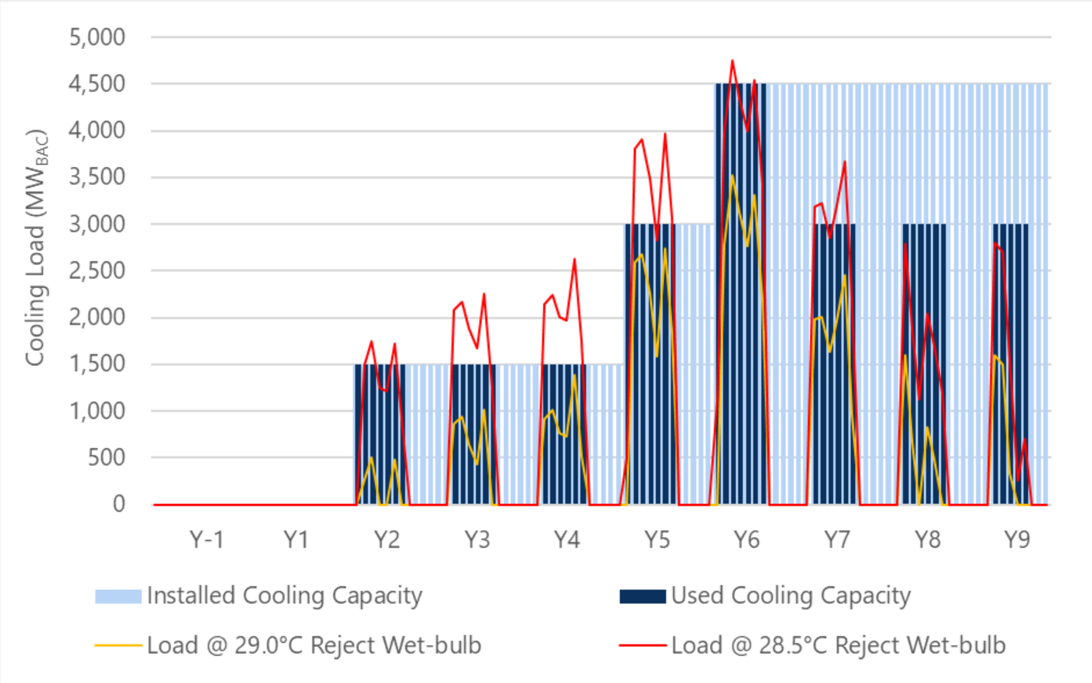

Figure 16.64: Bankan Gold Project Monthly Air-Cooling Requirements

16.8.6.24 Heat Loads and Heat Balance

Figure 16.65 shows the estimated heat load distribution for the underground mine during Year 5 of operation. The two largest contributors to the heat load are diesel equipment (31%) and auto-compression (54%).

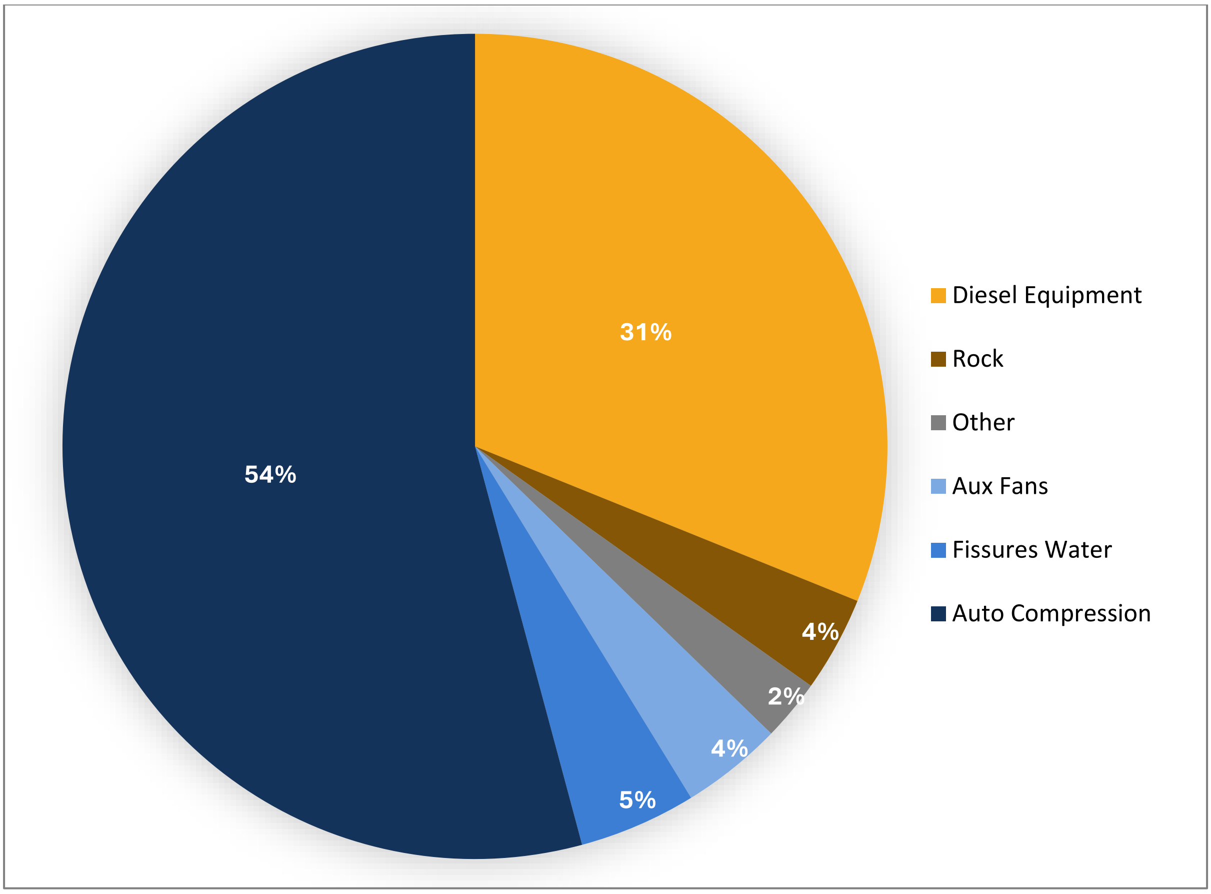

Figure 16.65: Bankan Gold Project Heat Load Distribution

The overall heat load is balanced by the natural cooling from the ventilation air, service water (0.4 t of water per ton of rock mined assumed) and mechanical cooling, as shown in Figure 16.66.

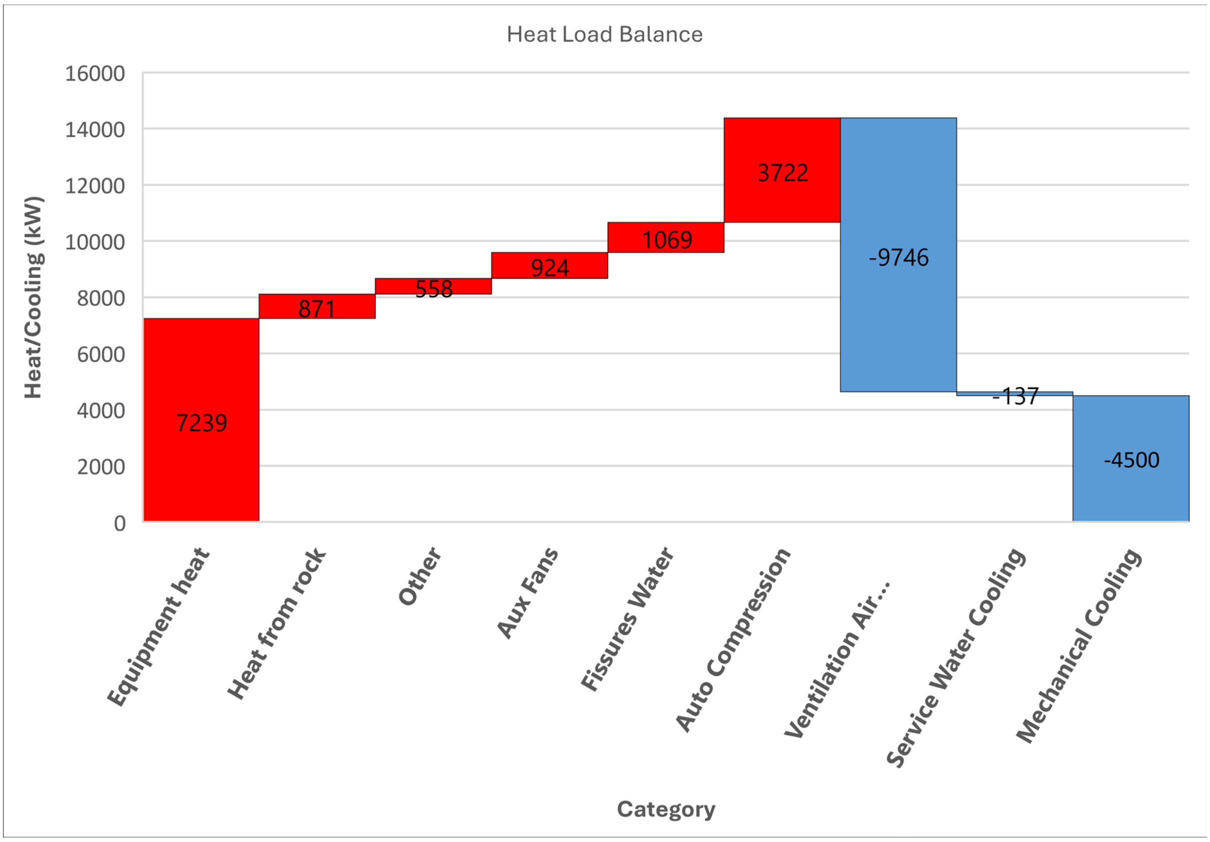

Figure 16.66: Bankan Gold Project Heat Load Balance

16.8.6.25 Refrigeration Strategy

A surface R134a refrigeration plant is proposed for the underground mine comprising modular chillers and condenser cooling towers (CCTs) to supply chilled water to a bulk air cooler (BAC). Initially designed to meet a 1.5 MW cooling demand, the system is modular and scalable to accommodate increased requirements as the project progresses. The BAC is divided into individual cells, each capable of delivering 1.5 MW of cooling, enabling a phased implementation. Chillers are energy-efficient, oil-free, water-cooled units equipped with advanced control systems for reliable and flexible operation. Each chiller is paired with dedicated CCTs to manage heat rejection, using make-up water primarily sourced from the BAC and supplemented with potable or treated mine water when necessary. A typical general arrangement drawing of the BAC is shown in Figure 16.67, while Figure 16.68 presents the general arrangement of a typical refrigeration plant, including a plantroom. Figure 16.69 provides a plan view showing the location of the refrigeration plant in proximity to the GBE portals.

It should be noted that the future of R134a refrigerant is unknown due to its high global warming potential (GWP). The time at which this refrigerant might be phased out is still uncertain, and suppliers are currently researching potential replacement refrigerants. The other alternative is an ammonia plant, but it would have to be located a minimum of 200 metres away from any surface infrastructure, intake shaft, or similar facilities.

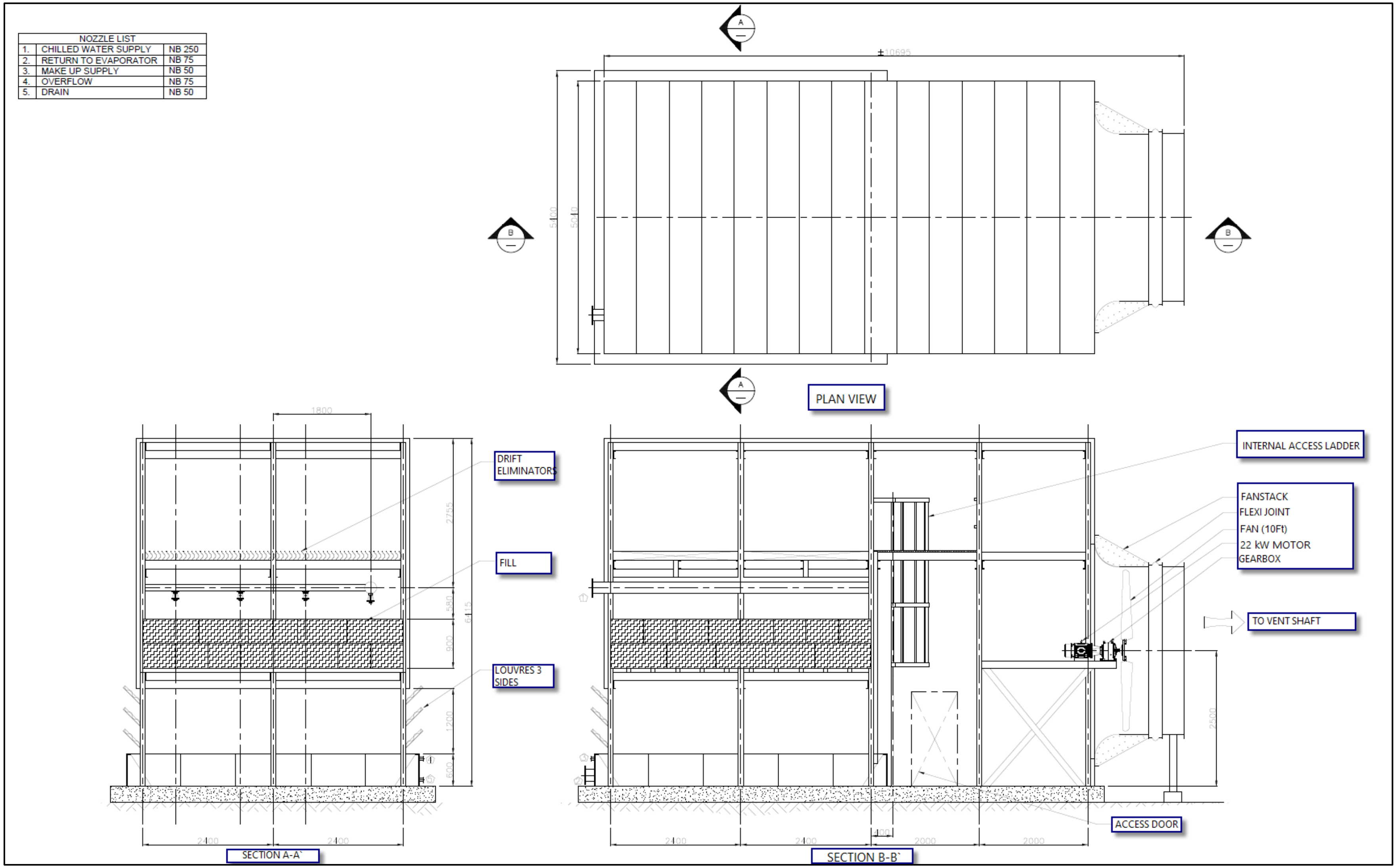

Figure 16.67: General Arrangement Drawing of a Typical Bulk Air Cooler (BAC) (IWC 2025)

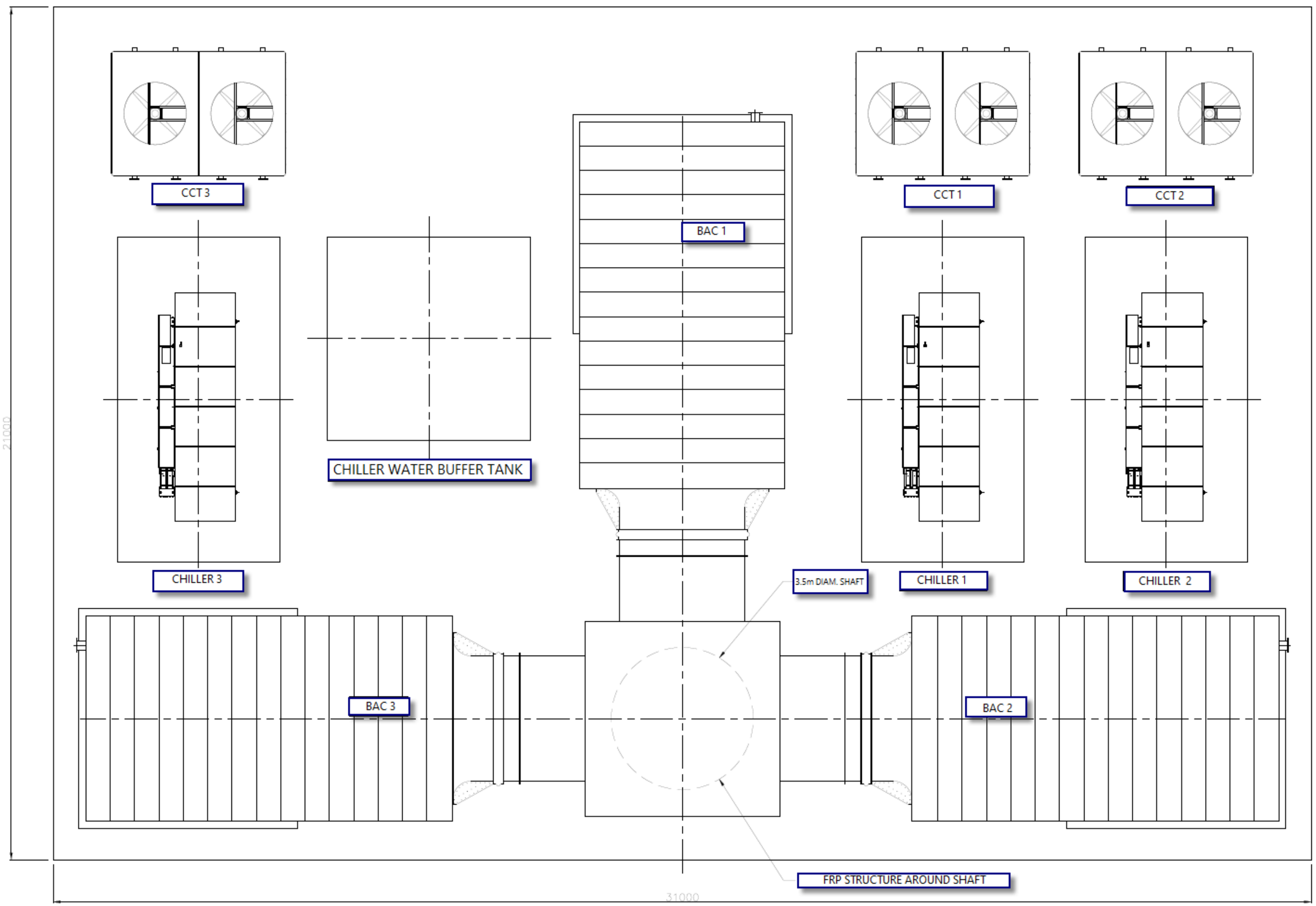

Figure 16.68: General Arrangement Drawing of a Typical Refrigeration Plant (IWC)

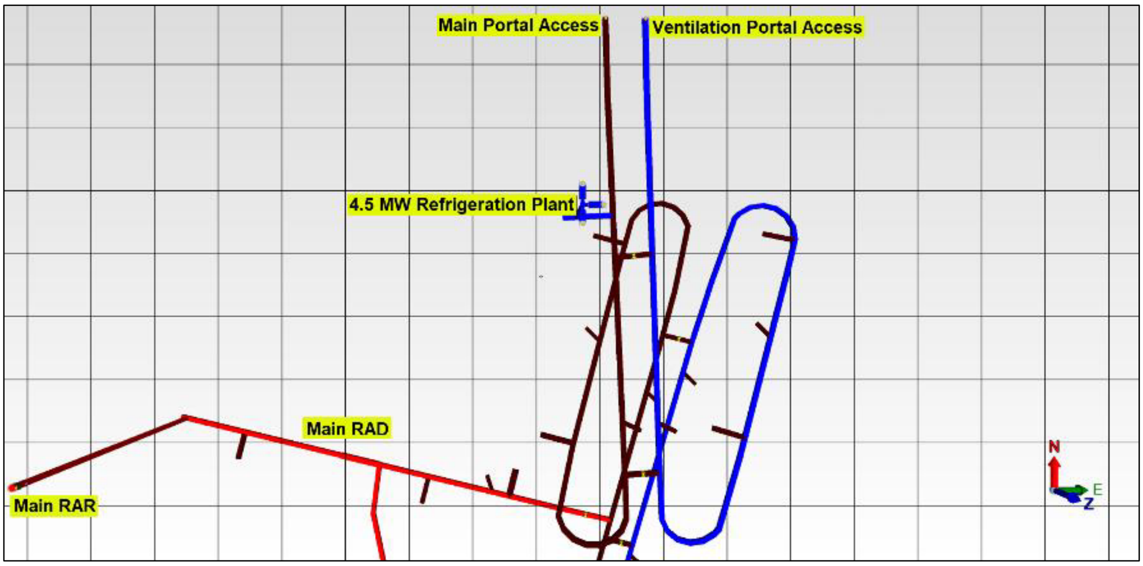

Figure 16.69: Location of 4.5 MW BAC Refrigeration Plant – Plan View

16.8.7 Paste Fill

Paste fill has been selected as the primary backfilling method for the underground operation, forming a critical role in the mine sequence, production continuity, and to maximise overall orebody recovery. Stope geometries, as detailed above, along with results from comprehensive tailings characterisation and rheology testing, have informed the design criteria for paste fill volumes, binder content, and target fill strengths, ensuring compliance with both geotechnical and operational requirements.

To tailor fill strength to varying stability needs, stopes have been classified as either undercut or non-undercut. Undercut stopes necessitate higher strength paste to maintain stability and safety during subsequent extraction phases. As outlined previously, a predominantly bottom-up extraction sequence will be employed. This approach minimises exposure to paste walls, reduces dilution risk, and enables the use of larger stope dimensions, ultimately optimising binder usage and enhancing overall resource recovery.

Figure 16.70 shows the stopes to be undercut, in red, looking east. Stopes to be undercut will be backfilled with a 10 m layer of high binder concentration paste to create a stronger “plug”, with the remaining volume filled using lower binder concentration fill. Figure 16.71 shows the paste fill strength and mine sequence. These figures include inferred stopes (below 9,900RL) which are excluded from the Mineral Reserve and LOM production schedule.

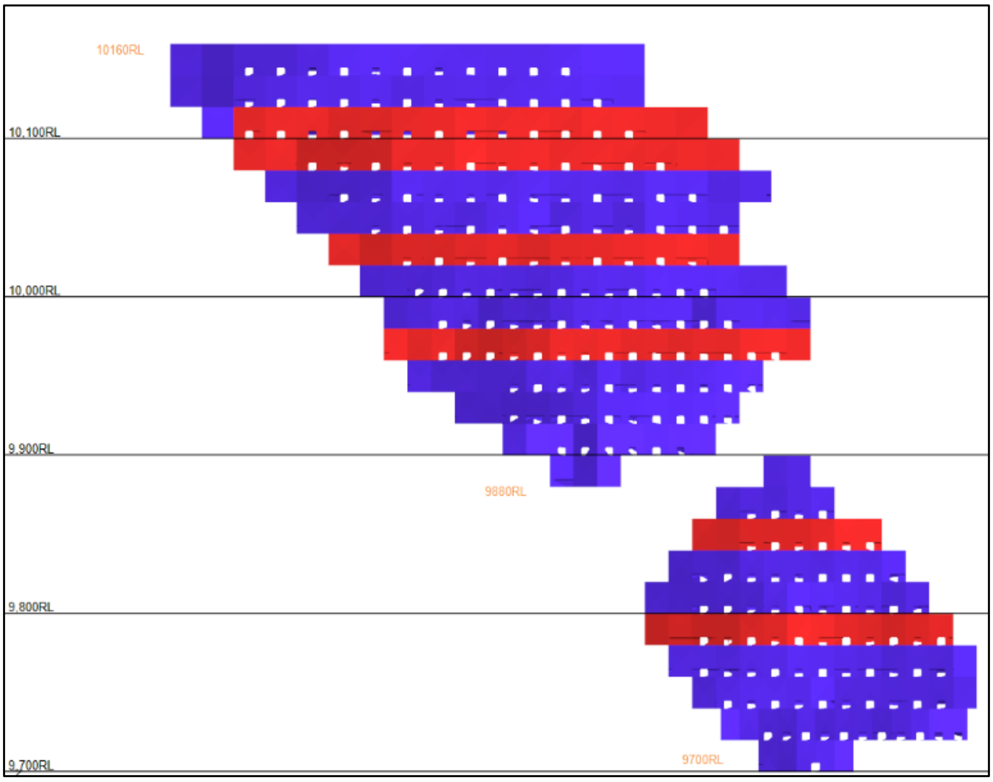

Figure 16.70: Undercut Stopes in Red – Looking East

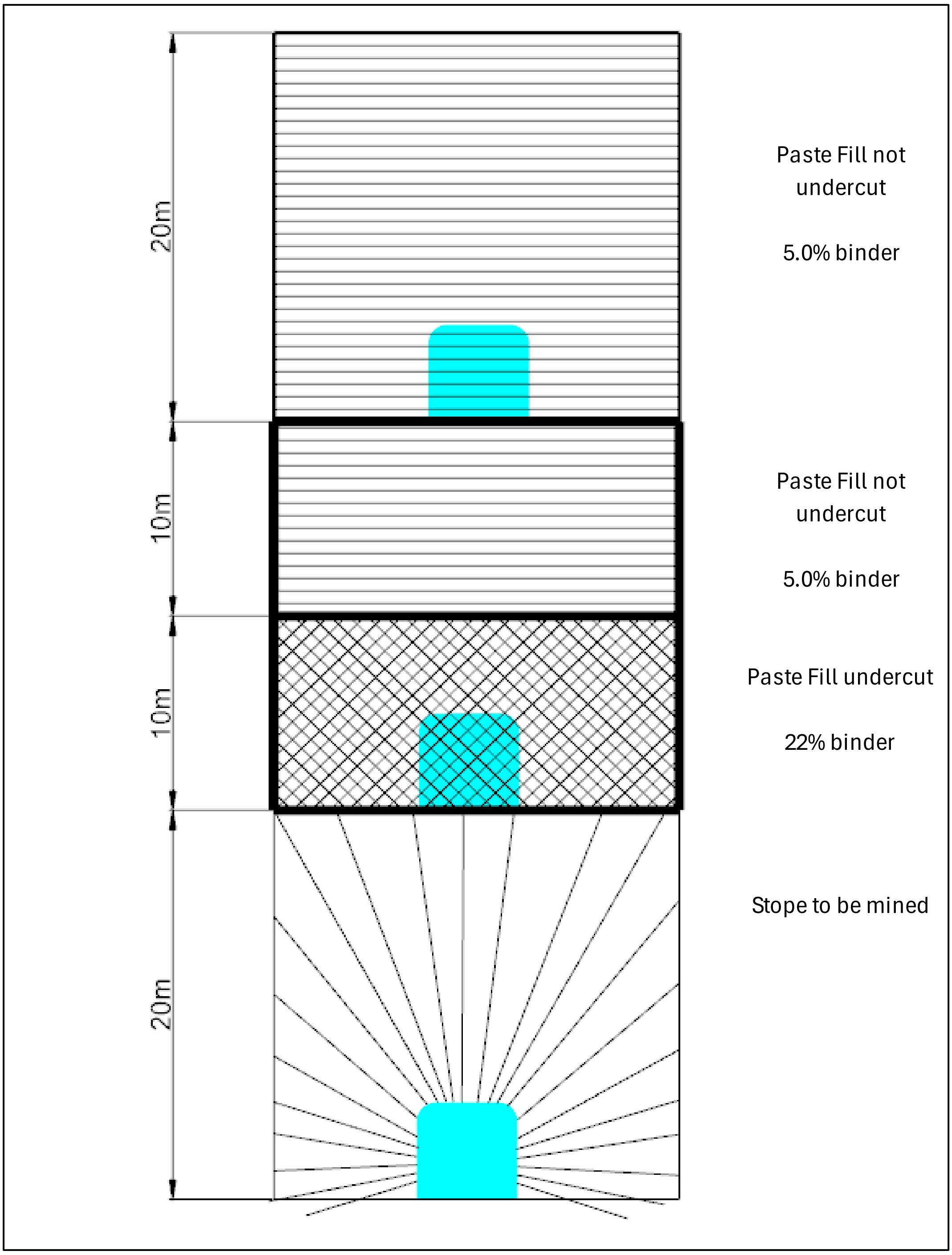

Figure 16.71: Paste fill Strengths and Mine Sequence (Orelogy 2025)

Stopes were analysed based on their geometry and strength requirements were calculated based on different stope dimensions. Table 16.31 shows the paste strengths by dimension for the stopes.

Table 16.31: Paste Strengths by Dimension (Minefill Services, 2025)

Stope Width (m)	10	15	20
Vertical Strength Upper 10 m (kPa)	150	180	200
Vertical Strength Lower 10 m (kPa)	200	260	300
Horizontal Strength (kPa)	1,700	2,500	3,800

Paste fill lines will generally be located in the north end of each level, with the paste fill surface plant located northwest of the NEB pit.

Once paste line reaches the required level, it will be extended toward the ore drive located above the target stope to be filled, allowing controlled placement of fill to ensure tight and complete filling is achieved. This approach ensures structural integrity, effective void management, and optimised stope recovery.

Figure 16.72 shows the paste fill retiulation in cross-section view while Figure 16.73 shows the location of the paste fill plant on the surface. These figures include inferred stopes (below 9,900RL) which are excluded from the Mineral Reserve and LOM production schedule.

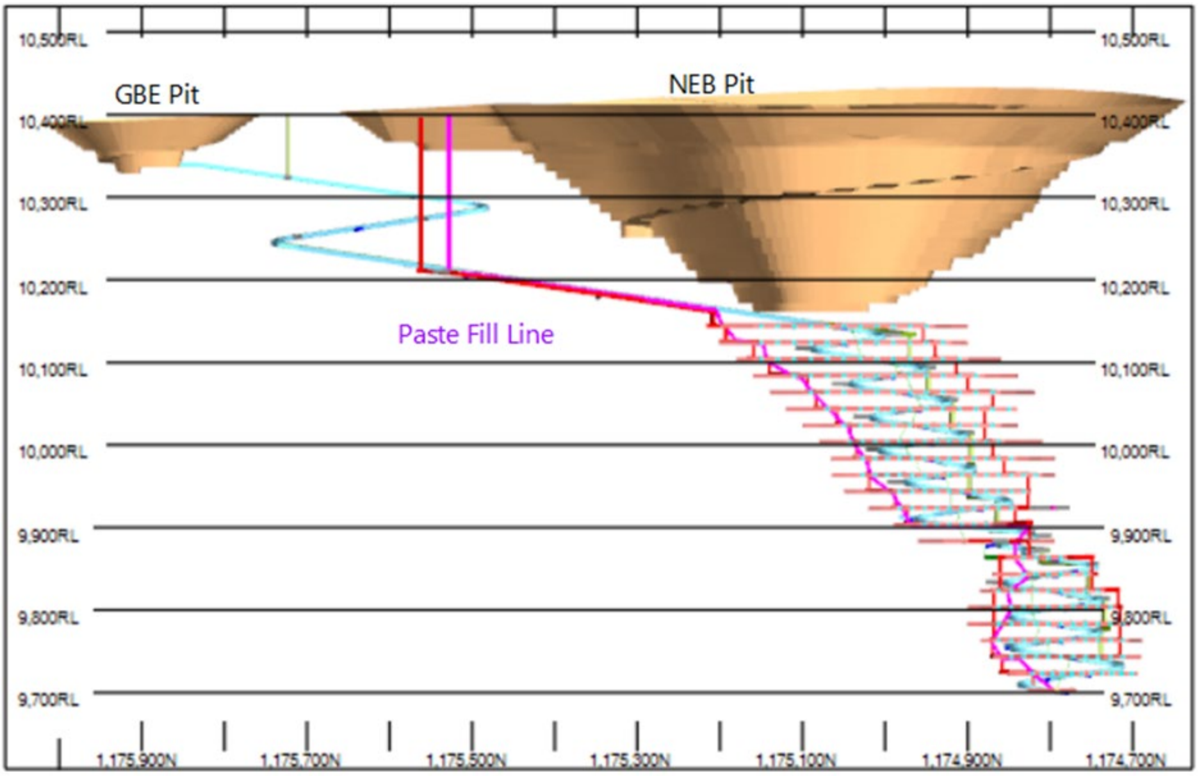

Figure 16.72: Paste Fill (Line in Pink)

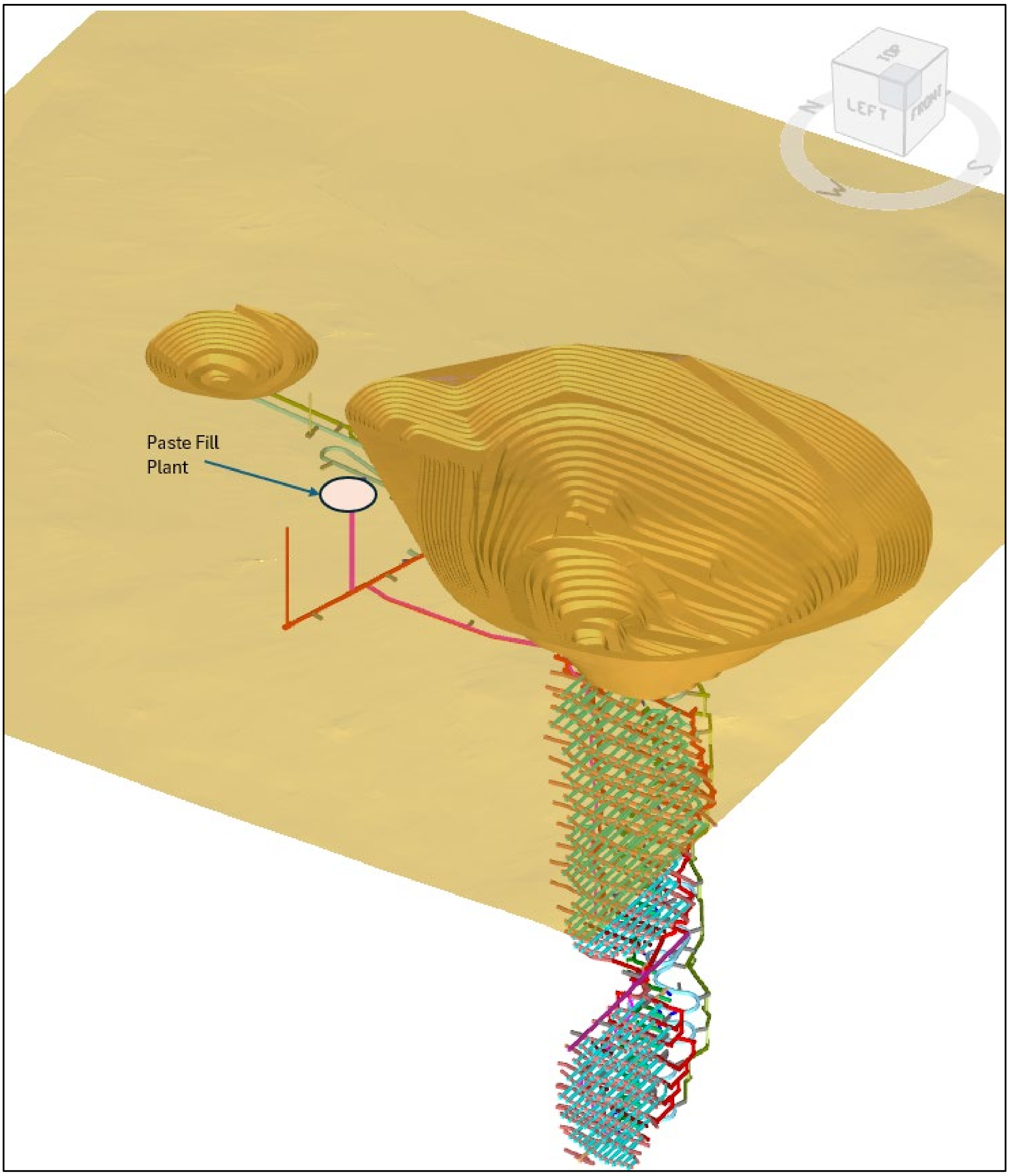

Figure 16.73: Paste Fill Plant Location

The paste plant is designed with a capacity of 80 m^3/h, with tailings transported by truck from the processing plant to the paste facility. To validate the efficiency of the paste reticulation system, a flow simulation was performed using the 10140 level layout, which represents the most challenging geometry in terms of horizontal to vertical distances. This model assessed friction losses and confirmed that paste flow can be achieved by gravity alone, thereby eliminating the need for booster pumps and simplifying operational requirements.

This infrastructure layout supports the mine's overall strategy of bottom-up extraction and tight backfilling, which is critical to maximising recovery, minimising dilution, and ensuring long-term stability of the underground workings.

Figure 16.74 (Minefill Services, 2025) shows the flow model to the 10140 mRL level.

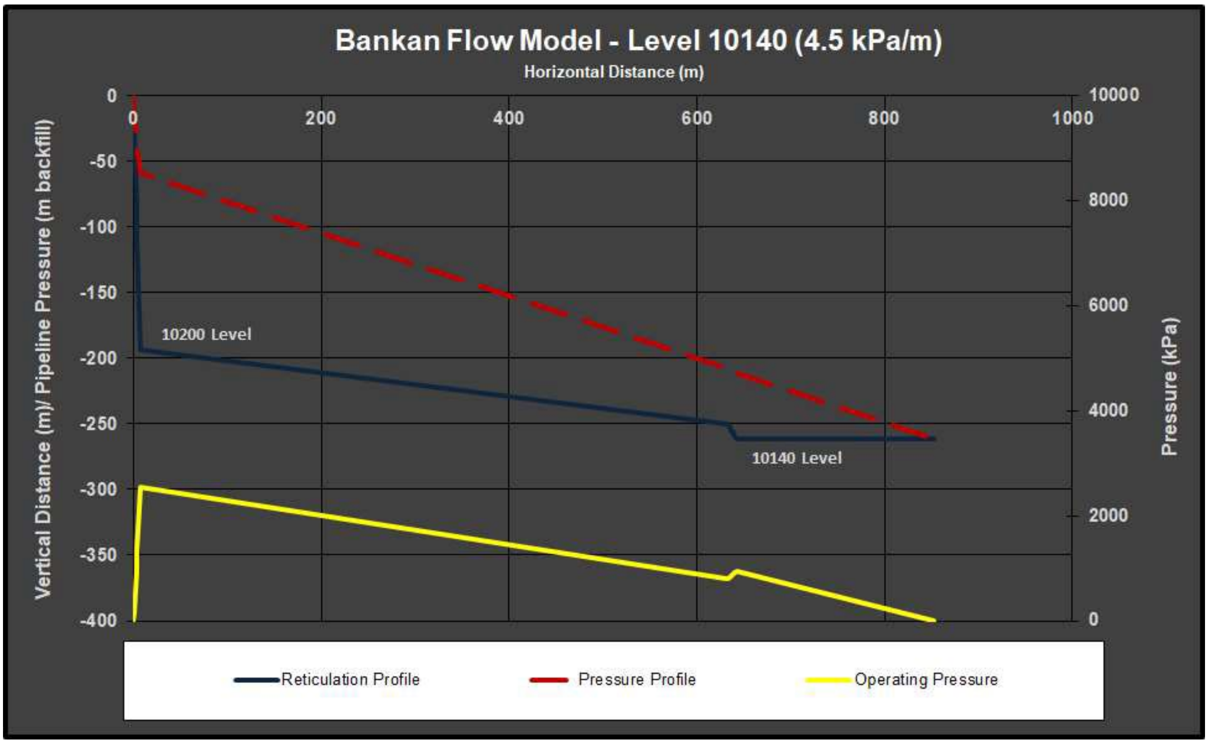

Figure 16.74: Flow Model 10140 mRL Level

When plotting the pressure difference on the tested level, a maximum friction loss of 4.5 kPa/m is observed, which falls well within industry standards. Additionally, as the mine extends to lower levels, the overall system pressure loss decreases. The maximum pressure of approximately 3,000 kPa occurs at the base of the surface borehole, also remaining within accepted industry limits.

16.8.8 Ground Stabilisation

Ground control is fundamental to maintaining a safe and efficient underground mining operation. Due to the geological conditions and the orebody's dip of approximately 45°, the ground control approach has been designed to provide stability across both development drives and production stopes, adapting to different ground conditions encountered throughout the mine.

16.8.8.1 Development Requirements

Development drives will be supported primarily using a combination of rock bolts and mesh throughout the mine. Fibrecrete will be required in the sill pillar drives. Support is planned for two different types of rock:

- Good quality rock, mesh surface support to within 3.5 m or less of the floor level, installed with pattern bolting using greater than 2.4 m long friction bolts to be installed at less than 1.3 m bolt spacings in the backs and sidewalls to within 2.0 m or less of floor level.
- Bad quality rock, greater than 50 mm fibrecrete (greater than 32 MPa 28 day strength required) surface support to within 1.5 m of floor level or less, installed with pattern bolting using 2.4 m or longer MD Bolts, installed at less than 1.2 m bolt spacings in the backs and sidewalls to within 1.5 m or less of the floor level.

Following geotechnical studies, the mine access decline has been repositioned to start in fresh rock through the GBE pit, reducing exposure to weaker weathered material and minimising ground support needs. This change improves development productivity and lowers operational risks. Ground support installation will be integrated into development activities, with development drill rigs responsible for both excavation and primary support installation. Regular ground condition monitoring and inspection will guide ongoing support requirements to maintain safe working conditions.

16.8.8.2 Stope Requirements

Stopes will require robust ground control due to the mining method and orebody geometry. Cable bolting is planned on the hangingwall of the stopes in contact with mafic rock, and additional support such as brow cables will be installed to control stability during blasting and extraction. A bottom-up mining sequence will help minimise dilution and maintain stope integrity, while paste fill will provide additional ground support by filling voids and reducing ground movement.

16.8.9 Dewatering

The primary underground dewatering system for the underground mine was designed to remove 40 L/s of water from underground, with redundancy in all three pump stations to accommodate higher inflows, due to the variability presented in the hydrogeology report and the weather pattern.

The system has been designed to manage operational and environmental water inflows, which is critical to operations in a region that experiences a monsoon wet season. The primary pumping system has been designed to not only to remove water generated from daily underground activities such as drilling, but also to manage additional inflows from surface runoff and water percolating through the rock mass.

The primary pumping system comprises of three underground pump stations situated and described below:

- Pump station 1, located off the return air drive on the 10,200 level. The pump station consists of 3 of WT114 helical rotor pumps, with 2 duty pumps and 1 standby pump. The duty flow is 40 L/s and the maximum designed flow is 99 L/s. The water will be pumped from this pump station up a dedicated DN200 SCH40 steel rising main to the surface (10,400 RL).
- Pump station 2, located on the 9,908 RL. This pump station comprises 3 of WT107 helical rotor pumps, with 2 duty pumps and 1 standby pump. The duty flow is 40 L/s and the maximum designed flow is 60 L/s. The water will be pumped from the 9,908 RL to pump station 1 (10,200 RL) via DN150 SCH40 steel pipes installed in services holes. Where steel pipe is unable to be installed, DN110 PN16 pipe will be used. This will comprise 3 of parallel pipes running from the top of the last steel rising main (~10,157 RL) up the return air drive to pump station 1, a distance of approximately 400 m and gradient of 1:7 upwards. The dynamic head is approximately 92 m and the PN16 pipe is adequately rated for the pressure.
- Pump station 3, located at the 9,760 RL. This pump station will comprise of 3 of WTX3 helical rotor pumps, with 2 duty pumps and 1 standby pump. The duty flow is 40 L/s with a maximum designed flow rate of 60 L/s. The water will be pumped from the 9,760 RL to pump station 2 located at the 9,908 RL via DN150 SCH40 steel pipes installed in service holes.

The upper-level pumps have been designed to cater for high rainfall events to prevent the bottom of the mine flooding, it is common for rain to be caught and directed to the upper levels for removal and thus the pump stations here typically have a higher capacity.

Once the water is pumped from the primary pumps up the rising main, it will exit near the paste plant and be delivered to a turkey's nest, approximately 300 m away, for settling and treating.

Figure 16.75 shows the locations of the 3 primary pump stations. This figure includes inferred stopes (below 9,900RL) which are excluded from the Mineral Reserve and LOM production schedule.

The secondary pumping system consists of sumps located throughout the mine for catchment of water and delivery to the primary pump stations. Along the decline, sumps are located every 40 vertical metres, and on the levels they are situated in the access. The level sumps have planned drain holes to minimise the usage of pumps throughout the mine life. Once a level is complete, the drain hole will be drilled, a mesh plug installed to prevent access by personnel, and the water will then be collected on the level below. The pump from the sump isn't required once the drain hole is in place and will be moved to the sump on the level below. Water collected in the sumps is then pumped to the closest primary pump station through appropriately rated poly pipe.

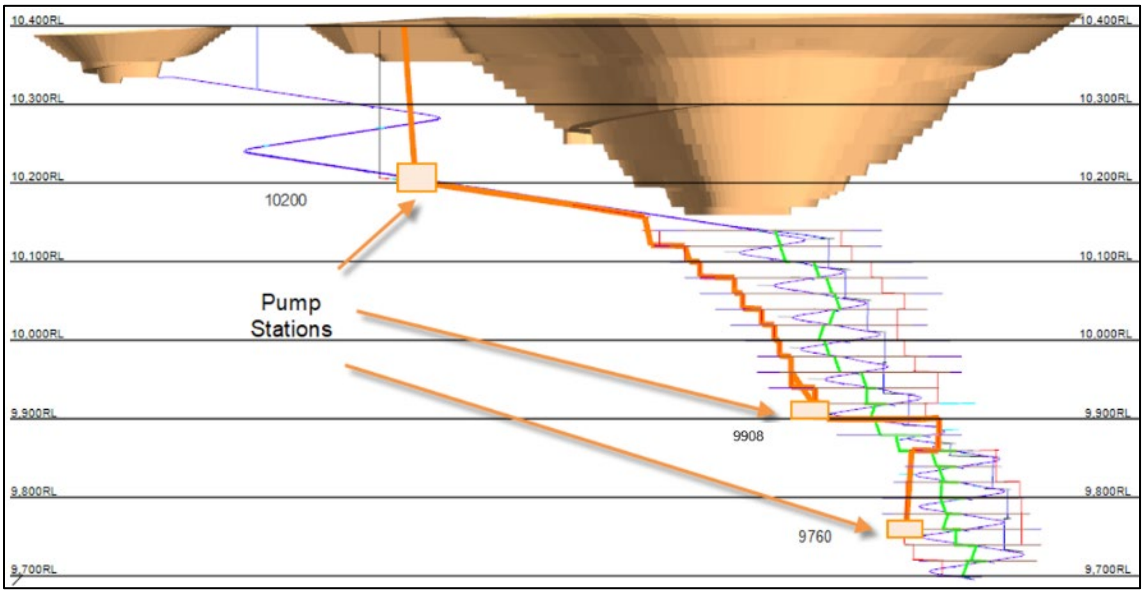

Figure 16.75: Primary Pump Stations and Dewatering Route

Primary pump stations will be mined to the dimensions of 8.0 m W by 6.0 m H at a gradient of 1:40 upwards and have a dedicated sump adjacent, to settle fines before delivery to the helical rotor pumps. Each primary pump station will have a concrete pad for the pumps and electrical installations. A monorail for motor removal for maintenance purposes will be installed in the backs. Figure 16.76 shows a typical pump station installation.

Figure 16.76: Photo of a Typical Underground Pump Station with Capacity of 60 L/s (Challenge Pumps Pty Ltd)

To ensure the mine water is as clean as possible prior to delivery to the primary pump station hopper tanks, the fines are settled in a sump adjacent to the pump station prior to being pumped into the hopper tanks.

As the underground portal is located inside the GBE pit, a sump was designed at the base of the pit to catch water and prevent it from flowing to the underground workings. This sump has a capacity of 8,800 kilolitres, and a 37 kW submersible pump will be located in the sump to pump the water to the turkey's nest on the surface for continuous sump dewatering.

Water level management will be the responsibility of operations, guided by clearly defined milestones and trigger points to ensure the safe and continuous functioning of the underground mine. These protocols will be communicated across all operational teams and maintained throughout the life of mine. Given that Guinea experiences an extensive rainy season, effective water management represents a significant operational risk and requires proactive planning and regular monitoring to prevent disruptions and maintain safety standards.

16.8.10 Power Supply

The initial power supply for the underground mine is planned to be supplied by generators with connection to the mine power grid scheduled for a later date.

The underground mine has been planned as a 1,000 V feed and an 11 kV to 1 kV step-down transformer will be utilised on the surface to achieve this.

Power will be supplied from the site grid to the underground mine substation, located near the paste plant, and delivered underground via service holes connecting to the 10,200 level. Figure 16.77 shows the planned underground power backbone reticulation route. This figure includes inferred stopes (below 9,900RL) which are excluded from the Mineral Reserve and LOM production schedule.

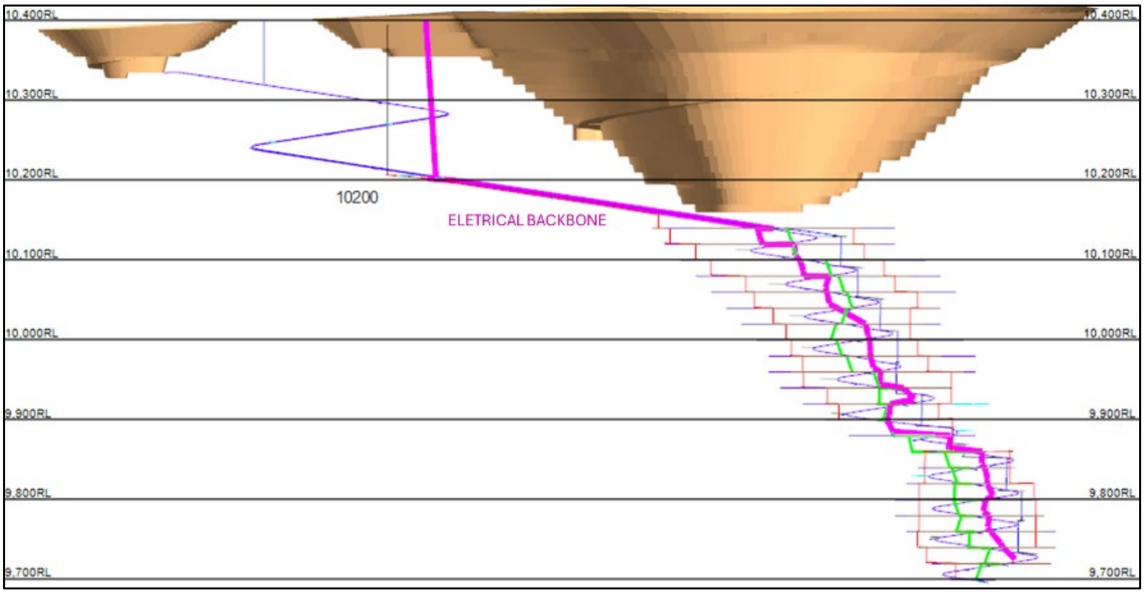

Figure 16.77: Underground Power Reticulation Backbone – Cross Section View

Surface power requirements will be distributed from the underground mine substation to the primary ventilation system and refrigeration plant.

Four 11 kV to 1 kV 2 MVA substations are required underground. These will supply eight 6-way distribution boards to supply the required power for up to eight levels, including primary pumping, ventilation and refuge chambers.

All refuge chamber drives have designed service holes to connect them, which will be used to distribute power between levels as shown above in Figure 16.77.

Figure 16.78 shows the areas on the surface requiring power and the surface layout displaying the locations for the paste plant (and underground substation), primary ventilation fans and refrigeration plant.

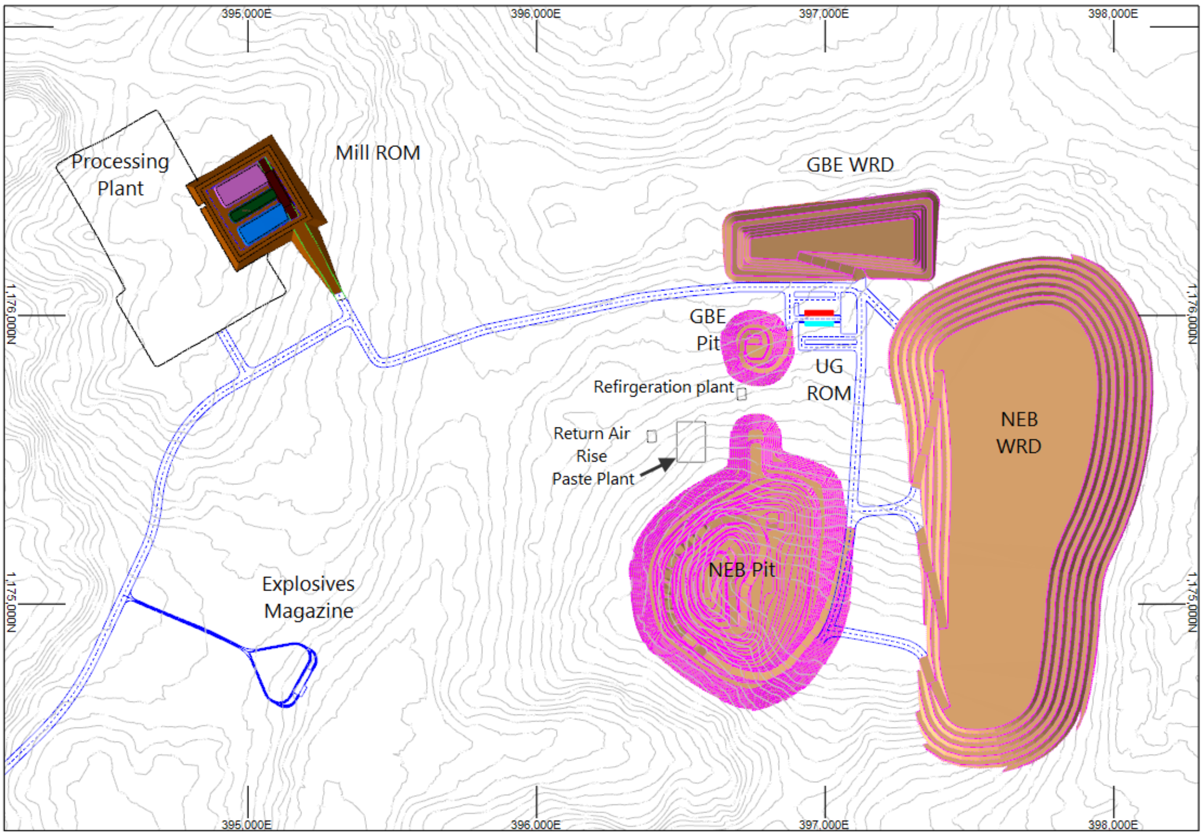

Figure 16.78: Surface Power Required Areas

The power requirements for the underground mine were calculated based on the mining equipment and infrastructure required to achieve the underground mining schedule. This information was calculated based on engine and equipment power ratings, productivity/utilisation, and an operating percentage of full load power to calculate an operating power for each unit. These calculations were checked and adjusted as required by Hahn Electrical, a specialist underground electrical consultant. The underground electrical reticulation requirements and associated electrical purchases were recommended by Hahn Electrical and used for the purposes of calculating capital and operating expenditure. The annual connected load is shown in Figure 16.79 and the annual GWh are shown in Figure 16.80.

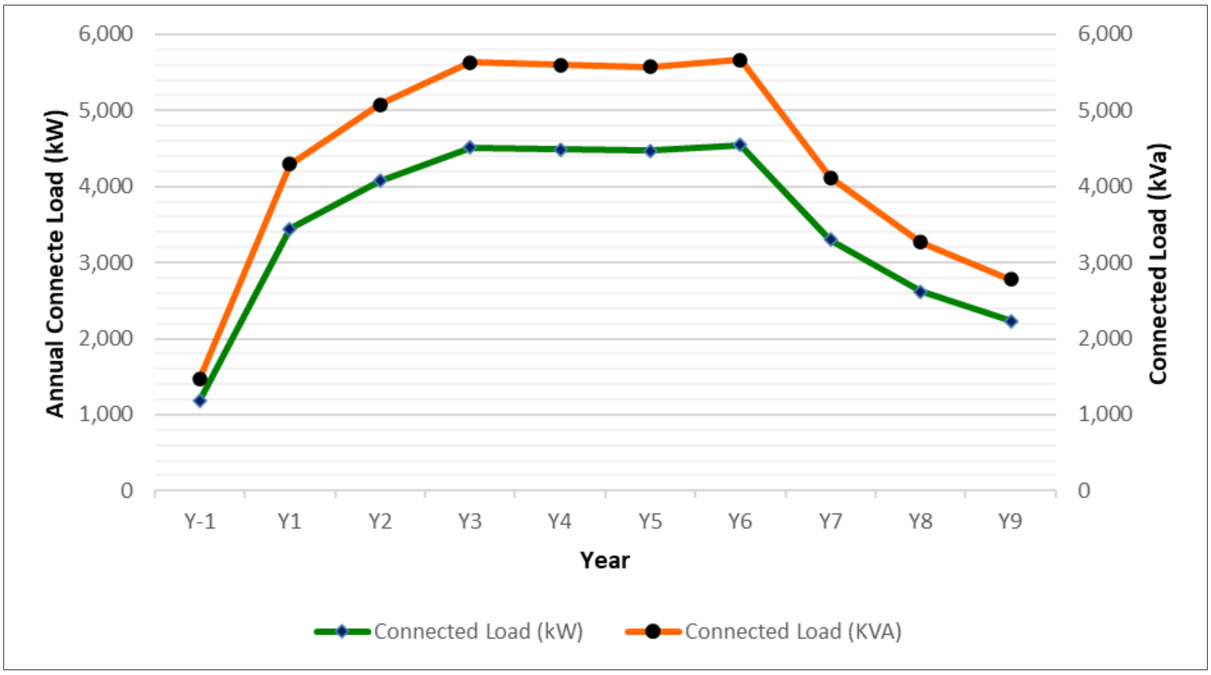

Figure 16.79: Annual Connected Load Bankan Underground

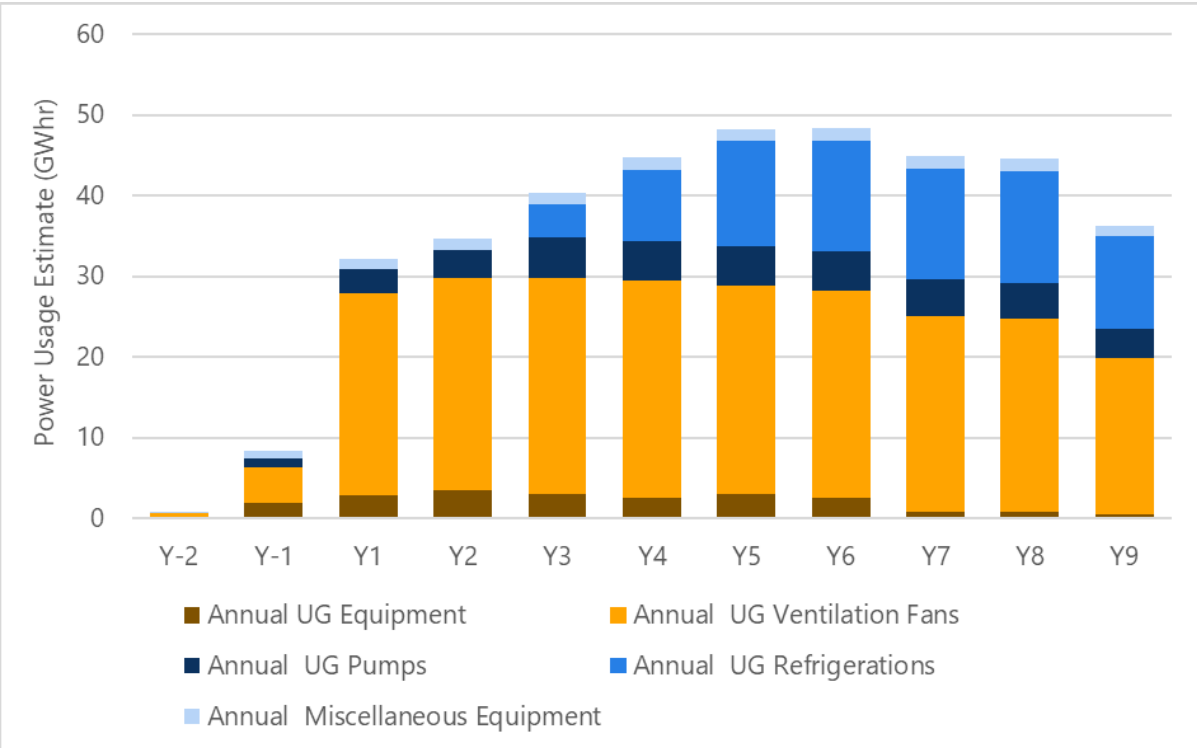

Figure 16.80: Annual Power Usage Bankan Underground in GWh

16.8.11 Air and Water Supply

An Atlas Copco GA200 compressor will supply air to the underground workings for equipment such as development drills, production drills, the spraymec, and to enable charge-up operations to be carried out. PN16 poly pipelines will be installed underground as part of the mine services to deliver air and water to the decline, production levels and other working areas as required.

16.8.12 Communications

A leaky feeder system is planned to be installed throughout the entire underground mine, providing continuous and reliable radio communication coverage.

16.8.13 Escapeways

A robust escapeway system has been incorporated into the underground mine design, in line with Western Australian mining regulations and uphold industry best practices for underground safety. In line with these standards, each operational level is equipped with two independent means of egress, ensuring safe evacuation routes for personnel during emergencies

Each level connects to a dedicated escapeway system that provides access either to the main decline or another level, ultimately leading to surface exits. These escapeways are strategically linked via vertical raises and horizontal connections, enabling personnel to ascend to a safe refuge level if required. From there, personnel can proceed to the main decline or the dedicated ventilation decline to exit the mine.

Escape ladderways will be installed in the escapeway raises, depicted in green in Figure 16.81. Each ladderway raise is designed at an incline of approximately 70 degrees or less, with a typical escapeway length of around 25 meters.

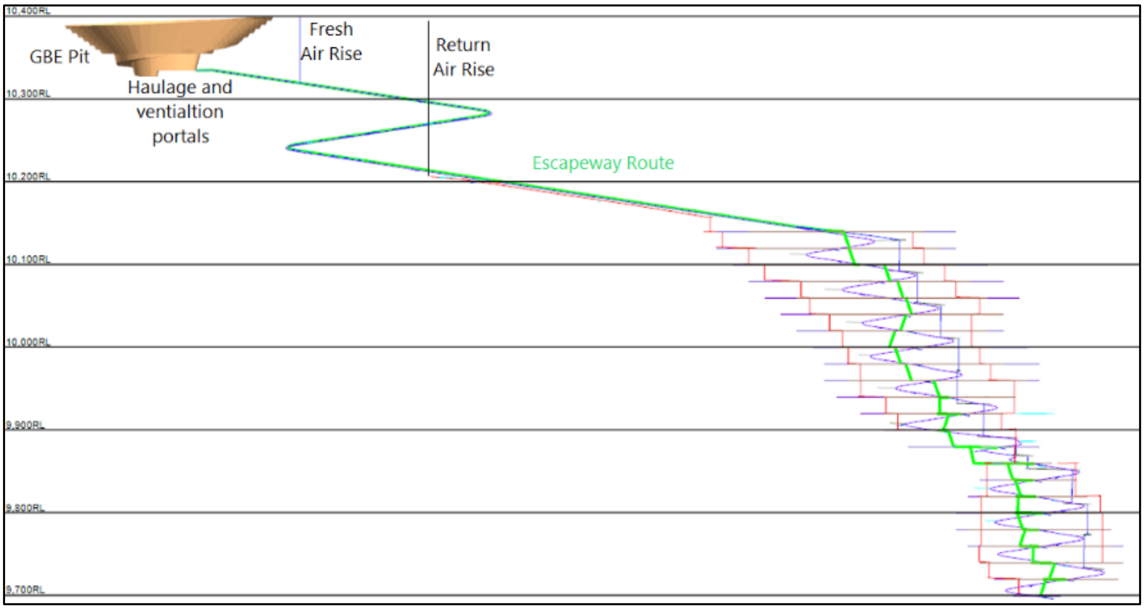

Figure 16.81: Escapeway Route (Shown in Green)

Underground personnel should not use the emergency egress during an underground fire or hazardous atmosphere, as it may lead to dangerous conditions. The emergency egress system is designed to help personnel exit the mine if main access routes are blocked due to ground support failure or equipment issues. In the event of a hazardous atmosphere, personnel must immediately use self-contained self-rescuers and proceed to a purpose-built refuge chamber.

16.8.14 Refuge Chambers

Self-contained refuge chambers offer a safe haven if the underground atmosphere becomes hazardous. When properly specified, these chambers can provide breathable air for at least 36 hours without relying on external mine services. Their effectiveness can be extended if they receive a continuous supply of fresh air, such as compressed air from the surface, or if they operate below full capacity.

Fixed refuge chambers are planned at intervals not exceeding 750 metres walking distance, in alignment with industry best practices. Each underground level will include a designated refuge chamber drive, positioned along the level access.

To ensure uninterrupted operations and safety, especially during simultaneous activities such as truck loading and development, a 4-person self-contained refuge chamber will be positioned beyond the stockpile area on each level. Additionally, each LHD unit will have an associated mobile refuge chamber available, providing redundancy and enhancing emergency preparedness across the mine.

17 RECOVERY METHODS

17.1 Introduction

The Bankan gold processing plant has been designed by DRA Global Limited (DRA) with a nameplate ore treatment rate of 4.5 Mt/a.

This section summarises key technical data for the recovery of gold from the Bankan Gold Project ore types within the Bankan processing plant facility. Section 17.3 summarises the development of design criteria for the Bankan processing plant, based on findings from metallurgical testwork, circuit modelling and design engineering. Section 17.4 presents a simplified version of the Bankan processing plant flowsheet and highlights the major equipment specifications. Section 17.5 describes the gold recovery process and the process flowsheet in greater detail by plant area. Sections 17.7 and 17.8 summarise power and water requirements for processing Bankan Gold Project ore deposits.

17.2 Site Location and Layout

The plant layout was developed based on an appropriate level of engineering effort required to support a Feasibility Study level capital cost estimate. The plant has been located to suit the available usable area within the topography, whilst providing appropriate access for all supporting infrastructure including mining, power supply and tailings storage. Due consideration for interaction of these activities is critical for the effective operation of the processing plant facility and is captured in the overall site plan and layout.

The design approach adopted reflects typical industry good practice, which will be continued and further developed during the detailed engineering of the Project.

As shown in Figure 17.1, project facilities, including the mining infrastructure area and administration have been located within close proximity of the process plant site to maintain a single centre of operations. A double security fence surrounds the entire process plant, excluding some NPI buildings, reagent sheds, raw water pond, tailings stockpiles, power station and the mining infrastructure area. Security guards at the gate house will allow workers and guests into the process plant as required. A single fence, with associated gate house, surrounds all of the central infrastructure.

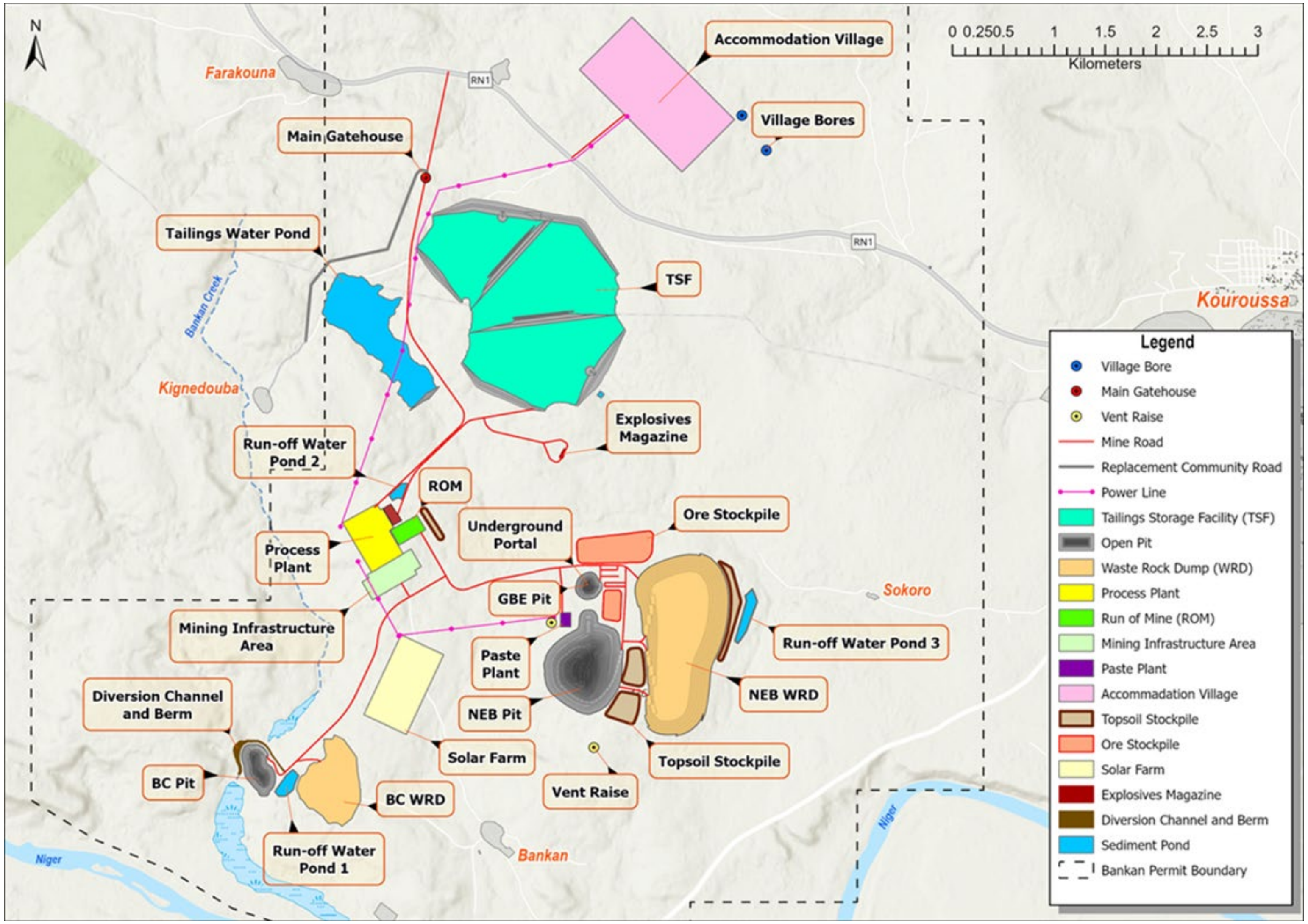

Figure 17.1: Overall Site Layout

17.3 Design Criteria Development

The process plant design considers the treatment of a range of lithologies, at a throughput of 4.5 Mtpa, including:

- Saprolite.
- Laterite and saprock.
- Fresh ore.

Of these lithologies, saprolite, laterite and saprock are considered as weathered ore.

Fresh (equivalently 'hard') ore and any competent weathered ore (laterite and saprock) will be crushed using a primary jaw crusher and conveyed to the crushed ore stockpile.

Sticky saprolitic ore was considered to provide a significant risk to materials handling if it was allowed to combine with fresh ore and report to the crushed ore stockpile. This risk was eliminated by designing receipt and crushing of saprolitic ore separately from fresh ore. This design keeps sticky ore live and in motion as far as possible, with crushed oxide ore reporting directly to SAG mill feed (it is not allowed to report to the stockpile).

A design blend of 75% fresh ore and 25% weathered ore was adopted for the comminution circuit design (Orway Mineral Consultants, 2025). OMC considered this blend and the 85th percentile ore physical properties for the design and sizing of the grinding mills and other comminution equipment.

Testwork indicates significant quantities of gravity recoverable gold justifying the inclusion of a gravity circuit within the comminution circuit for the following reasons:

- Intensive cyanidation of the gravity concentrate has improved potential of leaching gold from semi-refractory minerals that may be present.
- Removing gravity recoverable gold in a dedicated circuit reduces the amount of coarse gold that may accumulate in gravity traps across the comminution circuit, improving gold accountability and security outcomes.

Gravity/cyanidation variability testwork on Bankan Gold Project ore samples determined that gold extractions ranged between 81.2% and 97.6% and that the level of extraction is influenced by the gold head grade. With gravity included to remove coarse gold, gold leaching kinetics were rapid with most gold leaching occurring within four to eight hours. No evidence of preg-robbing was identified, and the plant design comprises a single leach tank without carbon, followed by six-stages of carbon in leach (CIL) with a nameplate residence time of 24 hours.

The variability leaching indicated a small proportion of samples presenting nuisance levels of soluble copper that will build up on carbon until it reaches a problematic level. The elution has been specified to include an initial cold cyanide wash step that effectively removes copper from loaded carbon.

Blends with increased proportions of saprolite tested in rheology, thickening and filtration testwork exhibited performance limitations at greater than 60% w/w solids if the saprolite proportion was greater than 50% w/w. A pre-leach thickener has been included in the flowsheet with CIL design pulp density of 50% w/w solids selected and supported by testwork.

Cyanide detoxification of the tails stream prior to filtration is included to ensure that moisture in the filtered tailings discharge achieves CN_{WAD} levels less than 50mg/L, which aligns with industry guidelines for TSFs such as Standard of Practice 4.4 of International Cyanide Management Institute's Mining Operations Verification Protocol (International Cyanide Management Institute, 2021).

Tailings filtration has been included as filtered stacked tailings is the approved basis for tailings deposition for the project.

For volumetric design in the plant (which includes equipment such as screens, pumps, thickeners, tanks, etc) two design cases were considered and used for equipment sizing:

- 100% fresh ore.
- A blend of 50% weathered and 50% fresh ore.

The worst-case value (typically highest) was used as the design value.

Table 17.1 lists the key process design criteria and equipment sizing.

Table 17.1: Key Process Design Criteria and Equipment Sizing

Parameter	Units	Value
Operating Schedule		
Annual Plant Throughput (dry solids)	Mtpa	4.5
Primary Crushing Throughput (dry solids)	t/h	685
Fresh Ore (dry solids)	t/h	685
Weathered Ore (dry solids)	t/h	342
Grinding and Plant Throughput (dry solids)	t/h	563
Nominal Gold Feed Grade	g/t	1.86
Design Gold Feed Grade	g/t	2.50
Design Gold Recovery	%	93
Average Production	koz/a	249
Maximum Production (equivalent rate)	koz/a	375

Parameter	Units	Value
Physical Ore Characteristics		
Fresh Ore		
SMC (Axb)		23.2
Bond Rod Mill Work Index	kWh/t	23.5
Bond Ball Mill Work Index	kWh/t	23.6
Bond Abrasion Index	g	0.367
Specific grinding power (to P_{80} 75µm)	kWh/t	29.2
Laterite and Saprock Ore		
SMC (Axb)		144
Bond Rod Mill Work Index	kWh/t	13.0
Bond Ball Mill Work Index	kWh/t	10.8
Bond Abrasion Index	g	0.010
Specific grinding power (to P_{80} 75µm)	kWh/t	12.8
Saprolite Ore		
SMC (Axb)		150
Bond Rod Mill Work Index	kWh/t	3.0
Bond Ball Mill Work Index	kWh/t	3.0
Bond Abrasion Index	g	0.010
Specific grinding power (to P_{80} 75µm)	kWh/t	5.5
Crushing		
Fresh Ore Crushing Circuit		
Primary crusher		Single toggle jaw crusher, Metso C140 or equivalent
Feed size, F_{100}	mm	900
Product size, P_{100}	mm	391
Crushed ore stockpile, live capacity	t	9,450
Weathered Ore Crushing		
Primary Crusher		Double roll mineral sizer
Feed size, F_{100}	mm	900
Product size, P_{100}	mm	361

Parameter	Units	Value
Grinding Circuit		
Circuit type		SABC
Feed size, F_{100}	mm	391
Product size, P_{80}	µm	75
SAG mill		9.75 m diameter x 5.0 m EGL, grate discharge, 10 MW installed, high speed VSD
Ball mill		7.32 m diameter x 10.0 m EGL, overflow discharge, 10 MW installed, high speed VSD
Pebble crusher		Cone crusher, Metso HP200 or equivalent
Gravity Circuit		
Gravity gold recovery	%	32
Concentrator capacity	t/h	420
Intensive leach capacity	kg/day	2,256
Pre-leach, Leach and Adsorption		
Pre-leach thickener settling rate	t/h/m²	0.6
Pre-leach thickener		High rate, 35 m diameter
Leach & CIL slurry residence time	h	24
Leach tanks		1
CIL tanks		6
Nominal volume	m³	3,000
Carbon loading	g/t	2,500
Elution, Electrowinning and Carbon Regeneration		
Elution process		Pressure Zadra
Acid wash and elution column capacity	t	10
Elution frequency	per week	6
Carbon regeneration kiln		Diesel fired, 450 kg/h
Cyanide Detoxification		
Process		INCO SO2 Cyanide Destruction
Detoxification discharge WAD cyanide	mg/L	20
Detoxification slurry residence time	h	1.5
Number of tanks		2
Tank volume, nominal	m³	850

Parameter	Units	Value
Tailings Filtration		
Type		Plate and Frame Pressure Filters
Tailings filtration throughput (dry solids)	t/h	734
Tailings filtration filter cycle time	minutes	10
Paste (backfill) plant production rate (maximum)	m^3/a	498,800
Number of filters/plates per filter		3 / 74
Filter size		2.5 m x 3.5 m

17.4 Process Flowsheet

The selected process flowsheet will include primary crushing of fresh (hard) ore and soft (weathered) ore in separate circuits using a primary jaw crusher and mineral sizer respectively. Crushed fresh ore will report to a crushed ore stockpile, while crushed weathered ore will report directly to SAG mill feed.

The grinding circuit will consist of a SAG mill with a pebble crusher and a ball mill operating in closed circuit with the grinding circuit cyclones to a grind product size of 80% passing (P_{80}) 75 µm. Gravity concentration for removal of coarse gold from the circulating load within the milling circuit, and treatment of gravity concentrate by intensive cyanidation and electrowinning.

A leach feed thickener will be included in the flowsheet to provide a consistent thickened feed to the hybrid leach-CIL train. Loaded carbon will be eluted using the Zadra method, with simultaneous electrowinning of gold and silver from the eluate solution. Precious metal sludge will be recovered from the electrowinning cells and cathodes, then filtered and dried. Dry precious metal sludge will be mixed with fluxes and smelted to produce doré bullion bars.

CIL tailings will be detoxified for cyanide destruction using sodium meta-bisulphite and air prior to being filtered using pressure filtration to produce filter cake for disposal in the TSF. A portion of the tailings will be deslimed in cyclones for separate filtration to produce feed for the paste backfill plant.

A schematic of the process flowsheet is provided in Figure 17.2.

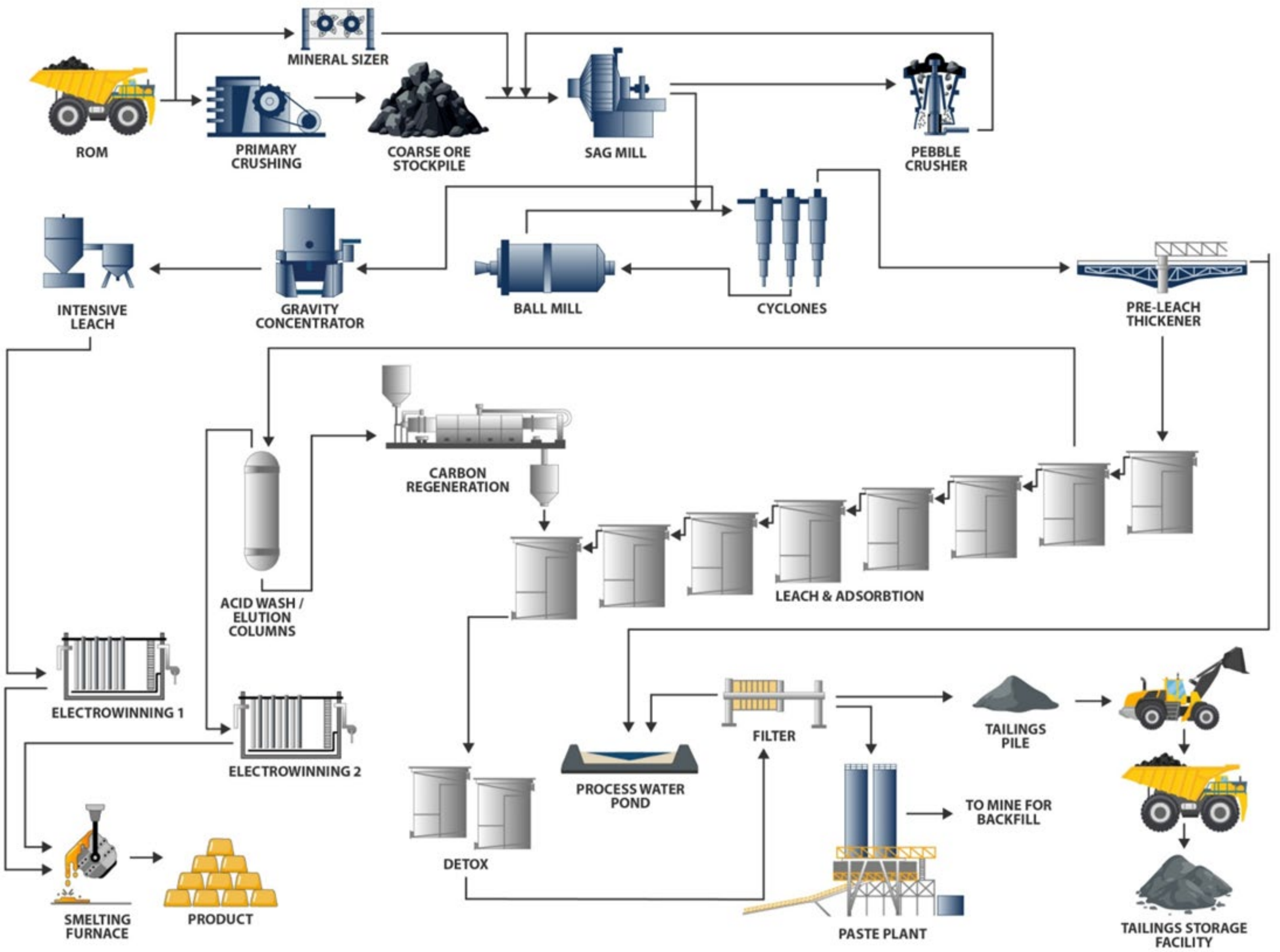

Figure 17.2: Process Flowsheet

17.5 Process Plant Description

17.5.1 ROM Pad

The ROM pad is used to provide a buffer between the mine and the plant. ROM stockpiles allow blending of ore types and grades to ensure a consistent feed type and feed rate to the plant.

Mining haul trucks operating from the open pit mine will deliver ROM ore to the ROM pad, where it will be direct tipped to the ROM bin or dumped to blending stockpiles. In addition to the blending capability, the ROM pad will have designated areas to keep fresh and semi-competent weathered ore separate from saprolitic “sticky” ore.

A front-end loader (FEL) will be used to reclaim ore from the various ROM pad stockpiles to the ROM bin.

The ROM bin will be equipped with raw water sprays to provide dust suppression whilst ore is fed to the ROM bin.

17.5.2 Crushing Circuit

17.5.2.1 Fresh Ore

ROM fresh (or equivalently ‘hard’) ore will be tipped into the ROM 1 bin by dump truck or by loader. A static grizzly on the ROM 1 bin will exclude very large rocks, which will be removed using a loader or an excavator. Ore will be withdrawn from the ROM 1 bin by ROM 1 primary feeder and fed to the ROM 1 primary vibrating grizzly. The vibrating grizzly will separate fines from coarse and will feed the coarse fraction to the ROM 1 primary crusher. Primary feeder dribble (adhering fines), vibrating grizzly feeder fines (undersize), and Primary crusher product will all report to the Stockpile feed conveyor.

ROM 1 primary crushing solids rate will be measured by the ROM 1 primary crusher discharge conveyor weightometer. A self-cleaning tramp metal magnet located above the conveyor will remove any magnetics which discharge to a bunker. A metal detector located above the Stockpile feed conveyor (after the tramp magnet) will detect any large tramp metal item that is not recovered by the magnet.

Raw water will be provided from the Raw water distribution main for dust suppression at the ROM 1 bin tip and in the ROM 1 crushing area.

The ROM 1 area will be serviced by a crane for maintenance activities such as crusher liner changes.

17.5.2.2 Sticky Weathered Ore

ROM sticky weathered (or equivalently ‘soft’) ore will be tipped into the ROM 2 bin by dump truck or by loader. Ore will be withdrawn from the ROM 2 bin by ROM 2 primary feeder and fed directly to the ROM 2 primary sizer. ROM 2 primary feeder dribble (adhering fines) and ROM 2 primary sizer product will report to the ROM 2 discharge conveyor.

ROM 2 primary crushing solids rate will be measured by the ROM 2 discharge conveyor weightometer. A self-cleaning tramp metal magnet located above the conveyor will remove any magnetics, which will discharge to a bunker. A metal detector located above the Sizer product conveyor (after the tramp magnet) will detect any large tramp metal item that is not recovered by the magnet. The Sizer product conveyor will discharge directly to the SAG mill feed conveyor.

Raw water will be provided from the Raw water distribution main for dust suppression at the ROM 2 bin tip and in the ROM 2 Crushing area.

17.5.3 Coarse Ore Stockpile and Reclaim

The Stockpile feed conveyor will discharge to the Crushed ore stockpile. Dead stockpile capacity can be reclaimed using a bulldozer and/or excavator to push material to the stockpile reclaim slots.

A reclaim tunnel beneath the stockpile will enable withdrawal of stockpiled ore by the reclaim feeders (apron feeder type). Each feeder will have a capacity of 100% of SAG mill new feed, but in normal operation both feeders will operate at an equal rate. This will allow even drawdown and filling of the stockpile. The reclaim feeders will discharge onto the SAG mill feed conveyor, which will convey crushed ore to the SAG mill feed chute. The SAG mill feed conveyor will be fitted with a weightometer to measure the new (reclaimed) solids feed rate required for process control and metallurgical accounting.

A manual hoist will be provided at the head and tail end of each apron feeder, to facilitate the removal and installation of new feeder pans, maintenance of motors, etc. An additional powered monorail runs alongside the feeders for the entire length of the upper chamber and includes a drop zone to the lower section of the tunnel.

The Reclaim tunnel ventilation fan will force clean air into the reclaim tunnel thus reducing the dust concentration and preventing stagnant air in the tunnel. The stockpile tunnel will be sloped to drain water outside and into a channel.

Quicklime in one tonne bulk bags will be lifted to the Quicklime bag breaker which will discharge into the Quicklime receiving hopper. A blower will be used to transport lime from the Quicklime receiving hopper into the Quicklime silo. A dust collector will be provided on the silo to prevent dry lime dust emission during offloading and pneumatic transfer. Quicklime will be withdrawn from the silo by a rotary valve together with an activator in the silo. The rotary valve discharges quicklime directly onto the SAG mill feed conveyor.

17.5.4 Grinding and Classification Circuit

The SAG mill feed conveyor will transport reclaimed crushed ore, crushed oxide ore, pebbles, and quicklime to the SAG mill feed chute. Process water will be added to the feed chute to adjust the mill discharge solids concentration. The SAG mill will operate in closed circuit with the SAG mill discharge screen, with pebbles (screen oversize) returning via the SAG mill feed conveyor after tramp metal removal and pebble crushing. The design and layout will allow for cyclone underflow to be returned to SAG mill feed if so required. SAG mill grinding media (steel balls) will be loaded using a kibble and hoisted to the SAG mill ball charging chute.

SAG mill discharge screen undersize, ball mill discharge screen undersize, and gravity concentrator tailings will report to the cyclone feed hopper, together with (intermittent) area spillage. Process water will be added to the cyclone feed hopper and the slurry will be pumped by the cyclone feed pump (duty and standby installed) to the grinding cyclone cluster.

The ball mill will operate in closed circuit with the grinding circuit cyclones, with cyclone underflow returning by gravity to Ball mill feed. ball mill discharge screen oversize (scats) will discharge to a bunker. Ball mill grinding media (steel balls) will be loaded using a kibble and hoisted to the ball mill ball charging chute.

Two spillage pumps in the grinding area will operate intermittently, discharging spillage to the cyclone feed hopper. The grinding area spillage bund will be accessible to a skid steer loader (drive in access) via a ramp.

Two HCN detectors will be installed, at the cyclone cluster and in the grinding area bund.

17.5.5 Pebble Crushing

Pebbles from the SAG mill discharge screen (oversize) will report to the pebble transfer conveyor which will feed the pebble crusher feed conveyor. A weightometer on the pebble crusher feed conveyor will measure the pebble solids feed rate. A self-cleaning magnet on the Pebble crusher feed conveyor will remove any tramp steel, which will discharge to a bin. A metal detector, located after the tramp magnet, will detect any large tramp metal item that is not removed by the magnet. Pebbles will normally report to the pebble crusher feed bin, with the option to discard pebbles from the circuit using a diverter chute. A vibrating pan feeder will (intermittently) feed pebbles to the pebble crusher, which will discharge onto the SAG mill feed conveyor.

17.5.6 Gravity Circuit

A dedicated gravity feed pump will take suction from the ball mill discharge side of the cyclone feed hopper. This pump will discharge slurry to the gravity scalping screen feed box. Process water will be added for dilution and as spray water on the gravity scalping screen and will also be used in the gravity concentrator for fluidisation. Gravity scalping screen oversize will report to the ball mill feed chute. Gravity scalping screen undersize will feed the gravity concentrator, with concentrator tailings reporting to the cyclone feed hopper. The concentrator will operate on a timed cycle. At the end of the cycle, concentrator feed will be isolated and gravity screen underflow will report to the cyclone feed hopper. The accumulated gravity concentrate will be flushed out of the concentrator bowl and into the intense leach reactor (ILR) feed hopper. After concentrate flushing is complete, slurry feed to the concentrator will resume.

Intensive cyanidation will be accomplished using a vendor package that will be located within a secure and restricted access portion of the grinding area. Gravity concentrate will accumulate batch by batch in the ILR feed hopper. Once per day the concentrate will be transferred into the ILR drum and intensive leaching will start, with cyanide, caustic soda, and peroxide added to the drum feed. The drum will rotate while solution is recirculated from the ILR pump hopper to the drum feed, for a total batch duration of up to ~21 hours. Flocculant solution will be added if required. The intensive leach solution will be pumped to the Gravity loaded eluate tank. The leach residue solids will then be rinsed, and the rinse solution will be pumped to the gravity loaded eluate tank. The leach residue will then be pumped to the cyclone feed hopper, leaving the drum ready for the next batch of concentrate.

Intensive cyanidation area spillage will be pumped to the cyclone feed hopper.

One hydrogen cyanide gas detector will be installed in the intense cyanidation area bund.

17.5.7 Pre-Leach Thickening

Grinding cyclone overflow will flow by gravity to the trash screen feed box. The trash screen will remove any coarse material such as vegetation, grit, plastic, and blasting cord. Trash will discharge from the screen into a skip, for periodic removal by a forklift. Screened slurry will flow to a multi-stage metallurgical sampler and then to the Pre-leach thickener feed box. The final cut of the metallurgical sampler will report to a bucket, providing a composite sample per shift.

The following streams will report to the Pre-leach thickener feed box:

- Trash screen underflow – normal, continuous flow.
- Coagulant solution - continuous flow, when coagulant is in use.
- Pre-leach thickener area spillage – intermittent flow.

Dilute flocculant solution will be added to the pre-leach thickener feed pipe to improve the solids settling rate and thickener overflow clarity. Coagulant will be added to the pre-leach thickener feed box (upstream of flocculant addition) when the ore is slow settling or difficult to settle and the thickener overflow becomes unacceptably turbid. Pre-leach thickener underflow will be pumped to the leach feed box.

Pre-leach thickener overflow will report to the process water tank. Raw water, clarified water, and TSF decant water will be used to top up the process water tank. The process water pumps will distribute process water to the following points of use:

- Cyclone feed hopper.
- SAG mill feed.
- SAG mill discharge screen.
- Ball mill feed.
- Ball mill discharge screen.
- Gravity scalping screen feed box.
- Gravity scalping screen.
- Gravity concentrator.
- Trash screen.
- Pre-leach thickener flocculant dilution.
- General plant distribution for washdown.

One hydrogen cyanide gas detector will be installed in the pre-leach area bund, with another installed at the process water pumps.

17.5.8 Leach and Adsorption Circuit

The leach and CIL train will consist of one leach tank followed by six CIL tanks, all operating in series. Each tank will be equipped with a two-stage slurry agitator. Each CIL tank will also be equipped with an interstage screen (for carbon retention) and a recessed impeller pump (for carbon advance). The leach tank will be designed and equipped to operate as a CIL tank but will be installed without an interstage screen or a carbon advance pump. Slurry will flow down the tank train from the leach tank to CIL tank 1 and then through to CIL tank 6. Any single tank can be bypassed when required for maintenance, using the tank top launders and gate valves.

The following streams will report to the Leach feed box:

- Pre-Leach thickener underflow slurry (continuous flow).
- Feed slurry dilution - process water (as required).
- Cyanide solution (continuous flow).
- Caustic soda solution (intermittent addition for pH trim).
- CIL spent solution (intermittent).
- Gravity IC spent solution (intermittent).
- ILR wash solution (intermittent).
- Leach area spillage (intermittent).
- Intensive cyanidation area spillage (intermittent).
- Elution area spillage (intermittent).
- Gold room spillage (intermittent).

Pre-leach thickener underflow will be pumped to the leach feed box, where process water will be added for dilution as required. The leach feed box will allow feed to CIL tank 1 when the Leach tank is off-line. Sodium cyanide solution will be added to the leach feed box. A cyanide analyser at the leach tank will provide on-line free cyanide measurement and will be used for cyanide dosing control. Cyanide solution will be able to be added to CIL tanks 1,2, and 3 if required.

Caustic soda solution will be added to the leach feed box if required for pH modification. Caustic soda solution can be added to CIL tanks 1 and 2 if required.

Air will be added to the leach tank and the first three CIL tanks, using lance sparges located at the tank base.

The leach and CIL area tower crane will be used to remove intertank screens for scheduled cleaning. An intertank screen trolley and screen maintenance bay will be provided, with a high pressure water washer for cleaning screen surfaces. One spare intertank screen will be kept ready for replacement when an operating screen is removed for cleaning.

Two leach area sump pumps will operate intermittently to remove spillage. Leach area sump pump 1 will discharge to the Leach feed box. Leach area sump pump 2 will discharge to CIL tank 4.

Four hydrogen cyanide gas detectors will be installed in the leach and adsorption area, two at the tank top level and two at the bund level.

17.5.8.1 Adsorption Carbon Management

Activated carbon in each CIL tank will be retained by an immersed screen. The screens will be cylindrical, mechanically swept, with stainless wedge wire screening surface. Carbon will advance counter-current to slurry flow by regular intermittent upstream slurry pumping known as "carbon advance" or "interstage carbon transfer". Vertical recessed impeller pumps, one per CIL tank, will be used for this purpose. The carbon advance pumps will typically start simultaneously and run for a controlled duration. One transfer will be scheduled each day to achieve the required carbon advance rate.

Carbon in the first CIL tank will be referred to as 'loaded carbon' since it will have the highest gold grade. One loaded carbon transfer will be scheduled each day to match the carbon advance rate. Loaded carbon recovery is described in Section 17.5.9.1.

17.5.9 Desorption

The desorption area includes loaded carbon recovery, acid washing, elution and carbon reactivation.

17.5.9.1 Loaded Carbon Recovery

Slurry from the first CIL tank will be pumped to the Loaded carbon screen by the Loaded carbon recovery pump. Raw water will be used for screen spray. Loaded carbon screen underflow slurry will return by gravity flow to the first CIL tank, while the washed loaded carbon (screen oversize) will report to the acid wash column.

17.5.9.2 Acid Wash

The purpose of acid washing is to remove inorganic foulants such as calcium scale before the carbon is eluted.

Once a full loaded carbon batch is ready, hydrochloric acid and water will be pumped into the acid wash column feed manifold to achieve 3% HCl concentration in solution. Acid solution will be circulated through the column and back to the acid wash pump suction for a pre-set time. The carbon in the acid wash column will then be allowed to soak for a pre-set time. The carbon will then be rinsed with a pre-set volume of fresh water to remove the residual acid. Acid wash effluent (rinse) will report to the tailings hopper. After rinsing, carbon will be hydraulically transferred from the acid wash column to the elution column.

One detector will be installed in this area to detect hydrogen cyanide gas.

17.5.9.3 Elution

The elution area will be designed to treat one batch of carbon per day and a total of xix batches of carbon per week, though back-to-back stripping can treat seven batches of carbon per week when required.

Elution will be a batch process using the Zadra method, including the following main steps:

- Cold cyanide wash (as required).
- Elution solution makeup – caustic and cyanide addition.
- Pre-heat.
- Elution.
- Cooling.
- Carbon transfer to reactivation.

The cold cyanide wash will only be performed when copper loading on carbon is shown (by assay) to exceed a pre-set threshold. All other steps will be routinely performed.

The first step of the cold cyanide wash will be to fill the elution water tank with treated water and to add a measured volume of sodium cyanide solution to achieve a target NaCN concentration. The cyanide solution will then be pumped into the elution column feed manifold and circulated back to the feed pump for a set time. After this timer elapses, solution leaving the elution column will report to the copper precipitation tank, until the elution water tank is empty. Treatment of the cyanide wash solution is described below.

The next desorption step will be to refill the elution water tank with treated water and add measured volumes of sodium cyanide and sodium hydroxide (caustic soda) solutions to the tank, in order to achieve target NaCN and NaOH concentrations. The next step is pre-heat, in which the caustic cyanide solution will be pumped through the column and the heat exchangers, then back to the elution water tank. The solution heater (heated by a diesel burner) will operate at full output until the column outlet temperature reaches 95ºC. The elution pump will then stop, and the carbon bed will be allowed to soak in the hot caustic cyanide solution for a pre-set time. Following this pre-soak the elution water pump will re-start and the column outlet temperature will be increased to 130ºC. In the Zadra process, electrowinning proceeds simultaneously with elution. Solution leaving the recovery heat exchanger will report to the flash vessel, then to the electrowinning cells, then will return to the elution water tank. Solution pumping and recirculation will continue until the gold grade in solution leaving the electrowinning cells drops below the operating target.

Two detectors will be installed in this area to detect hydrogen cyanide gas.

17.5.9.4 Cyanide Wash Solution Treatment

When required by high copper loading levels on the carbon, cyanide wash solution will be treated batchwise in the copper precipitation tank. The tank agitator will be started, and sulphuric acid will be added until pH 2 is reached. The agitator will stop allowing precipitated copper cyanide sludge to settle into the tank's bottom cone. The agitator will re-start and caustic soda will be added to achieve a pH greater than 8. The agitator will then stop, and neutralised solution will be pumped to the process water tank. Precipitated copper sludge will be pumped from the tank underflow (as required) to a lined pond for disposal.

17.5.9.5 Carbon Reactivation

Eluted carbon will be hydraulically transferred from the elution column to the carbon dewatering screen feed box. Dewatered carbon (screen oversize) will report to the carbon reactivation kiln feed hopper, while screen undersize will flow to the carbon quench hopper. Excess/residual water drains from strainers in the carbon reactivation kiln feed hopper.

Carbon will be withdrawn from the feed hopper by the carbon reactivation kiln screw feeder and fed to the diesel burner heated carbon reactivation kiln. The kiln will operate at 750°C in a steam-rich atmosphere to break down organic foulants on the carbon and to reactivate the carbon after each elution cycle. Fumes will be removed by the carbon reactivation kiln fan and discharged to atmosphere via the carbon reactivation kiln exhaust stack.

Reactivated carbon will discharge from the kiln into the carbon quench hopper which will be kept full of water.

The carbon will be transferred (batchwise) by the carbon transfer pump to the carbon sizing screen on top of the CIL tanks. Raw water will be used for screen spray. Carbon sizing screen oversize (sized carbon) will return to the adsorption train at CIL tank 6, or to CIL tank 5 when tank 6 is off-line. Carbon sizing screen undersize will report to the safety screen feed box.

A bag of fresh carbon will be periodically dumped into the carbon quench tank as required to make up carbon losses.

The carbon reactivation area sump pump will discharge area spillage (intermittently) to the carbon sizing screen feed box.

17.5.10 Electrowinning and Gold Room

The electrowinning area will be a secure area with controlled access, adjacent to or inside the gold room. The gold room will be a secure structure with controlled access. Both the electrowinning area and the gold room will have multiple closed-circuit monitors installed for security monitoring.

Eluate solution from the elution column will flow first to a flash vessel, then to the electrowinning cells (two operating in parallel). Solution discharging from the electrowinning cells will return to the elution water tank.

A single gravity electrowinning cell will operate for the recovery of gold and silver from the gravity loaded solution. Solution discharging from the gravity electrowinning cell will return to the gravity electrowinning feed tank.

Dissolved gold and silver will be recovered by plating onto the stainless-steel wool of the cathodes. When the gold grade in solution drops below the operating target, electrowinning will be considered complete. The elution process will conclude and the spent elute solution will be pumped to the leach feed box.

The electrowinning fume extraction fan will operate continuously drawing off the electrowinning cells and will vent any noxious off-gas to atmosphere. Electrowinning area spillage will be discharged to the Leach feed box.

One cell at a time will be taken off-line for sludge harvesting or maintenance. The cathodes will be removed using a lifting frame and an electric hoist and taken to the cathode wash station. A high pressure, low flow pump will be used to wash metal sludge from each cathode. Recovered sludge will flow by gravity to the gold sludge collection hopper. Any sludge in the electrowinning cell will be washed out and drained to the gold sludge collection hopper.

Sludge will be fed in batches to the sludge filter, producing a dewatered precious metal cake. The filter cake will then be spread on trays and dried in one of the electrically heated drying ovens. Dry sludge will be mixed with fluxes in the flux mixer, pre-weighed, and charged to the smelting furnace for smelting to produce doré bullion. Slag will be cleaned off the bullion bars by hand before sampling, weighing, stamping for identification, and storage in the safe and strong room prior to export.

A sample will be taken from each gold bar by drilling with an electric drill and using a small diameter drill bit. The fragments produced by drilling are called prills. The sample balance will be a laboratory type electronic scale used for weighing the total mass of prills (sample) for each bar.

The gold room area spillage pump will discharge to the leach feed box.

17.5.11 Tailings Detoxification

Tailings slurry from the last CIL tank will flow by gravity to the detoxification feed box, together with the following other streams:

- TSF decant (continuous flow, when available).
- Raw water (if no decant is available, continuous flow).
- Sodium meta-bisulphite (SMBS) solution (continuous flow).
- Copper sulphate solution (continuous flow).
- Caustic soda solution, as required.
- Acidic reagent area spillage (intermittent).

SMBS is required as a source of SO_2 for the detoxification reaction, while copper sulphate is required to catalyse the reaction. Caustic soda will be used to neutralise acid formed by the detoxification reaction and maintain protective alkalinity.

There will be two cyanide detoxification tanks, fitted with two stage agitators and operating in series. An on-line analyser will take liquor samples from the detoxification feed box and the first detoxification tank. The analyser will measure free cyanide, weak acid dissociable (WAD) cyanide, and pH required for reagent addition control and discharge cyanide monitoring. Compressed air will be added using horizontal lance sparges located at the base of the tanks.

Detoxification discharge slurry will report to a multi-stage tailings slurry sampler. Sampler discard slurry will report to the carbon safety screen feed box. Raw water will be used for carbon safety screen spray. oversize (carbon) recovered by the safety screen will report to a bulk bag (at ground level) for subsequent treatment or disposal. Carbon safety screen underflow slurry will gravitate to the tailings hopper.

Two hydrogen cyanide gas detectors are installed in this area, one at detoxification tank top level and the second at bund level.

17.5.12 Tailings Filtration

The tailings pumps (duty and standby) will discharge (post-detoxification) tailings slurry to the tailings filtration feed box, together with filtration clarifier underflow slurry. The tailings filtration feed box will split feed slurry to all three filter feed surge tanks. Filter feed pumps 2 and 3 will draw slurry from surge tanks 2 and 3 (operating with a common suction) and will feed tailings filter 2 and tailings filter 3 respectively. Filter feed pump 1 will draw slurry from surge tank 1 and will feed tailings filter 1.

Dirty water streams from the filters will include filtrate, core wash, and cloth wash – these will report to the filtration clarifier together with spillage discharged by the filtration area spillage pump. Clarifier overflow will report to the clarified water tank for re-use in the process.

Filter cake will be discharged from each filter onto its discharge conveyor. All three discharge conveyors will operate in the same direction and will discharge onto the tailings transfer conveyor. This conveyor will discharge to the Tailings stacker conveyor, which will slew to form the tailings stockpile. Tailings filter cake will be reclaimed from the stockpile, loaded, and hauled to the tailings storage facility for dry stacking.

17.5.13 Paste Feed Production

When paste feed production is active, the tailings filtration feed box will split slurry to filter feed surge tanks 2 and 3 only. The deslime feed pump (duty and standby) will draw slurry from surge tanks 2 and 3 (operating with a common suction) and will feed the deslime cyclone cluster. Deslime cyclone underflow will report to tailings surge tank 1, while deslime cyclone overflow will report to the filtration clarifier. Filter feed pumps and tailings filters will operate as described in Section 17.5.12.

Tailings filter 1 discharge conveyor will operate in the opposite direction to the other two filter discharge conveyors, discharging to the paste feed transfer conveyor. This conveyor will discharge to the paste feed stacker conveyor, which will slew to form the paste feed stockpile. Paste feed filter cake will be reclaimed from the stockpile, loaded, and hauled to the paste plant stockpile.

17.5.14 Paste Plant

Binder will be delivered by road tanker and pneumatically offloaded to the binder silo. The silo will include an activator, a rotary valve, a weigh feeder, and a screw feeder to provide a controlled rate of binder to the paste mixer.

Clean raw water will be pumped from the process plant to the paste raw water tank. The paste raw water pumps (duty and standby) will provide water in the paste plant.

Filter cake will be reclaimed from the local stockpile and tipped into the paste feed hopper. Filter cake will be fed by the paste feeder and lifted by the paste mixer feed conveyor to the paste mixer. Binder (assumed to be cement) and clean raw water will be added to the paste mixer. Mixed paste will discharge over a screen where any coarse lumps will be removed. Fine mixed paste (screen underflow) will report to the paste product hopper. High pressure positive displacement paste pumps (duty and standby) will discharge paste into the borehole(s) to underground.

17.5.15 Sampling and Process Monitoring

17.5.15.1 Plant Feed Solids

The SAG mill feed conveyor weightometer will measure the (wet) new solids feed into the SAG mill. The control room operator (or metallurgist) will input an ore moisture value on the plant control system (PCS). This value should be updated as and when required to reflect change in feed condition and/or prevailing weather. The PCS will use the wet solids and moisture values to calculate the dry feed solids and will record a total for each shift, day, and week, as required by plant management.

17.5.15.2 Feed Sampling

Trash screen underflow will be sampled using a metallurgical sampler to produce 8-hour shift composites each with volume of 10 litres or less. Given the presence of cyanide in the process water, gold assays of the shift composite liquor and the solids will be required.

17.5.15.3 Gravity and Intensive Cyanidation

A flowmeter on the intense cyanidation product solution pipeline will measure the volume of each batch of solution discharged into the gravity eluate tank. A sampler will take a composite sample (multiple cuts at a specified interval) of the incoming solution. After electrowinning, a second liquid sampler will take a composite sample of the spent solution discharged to the leach feed box. The batch composite feed (loaded eluate) and spent samples will be sent for dissolved gold assay.

17.5.15.4 Elution and Electrowinning

Samplers on the electrowinning feed and spent eluate discharge pipelines will take samples for gold assay. A flowmeter on the spent eluate line will measure and totalise spent solution volume per batch.

17.5.15.5 Tailings Sampling

Detoxification feed slurry is sampled using a three-stage metallurgical sampler to produce a 12 hour shift composite with volume of 10 litres or less. Gold assays of the shift composite liquor and solids are required.

17.5.15.6 Tailings Solids

The tailings pump discharge flowmeter and density meter will be used to calculate the dry plant tailings solids rate and to record a total for each shift, day, and week, as required by plant management.

17.5.15.7 Gold Call and Metal Accounting

The plant gold call will be calculated by a metallurgist based on the data described above. Gold call will be compared with gold production per shift, and also on daily basis. Trends will be monitored and reported for action if required.

17.5.16 Reagents and Consumables

17.5.16.1 Storage

Reagents will be received on site either in bulk or in shipping containers, with a 28 to 60 days' worth of stock stored on site, depending on the reagent and source of supply. This will ensure that supply interruptions due to port, transport or weather delays do not restrict production.

17.5.16.2 Grinding Media

SAG and ball mill grinding media will be delivered to site either in bulk, in drums or in bulk bags. SAG mill grinding media will be reclaimed by FEL from the SAG mill ball bunker and loaded into the SAG mill ball charger. The SAG mill ball charger will periodically meter SAG mill grinding media onto the SAG mill feed conveyor.

Ball mill grinding media will be reclaimed by FEL from the ball mill ball bunker and loaded into the ball mill ball charger. The ball mill ball charger discharges ball mill grinding media to fill the ball mill ball charging kibble, which is lifted and discharged into the ball mill feed box.

Grinding media charging will be carried as required to maintain the target power draw in the SAG mill and ball mill.

17.5.16.3 Quicklime

Quicklime will be delivered to the site as a dry powder in bulk lime tankers and pneumatically transferred into the lime silo using a lime blower. The lime silo will be equipped with a reverse pulse bag filter, to minimise dust emissions during loading, and a bin activator.

Quicklime will be metered from the lime silo via a variable speed rotary valve directly onto the SAG mill feed conveyor to achieve the desired pH downstream in the cyanide leach circuit.

17.5.16.4 Flocculant

Flocculant will be delivered to site in bulk bags as a dry powder and will be added to the flocculant bag breaker. The vendor supplied flocculant mixing plant will automatically mix batches of flocculant with raw water and transfer the mixed flocculant to the flocculant storage tank after each mixing cycle is complete. Dedicated pumps will meter the flocculant solution to the pre-leach thickener and tailings thickener, where the flocculant solution will promote settling of solids and improve solid-water separation within the thickeners.

Liquid flocculant will be delivered to site in 1,000 L intermediate bulk containers (IBCs) and pumped to the intensive cyanidation reactor as required using a dedicated dosing pump.

Area spillage will report to the flocculant area sump pump which will discharge to the Tailings hopper.

17.5.16.5 Cyanide

Sodium cyanide (NaCN) will be delivered as dry briquettes in bulk bags. The cyanide bags will be lifted into the cyanide bag breaker above the agitated cyanide mixing tank and dissolved in process water to achieve the required solution concentration. The mixed cyanide solution will be transferred to the storage tank for use in the process.

Cyanide will be reticulated via a ring main from the cyanide storage tank to the intensive cyanidation reactor and the CIL circuit to act as a gold lixiviant, and to the elution circuit for the desorption of gold from loaded carbon.

Cyanide area spillage will be directed to the lime/cyanide area spillage pump.

17.5.16.6 Activated Carbon

Activated carbon will be delivered in bulk bags and will be added via the carbon quench hopper to top up the inventory of activated carbon within the CIL circuit as required. This addition point will allow any fine carbon particles to be removed via the carbon sizing screen prior to entering the CIL circuit. Activated carbon will act as the adsorbent for precious metal species dissolved by cyanide in the CIL circuit.

17.5.16.7 Hydrochloric Acid

Concentrated hydrochloric acid (HCl) will be delivered to site in intermediate bulk containers (IBCs). The concentrated hydrochloric acid will be pumped into the HCl acid mixing and storage tank where it will be diluted with filtered water to achieve the required acid wash solution concentration. The dilute acid solution will be pumped to the acid wash column to remove inorganic impurities from loaded carbon into an effluent solution stream.

HCl area spillage will be collected in a sump and pumped to the tailings hoper, when required.

17.5.16.8 Caustic Soda

Caustic soda (sodium hydroxide, NaOH) will be delivered to site as dry pearl pellets in bulk bags. The caustic soda bags will be lifted into the caustic bag breaker above the agitated caustic mixing tank. A pre-set number of bags will be added to water in the caustic mixing tank and the caustic mixing agitator will be started. After a pre-set mixing time, the caustic transfer pump will start, transferring the 20% NaOH solution to the caustic storage tank.

Caustic dosing pumps will meter the caustic soda solution, as required, to the intensive cyanidation reactor, cyanide mixing and cyanide detoxification for pH control, to pregnant solution tanks to adjust solution conductivity for electrowinning, and into the elution circuit for the desorption of gold from loaded carbon.

Caustic area spillage will combine with cyanide area spillage.

17.5.16.9 Sodium Metabisulphite

Sodium metabisulfite (SMBS, $Na_2S_2O_5$) flake will be delivered to site in bulk bags. The SMBS bags will be lifted into the SMBS bag breaker above the agitated SMBS mixing tank. A pre-set number of SMBS bags will be added to water in the SMBS mixing tank and the SMBS mixing tank agitator will be started. After a pre-set mix time, the SMBS transfer pump will start, transferring the mixed SMBS solution into the SMBS storage tank.

Sodium metabisulfite solution will be metered to the cyanide detoxification circuit and is the chemical source of SO_2 required, in combination with oxygen, to react with WAD cyanide species to reduce the TSF supernatant WAD cyanide solution concentrations below target levels.

17.5.16.10 Copper Sulphate

Copper sulphate pentahydrate ($CuSO_4.5H_2O$) will be delivered to site as a powder in bulk bags. The copper sulphate bags will be lifted into the copper sulphate bag breaker and discharged into the agitated copper sulphate mixing tank. A pre-set number of bags will be added to water in the copper sulphate mixing tank and the copper sulphate mixing tank agitator will be started. After a pre-set mix time, the copper sulphate transfer pump will start, transferring the mixed copper sulphate solution into the copper sulphate storage tank. Copper sulphate solution will be metered to the cyanide detoxification circuit where it will act as a catalyst for the cyanide detoxification reactions.

17.5.16.11 Fluxes

Sodium tetraborate (anhydrous borax, $Na_2B_4O_7$), silica flour (SiO_2), potassium nitrate (KNO_3) and sodium carbonate (soda ash, Na_2CO_3) will be used as fluxes for gold smelting. The fluxes will be delivered in 25 kg bags and mixed in small quantities with the gold sludge prior to smelting.

17.5.17 Services

17.5.17.1 Raw Water

Raw water for the Project will be sourced from the mine dewatering bores and pumped to the raw water pond, located at the process plant.

The raw water pond will be sized with sufficient capacity to minimise the impact of short-term supply interruptions. Raw water pumps will distribute raw water to the process plant and other surface infrastructure for use. Raw water will also be used as feed to the filtered and potable water treatment plants.

17.5.17.2 Tailings Run-Off

Although no water is expected to be released from the filtered stack TSF, run-off will be collected and stored in the tailings water pond to be used as the pre-dominant source of water to top-up the process water demand as required.

17.5.17.3 Process Water

Water recovered as filtrate from the tailings filters and overflow from the pre-leach thickener will be directed to the process water tank, which will provide most of the water for the processing facility. Tailings run-off from the water storage facility will also be added to the process water tank as required to make up the process water requirements serviced by this tank.

The process water tank will service the following water requirements:

- Grinding area process water.
- Vibrating screen wash water.
- Reagent mixing, where appropriate.
- Gravity concentration.

Provision will be made to allow for the addition of antiscalant to process water to reduce fouling of the pipelines, spray nozzles and screen decks.

17.5.17.4 Fluidising Water

Fluidising water to the gravity centrifugal concentrators will be provided by dedicated fluidising water pumps. The pumps will draw water from the process water pond, and the water will be filtered inline to remove excess particulates.

17.5.17.5 Filtered Water

Filtered water for the process plant will be produced by treating raw water in the filtered water treatment plant to remove any suspended solids and bacterial contaminants. The filtered water treatment plant will be of a multi-stage particulate filtration design, producing filtered water product suitable for use within the process plant. The filters will be periodically backwashed to remove solids, with the effluent stream transferred to the tails hopper.

Filtered water will report to the filtered water tank and will be distributed to the plant by the filtered water pumps for use as required, predominantly for SAG and ball mill seal water and in elution, mercury abatement packages, the gold room, and reagent mixing.

17.5.17.6 Gland Seal Water

Water from the filtered water storage tank will be used as gland seal water to slurry pumps and will be pumped at the required service pressures by gland water pumps.

17.5.17.7 Fire Water

Fire water for the process plant and surrounding surface operations will be sourced from the raw water pond. The fire water suction from the raw water pond will be at a lower level than the raw water supply suction to ensure a fire water reserve always remains in the pond.

The firewater pumping system will contain:

- An electric fire water jockey pump to maintain fire water ring main pressure.
- An electric fire water pump to supply fire water at the required pressure and flow rate.
- A diesel driven fire water pump that will automatically start in the event that power is not available for the electric fire water pump, or if the electric pumps fails to maintain pressure in the fire water system.

Fire hydrants and hose reels will be placed throughout the process plant and surrounding surface operations at intervals that ensure complete coverage in areas where flammable materials are present.

17.5.17.8 Potable Water

Potable water will be produced by treating raw water in two identically sized potable water treatment plants. Each plant will include feed pumps, micro filtration, ultra-violet sterilisation and chlorination equipment.

Potable water will report to the potable water tank and will be reticulated to the plant (including safety showers), administration facilities, site ablutions and the MSA. Potable water will also be pumped to a camp potable water tank located on the hill above the permanent accommodation camp, and will be gravity fed to the camp accommodation and service facilities.

The potable water demand for the Project has been estimated on a per capita usage basis, with 75 L/person/day used for operational areas of the Project, and 300 L/person/day used for camp residents.

17.5.17.9 Sewage Treatment

Grey water and effluent from all plant, mining infrastructure and administration areas will drain to sewage transfer stations in their respective areas. The sewage transfer stations will pump effluent to the sewage treatment plant (STP) located adjacent to the process plant.

Treated grey water from the STP will be pumped to the tailings water storage dam. Treated sludge will be stored in the STP sludge tank and periodically disposed off-site of by sewage truck.

17.5.17.10 Plant and Instrument Air

Plant and instrument air will be supplied from air compressors. The air will be filtered and dried before distribution to separate area specific air receivers that will then supply air to the plant.

17.5.17.11 Diesel

Diesel will be delivered to site by bulk tankers and transferred to bulk storage tanks adjacent to the mining infrastructure area, for use by the heavy and light vehicle fleet. The bulk fuel storage facility is discussed in further detail in Section 18.11.

Diesel will be pumped from the bulk storage facility to the process plant diesel day tank, which will supply fuel to the strip solution heater, carbon regeneration kiln, and gold room smelting furnace.

17.5.18 Control System

17.5.18.1 General

The general control philosophy for the plant will be one with a high level of automation and remote-control facilities. Instrumentation will be provided within the plant to measure and control key process parameters.

The main control room will house the personal computer based operator interface terminals (OIT). The control room is intended to provide a central area from where the plant is operated and monitored and from which the regulatory control loops can be monitored and adjusted. All key process and maintenance parameters will be available for trending and alarming on the PCS.

The PCS will be based on modern best practice and follows typical equipment application for such a process plant. The control system selection components are based on a SIEMENS Supervisory and Data Acquisition System (SCADA) and Programmable Logic Controller (PLC) system architecture. The PCS will control the process interlocks and proportional-integral-derivative (PID) control loops for non-packaged equipment.

Vendor supplied packages will use vendor standard control systems throughout the project. Vendor packages will generally have limited interfaces with the PCS such that the control and set-point changes may have to be adjusted locally. General equipment fault alarms from each vendor package will be monitored by the PCS system and displayed on the OIT.

17.5.18.2 Field Input / Output

All instrumentation and field controls will be captured by the PCS via remote input/output (I/O) modules, generally located at each process module of the plant, primarily to minimise extended cable run lengths back to the switchroom. These remote I/O modules are electrical field enclosures housing power supplies, network equipment, marshalling terminals and PLC remote I/O nodes. All remote I/O nodes will be linked to the PLC via a PCS Ethernet network, supported over the site fibre optic cabling between switchrooms, remote I/O modules and key infrastructure buildings.

17.5.18.3 Drive Control

In general, the plant process drives will report their ready, run and start push button status to the PCS and will be displayed on the OIT. Local control stations will be in the field in proximity to the relevant drives. These will, as a minimum, contain start and latch-off-stop (LOS) pushbuttons which will be hard-wired to the drive starter. Plant drives will predominantly be started by the control system in automatic operation.

The OITs will allow drives to be selected to Auto, Local, Remote or Out-of-Service modes via the drive control pop up. Statutory interlocks, such as emergency stops and thermal protection, will be hardwired and will apply in all modes of operation. All PLC generated process interlocks will apply in Auto and Remote modes. Process interlocks will be disabled or bypassed in Local mode, except for critical interlocks such as lubrication systems on grinding mills.

Local selection will allow each drive to be operated by the operator in the field via the local start push button. Remote selection will allow the equipment to be started from the control room via the drive control pop up. Status indication of process interlocks, as well as the selected mode of operation, will be displayed on the OIT.

17.5.18.4 Control Loops

Regulatory control loops will be provided for all key process circuits to provide optimal functionality without regular operator intervention.

There will be two modes for loop-controlled set points available in the OIT. These are 'Loop Auto Mode' and 'Loop Manual Mode'. In Loop Auto Mode (analogous to cascade control), the set point will be predominantly controlled by the applicable master PID loop (e.g., for thickener underflow pumping control, the bed pressure PID controller output will supply a set point for the thickener underflow flow control loop, which ultimately controls the speed of the thickener underflow pump). In Loop Manual Mode, set point may be entered manually from the loop set point pop up in the OIT.

Where required, analogue set points from the PCS to vendor supplied control panels can be done either via the OIT or via vendor control panels.

17.6 Predicted Metallurgical Performance

Section 13.6 discusses the parameters selected for predicting metallurgical performance when processing different ore sources over the life of mine. Parameters have been selected based on metallurgical testwork and modelling. The metallurgical parameters are influenced by ore gold grade and lithology.

The life of mine processing production schedule by lithology and grade is presented in Figure 17.3 and the life of mine production schedule is presented in Figure 17.4.

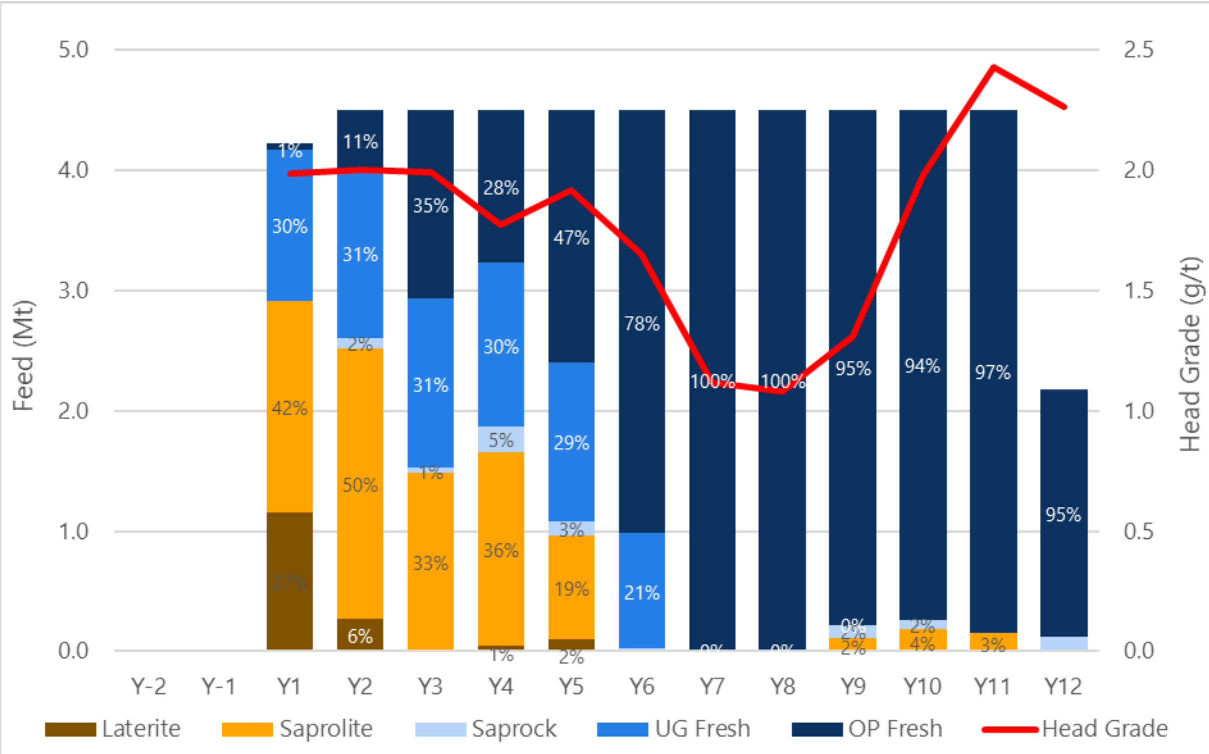

Figure 17.3: Life of Mine Schedule – Mill Feed by Lithology and Grade

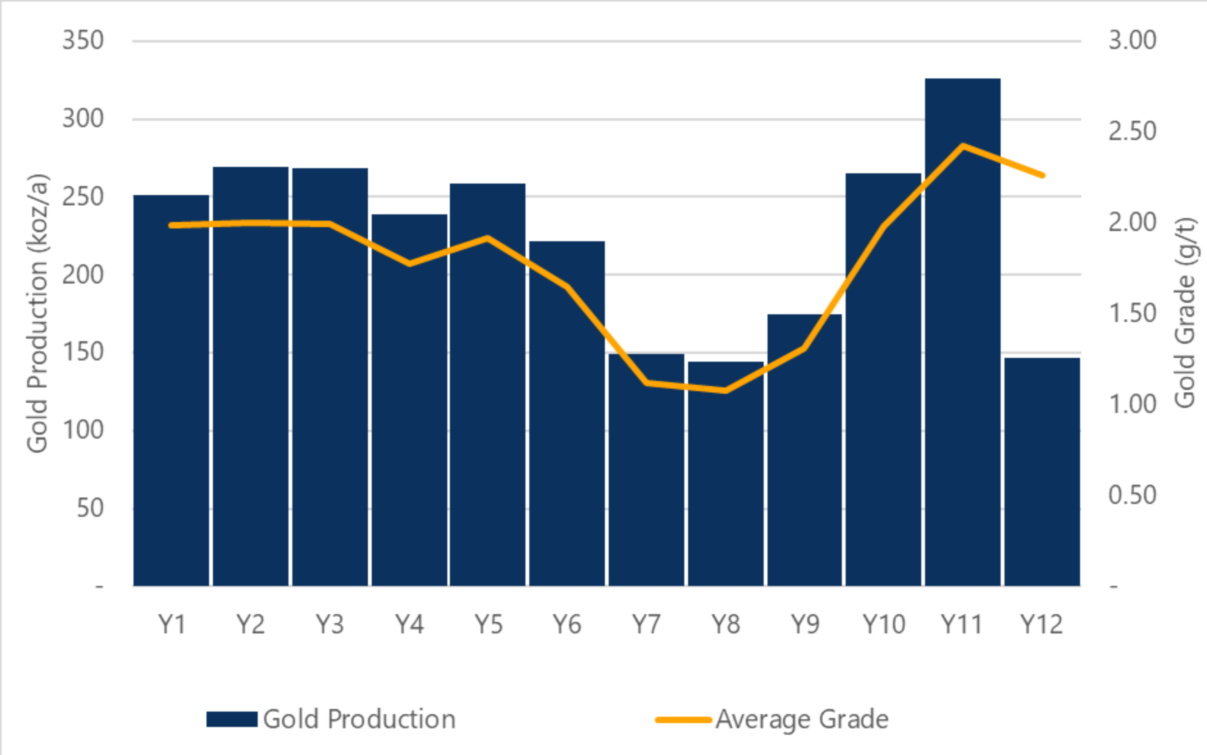

Figure 17.4: Gold Production and Grade

17.7 Power Requirements

Power is supplied to the Project from a dedicated power station, and associated solar farm, installed at site. The power cost has been calculated based on a fixed and variable charges under a power purchase agreement, HFO price and a degree of renewable energy penetration as outlined in Section 18.6.

The site wide average electrical power requirement covering the processing plant, mining and site infrastructure has been calculated based on the equipment sizing and demand. The connected load, average draw and average annual energy consumption for the Project is summarised in Table 17.2.

Table 17.2: Project Electrical Power Demand

Area	Installed	Average Continuous Draw	Annual Consumption
	kW	kW	GWh/a
Crushing	788	477	3.1
Stockpile and reclaim	427	354	2.6
SAG and ball mill	20,000	14,303	114.4
Grinding and classification	3,833	1,748	14.0
Gravity and intensive cyanidation	149	120	1.0
Pebble recycling	200	120	1.0
Pre-leach thickening	404	285	2.3
Trash Screening and CIL	1,367	1,044	8.3
Elution, gold room	579	397	1.7
Tailings Detox and Filtration	2,813	1,371	8.7
Paste Plant	110	62	0.4
Reagents	225	184	1.4
Services – water, air and diesel	1,857	735	5.9
TSF	230	88	0.7
Borefields	318	222	1.8
Plant, mining and admin buildings	1,374	941	7.9
Camp	1,119	783	5.1
Total	**35,790**	**23,233**	**180.3**

17.8 Water Requirements

The processing plant will use water sourced from the peripheral mine dewatering bores, TSF run-off via the tailings water storage dam and recovered process water via the pre-leach thickener and tailing pressure filters.

The average instantaneous demand for process water and raw water is approximately 1,500 m^3/h, however the majority of this supply is provided by the pre-leach thickener and tailings filter as water recovered within the processing facility.

The mass balance of the processing facility estimates an average makeup water volume of approximately 140 m^3/h (100,000 m^3 per month) will be required to balance water losses due to:

- 20% moisture to the filtered tailings stack.
- 33% moisture to paste feed.
- Evaporation.

The process mass balance estimates that, due to water quality constraints, 20% of this water demand must be of sourced from the raw water which is suitable for filtered water and potable water. The remaining water demand is provided from tailings water storage dam.

A 5,000 m^3 site raw water dam will provide buffer volume for process plant demand, if required.

Within the process plant, the process water tank will will receive the tailings filtrate and pre-leach thickener overflow and can be topped up with raw water or water from the tailings water storage dam.

Potable water treatment plants will be supplied by filtered raw water (from peripheral mine dewatering bores).

17.9 Conclusions

The Qualified Person considers that the recovery methods outlined above as suitable for implementation of the previously outlined mineral processing and metallurgical findings to treat the Mineral Reserve and achieve the performances predicted by the metallurgical testing of the samples.

18 PROJECT INFRASTRUCTURE

18.1 Introduction

A range of non-process infrastructure (NPI) will be required to enable operations at the Project. This will include the following:

- Earthworks.
- Access roads.
- Offices, warehouses, workshops and other buildings.
- Accommodation village.
- Power supply and site power distribution.
- Tailings storage facility.
- Surface water management structures and systems.
- Water supply and management.
- Mining infrastructure.
- Fuel storage and supply.
- Paste plant.
- Other infrastructure.

An overall site layout is provided in Figure 18.1, which shows the supporting infrastructure and services, mining locations, and processing plant infrastructure.

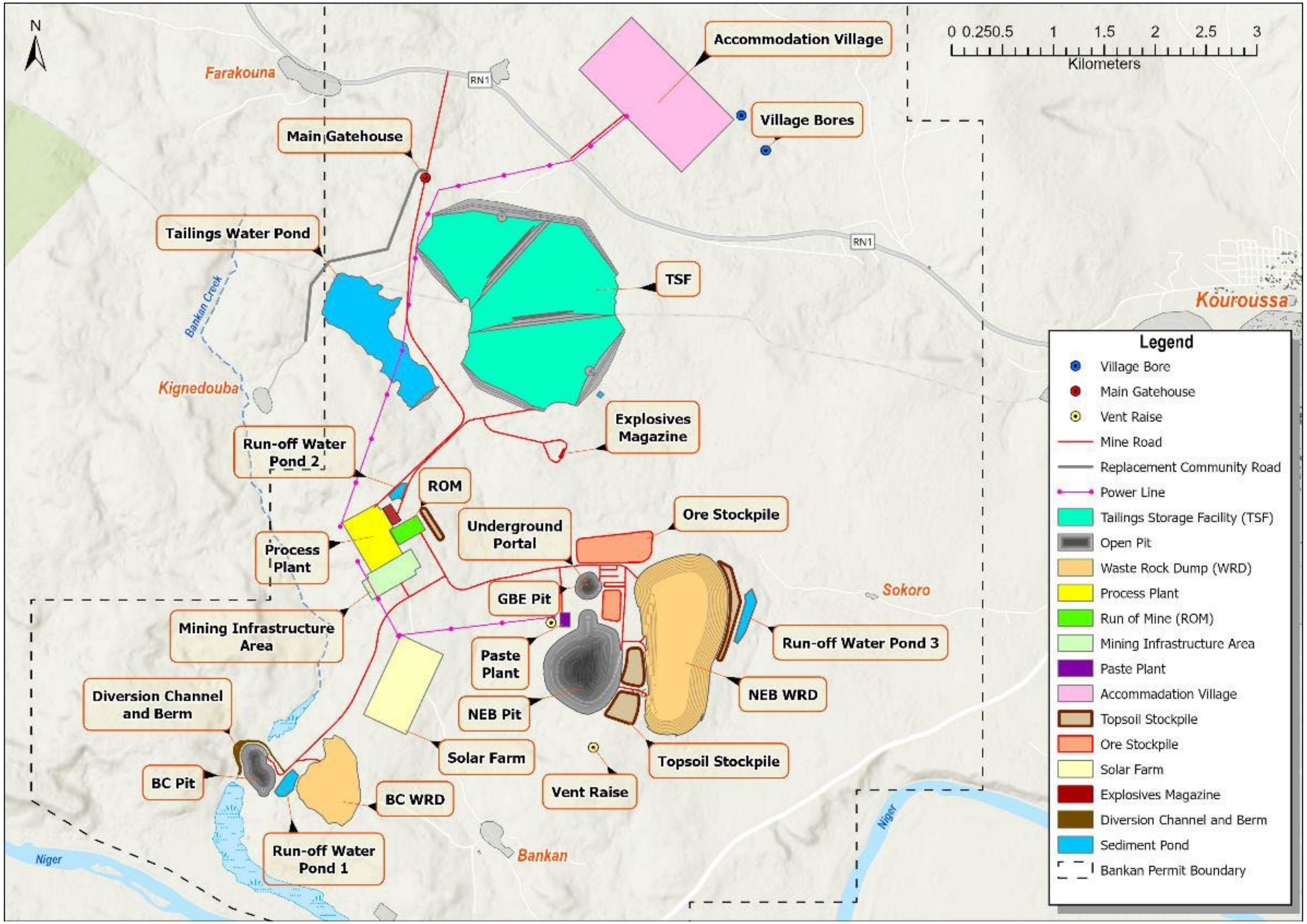

Figure 18.1: Overall Site Layout

18.2 Earthworks

The earthworks required for the Project include:

- Roads.
- Tracks.
- Pads for buildings and infrastructure.
- TSF.
- Water ponds.
- Surface water management infrastructure.

In order to provide a basis for the design and costing of the earthworks for the Project a detailed investigation of the geotechnical conditions was undertaken as part of the DFS. The infrastructure geotechnical investigation consisted of drilling, with standard penetrometer tests (SPT), test pits, dynamic cone penetrometer (DCP) tests, infiltrometer tests and bulk sampling for testing. Investigations were completed as follows:

- Six diamond core boreholes with SPT (three in the TSF and three in the process plant site including primary crusher, mills and tank locations). Samples were recovered using Shelby tubes.
- Eight test pits completed at the TSF.
- 193 DCP tests across the TSF, process plant site, site access road, mine haul roads, explosives storage area and along channels and bunds defining surface wate management infrastructure.
- 13 infiltrometer tests within key surface water catchment areas.
- Bulk samples taken from existing excavations from within the project site to evaluate the materials suitability for use as bulk fill and granular pavements.

Representative samples of the in-situ soils were taken from the boreholes and test pits for laboratory testing. The purpose of the laboratory testing was to classify and characterise the in-situ materials to assess their behaviour characteristics under embankment and foundation loading, and their suitability for use in earthworks.

A summary of the infrastructure geotechnical investigation concluded that:

- The TSF area has suitable borrow material to construct the starter and future embankments and will be supplemented with future mine waste. Whilst initial investigation findings indicated that the selected TSF is suitable for embankment and liner construction, additional foundation strength parameters need to be assessed. Groundwater is expected to be shallow and as such suitable dewatering measures are required.
- The plant site investigation indicates a surface hard cap exists below which a soft clay is present. As bedrock was not reached in the investigation, foundation design requires delineation of the extent and thickness of the hard cap and depth to competent rock on a structure case by case basis for heavier loaded areas of the process plant.

- The surface hard cap ground at those areas with lighter equipment loads e.g. modular buildings, light steel framed structures and the accommodation village will support typical strip footings and ground slabs typically associated with this type of infrastructure.
- The hard cap area will require ripping by bulldozer (or similar) as part of the bulk earthworks operations.
- Structural fill and road base can be sourced locally from select borrow pits. Concrete sand and aggregate were not found on site and will be required to be sourced from off-site suppliers.
- Other infrastructure including the access roads, surface water infrastructure and the accommodation village indicates suitable ground support conditions.

The details relating to the earthworks is outlined in the separate sections relating to the process plant, site access, tailings disposal and surface water management.

18.3 Site Access

The N1 sealed highway from Conakry to Kouroussa transects the Project exploration permits approximately 4.5 km to the north of the process plant location. Access to the mine and plant area will be enabled by construction of an all-weather unsealed road that connects to the N1 highway approximately 7 km along the highway to the west of Kouroussa. This access road will pass through the main site gate house, approximately 1 km from the highway, before running adjacent to the TSF, crossing the creek line to the process plant area gate house approximately 5 km from the highway.

Access to the mining areas at NEB/GBE and BC will be via separate unsealed roads, dedicated to mining heavy vehicles, from the mining infrastructure area, adjacent to the process plant. Access to the TSF will be via a dedicated unsealed heavy vehicle road from the tailings stockpile area at the process plant, running adjacent to the mine access road.

The existing unsealed road accessing the village of Kingédouba, which will be blocked through the construction of the Project, will be realigned to join the mine access road prior to the main site gatehouse.

Transport for personnel during construction and operations will either be via the via the highway from Conakry, a distance of approximately 570 km, or via charter flight to Kankan airport, which is approximately 105 km by road from site.

18.4 Offices, Warehouses, Workshops and Other Buildings

The key building infrastructure for the Project includes all non-mining facilities outside the processing plant that is required to support the plant or processing functions. Processing plant infrastructure includes:

- Administration building, that will accommodate all management personnel including administration, security, health safety and environment, processing, maintenance and mining. The building will consist of a combination of separate offices and open plan workstations.
- Two separate dining halls with a centralised meals preparation and dispensing area to cater for staff and operators respectively.
- Plant control room.

- Security buildings, including a main access gatehouse facility located at the site access road approximately 1.5 km from the N1 highway. A guardhouse will be located at the entrance to the administration area of the plant site and a second, main guardhouse will be located at the entrance to the process plant, opposite the gold room and adjacent the administration building. The process plant gatehouse will include provision for the first aid/medical room and emergency response team and safety personnel.
- Fixed plant workshop including office.
- Plant warehouse including office.
- Laboratory.
- Male and female ablutions.
- Prayer rooms.

18.5 Accommodation Village

The accommodation village is located approximately 8.5 km from the processing plant on the northern side of the N1 highway. The accommodation village will consist of permanent facilities, which are sized only for the owners team and engineering, procurement and construction management (EPCM) contractors during construction and for senior staff during operations. Additional area surrounding the permanent accommodation village facilities will be allocated for contractor's camp facilities, including temporary construction workforce camps provided by construction contractors during the construction period and potential mining contractor camp(s) during operations.

The permanent accommodation village will accommodate a workforce of 140 senior staff. Rooms will consist of:

- 44, 3-bedroom standard units.
- Three, 2-bedroom management units.
- Two, 1-bedroom VIP apartments including small kitchenette and an outdoor deck.

In addition to the accommodation rooms, the accommodation village will include all standard facilities found in typical west Africa operations.

The kitchen and dining facilities will be sized to provide messing to residents during both the construction and operations phases, but more specifically, additional facilities will be installed to provide messing to approximately 400 contractor junior staff during the construction phase.

The accommodation village will also include potable and wastewater treatment plants that will be sized to treat the entire construction phase workforce and then be scaled down to treat the smaller operations phase workforce. Two water bores located adjacent the accommodation village provide raw water via a buried pipeline to the potable water treatment plant.

Power during the construction phase will be supplied via 250 kVA generators housed within a sound attenuated canopy. These generators will be used as back up emergency power once the overhead powerline from the power station is operational.

Fuel for both generator supply and refuelling of light vehicles will be supplied in a 30,000 litre self-bunded diesel tank.

As the Project progresses from construction to operations, it is intended that the diner capacity be reduced (if required). During operations, the kitchen will be required to provide lunches to approximately 250 personnel each shift over the three 8-hour shifts.

18.6 Power Supply and Distribution

The estimated load for the Project is outlined in Table 18.1.

Table 18.1: Estimated Site Load

Area	Load (MW)
	Average
Underground Mining (steady state)	4.8
Processing Plant (excluding mills)	7.5
SAG and Ball Mills (based on LOM average blend)	14.0
Infrastructure	0.7
Accommodation Village	0.8
Total	**27.7**

Power will be supplied from an on-site heavy fuel oil (HFO) power station located adjacent to the processing plant. The power station will generate up to 32.5 MW of continuous power using reciprocating engines. In addition, a solar photovoltaic (PV) array will be employed with a battery energy storage system to minimise hydrocarbon fuel use. The solar PV will be installed between the processing plant and the mine areas and connected to an 11kV power line suitable for transmitting up to 40MVA.

The power supply is to be provided on a build-own-operate (BOO) basis and will consist of:

- Power station will have a nominal thermal capacity of 32.5 MW at 0.8 power factor on an N+1 basis at site conditions. The design at average load includes 12 HFO generating sets online with one idle HFO generating set and two idle high-speed diesel emergency generating sets providing a combined N+2 redundancy. Engines will be operated at a minimum of 30% load while the solar generation is online and with a minimum of two generating sets online at all times. The selected generating sets have a continuous capacity of 2,500 kWe at site conditions.
- Emergency power at the power station will consist of two 1.6 MW high speed diesel generating sets with a capacity of 2,250 kVA to provide back-up power to the HFO generators power during periods of low plant load.
- Solar capacity will be 30 MWp (DC)/26.9 MW (AC) with a battery energy storage system (BESS) with a capacity of 5 MWhr/5 MW. The battery will provide up to one hour of storage capacity

for critical load, and also provides the necessary reserve capacity to the system in managing the solar PV variability.

- Generating sets will be maintained online at all times providing full frequency and voltage control regulation to the system. Stabilising support will be provided by the BESS during periods of peak solar generation. During non-availability hours of battery and solar, the engines will cater for the full load demand.

Electricity from the power station will be distributed at 11 kV to the process plant and associated infrastructure via high voltage cables and using overhead power lines.

Local diesel generation will be included in the following areas:

- Accommodation village, where backup diesel generators will be permanently installed for use prior to the power station commissioning and in case of an outage thereafter.
- Mine power supply, comprising of three diesel generators rated at 1,875 kVA each will be installed for use prior to the power station commissioning and in case of an outage thereafter. These generators will provide sufficient power to maintain underground mine safety, however not sufficient to continue full mining operations. Allowance has, however, been included in the substation at the underground mine to install additional generation capacity and fully operate the mine in the event of a sustained outage of power supply from the power station.
- Main site gatehouse, where a 100 kVA diesel generator will be installed for use prior to the power station commissioning and in case of an outage thereafter.
- TSF water storage dam, where a 150 kVA diesel generator will be installed for use prior to the power station commissioning and in case of an outage thereafter.
- Diesel generators will also be used for the operation of the GBE and NEB dewatering bores prior to the power station commissioning.
- Diesel generators will be used for the operation of the BC dewatering bores at all times, although this is late in the LOM and is only a short-term requirement.

18.6.1 Grid Connection

The national electricity grid in Guinea is currently being expanded to connect the grid to Mali to provide additional power. This connection, the Mali interconnector, is currently under construction and will connect to the Transco Côte d'Ivoire–Liberia–Sierra Leone–Guinea (CLSG) interconnector at N'Zérékoré in the south of the country at 255 kV and run through Kankan and Fomi, approximately 75 km and 28 km to the southeast of the Project respectively. The Mali interconnector is due for commissioning by the end of 2025.

A second 255 kV interconnector is also planned from Fomi to Linsan, in the country's northwest, to link across the centre. This interconnector would run close to the Project site, however the timing of its design and construction is unknown and considered unlikely in the near term.

During the DFS investigations were undertaken into the connection of the Project to the expanding national electricity grid with the most feasible option being to connect to the Mali interconnector at Fomi and run a dedicated power line the short distance to the project site. This was costed, however, with the average cost of power from the grid being approximately the same as a BOO HFO fired hybrid power station, it was not deemed advantageous to proceed with this option.

18.7 Tailings Disposal

18.7.1 Tailings Storage Facility Selection and Operation

A filtered stack TSF will be adopted for the Project based on the key following reasons:

- Location of the Project within the Peripheral Zone of the Upper Niger National Park.
- Relative location of the Project to the regional centre of Kouroussa.
- Low risk of any leachate from the TSF entering groundwater.

The location, material technology and embankment alignment of the proposed facility have been selected by conducting a multi-criteria analysis (MCA) workshop. This selection made efficient use of natural topography and available land while remaining remote from the majority of mining and public infrastructure, and remaining outside of flooding areas and creeks.

In addition to storing filtered tailings, the design and operation of the TSF is aimed at managing the operational footprint, with sufficient evaporation area such that the water balance is negative over the life of mine, and no discharge of water from the operating facility to the environment is required. To achieve this, the facility will be a three-cell configuration with a total footprint area of approximately 270 ha with each cell effectively one third of this total footprint.

Each cell will be constructed and operated separately with the next cell constructed prior to the operating cell reaching full stacking design height. Once the next cell is in operation, the previous cell will be rehabilitated, and clean runoff water from this rehabilitated area will be discharged to the environment via a small sediment capture dam, which will also allow for ongoing monitoring of runoff water quality from the rehabilitated cells.

Each cell with contain a decant structure which will collect any tailings runoff water, and seepage. This collected water will be pumped to the adjacent tailings water storage dam for re-use in the process plant or will evaporate naturally from the surface of the dam.

A layout of the TSF and tailings water storage dam is shown in Figure 18.2.

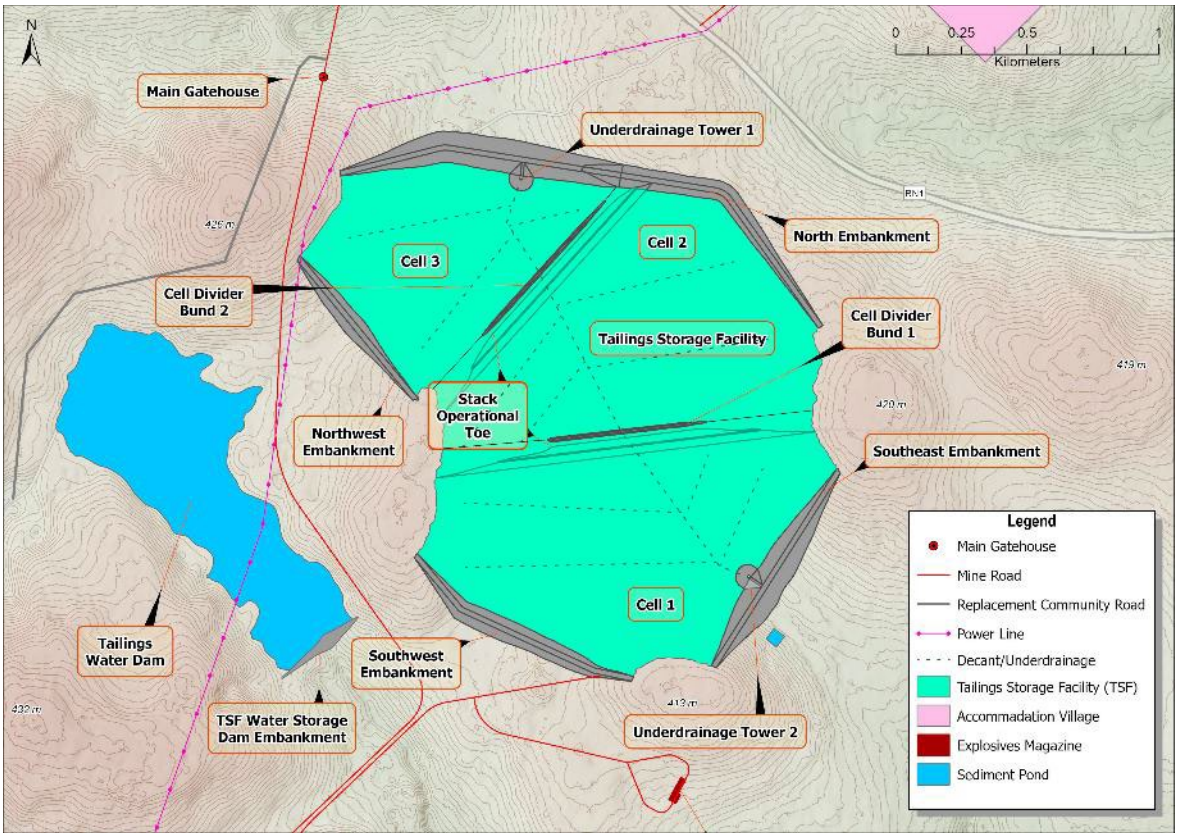

Figure 18.2: TSF and Tailing Water Storage Dam

18.7.2 Tailings Storage Facility Design and Construction

The design of the TSF was prepared in accordance with the requirements of international standards, the Global Industry Standard for Tailings Management (GISTM), and the Australian National Committee on Large Dams (ANCOLD) 'Guidelines on Tailings Dams - Planning, Design, Construction, Operation and Closure'.

The external walls of the TSF will consist of a two-zoned, downstream profile embankment with the upstream slope lined with a high-density polyethylene (HDPE) textured liner. The embankment will utilise basin borrow (fine-grained soil) and mine waste (mixture of fine-grained soil and waste rock) when it is available for future lifts. The basin will be lined with smooth HDPE liner which will be placed over the internal embankments between the individual cells to provide anchoring.

Internal embankments will allow access across the facility and provide runoff containment from the active tailings stacking area before being pumped from the facility. The filtered tailings will be placed at a distance from the bund with intermediate batter slopes to maintain stack stability. With this approach, raising of the internal embankments will not be required.

A nominal cut-off trench will be located beneath the entire length of the external embankments and excavated into a competent foundation layer. An underdrainage system, consisting of an embankment toe drain and main collector drains, will be installed on top of the basin HDPE liner. The underdrainage system will flow by gravity to the decant towers, where underdrainage water can be collected and pumped to the tailings storage water dam.

Tailings runoff water, from rainfall, and seepage will be removed from the TSF cells by a submersible pump located within a vertical concrete slotted decant tower in a sump adjacent to the internal bund. The decant tower will be located in a catchment basin in each cell and be accessed by a small causeway. Water recovered from the decant tower will be pumped to the tailings storage water dam.

Under the HDPE liner, a subsurface drain will be installed that flows by gravity underneath the southern embankment in cell 1. This will prevent groundwater uplift under the facility due to the shallow groundwater encountered.

The facility's external walls will be built in stages, with each cell having two stages. The first stage in cell 1 is sized to contain 2 years of production at an embankment level of RL 404 m (approximately 16 m high). Raises will be constructed on a 2 to 3 year cycle matching with the wet and dry season to the final elevation of RL 410 m (approximately 22 m high).

An access road from the process plant site will enter the facility from the southwest corner and deliver filtered material via low ground bearing pressure trucks (ADT60 or similar) and also serve as the decant return pipeline access track. Filtered tailings will be placed into the TSF in 1 m to 2 m layers where trucks will be used to traffic compact the material.

The facility is designed to contain the 1% average exceedance probability (AEP) 72-hour rainfall event without the decant operational and still maintain adequate freeboard to the spillway invert. Excess spillway runoff will, however, be contained in the adjacent cell where it will be allowed to evaporate minimising any potential impact on the environment.

A monitoring program comprising vibrating wire piezometers (VWP), survey pins/prisms and monitoring bores will be developed to monitor the facility during operations. In addition, solids material movement, moisture content of filter material, decant return rates and stack surveys will be tracked to calibrate in-situ densities and the water balance.

At closure of each cell, carried out progressively during the operation, the final surface profile will be suitable for rehabilitation with a capped, water shedding surface to several closure outlets located in abutments around the perimeter of the facility. A sediment control pond will be constructed downstream of cell 1 to allow this any sediment in this clean runoff to be settled and water quality monitored prior to final release. Planned material placement in the final year of each operating cell will ensure limited rehandling of filtered material is required. The surface will be covered with topsoil originally stockpiled from the project development. The final downstream embankment closure profile will also be constructed as part of the ongoing embankment construction and will only require minor reshaping and revegetation at closure.

The TSF is classified as being a "HIGH C" Dam Failure Consequence Category and a "SIGNIFICANT" Dam Spill Consequence Category according to the ANCOLD Guidelines on Tailings Dams (July 2019).

For the TSF facility to balance rainfall runoff, operational demand and evaporation losses without discharging water from operational areas to the environment, a HDPE lined tailings water dam will be constructed adjacent and to the southwest of the TSF.

The tailings water storage dam is a cross-valley embankment approximately 300 m in length, 10 m in height and consisting of a nominal cut-off trench located beneath the entire length of the embankments and excavated into a competent foundation layer. When at full capacity, it will have an evaporation area of approximately 611,000 m^2 and storage volume of 3.3 Mm^3, inclusive of a one metre freeboard allowance.

Cut-off diversion drains will be constructed around the dam to reduce catchment area and divert clean water downstream of the dam wall. A spillway will be included as part of the dam wall to maintain integrity in the event of the dam reaching capacity.

18.7.3 Tailings Storage Facility Water Balance

A life of mine water balance for the TSF and tailings water storage dam has been developed, on a monthly time interval, which accounts for catchment areas, water consumption in the process plant and movements in water inventory. In addition, the water balance takes into account:

- Average rainfall and evapotranspiration by month based on the AgERA5 datasets, which are the closest to the average of the public datasets.
- Typical annual variation in rainfall based on records from 1950 to 2019 applied as a normal distribution with a 9% standard deviation against the average.
- Allowance for climate change based on the projected near term (2021 – 2040) increase to average annual rainfall, which equates to an average 3.9% increase in rainfall above the historical average.

The life-of-mine TSF water balance was simulated, using 30 iterations, to develop a probabilistic estimate, along with P_5 and P_{90} estimates, of the seasonal water level in the tailings water storage dam across the life of mine. The results of this analysis are presented in Figure 18.3 which demonstrates that a maximum anticipated storage requirement of 88% of capacity will be required covering 95% of rainfall outcomes without discharge of water from the TSF system.

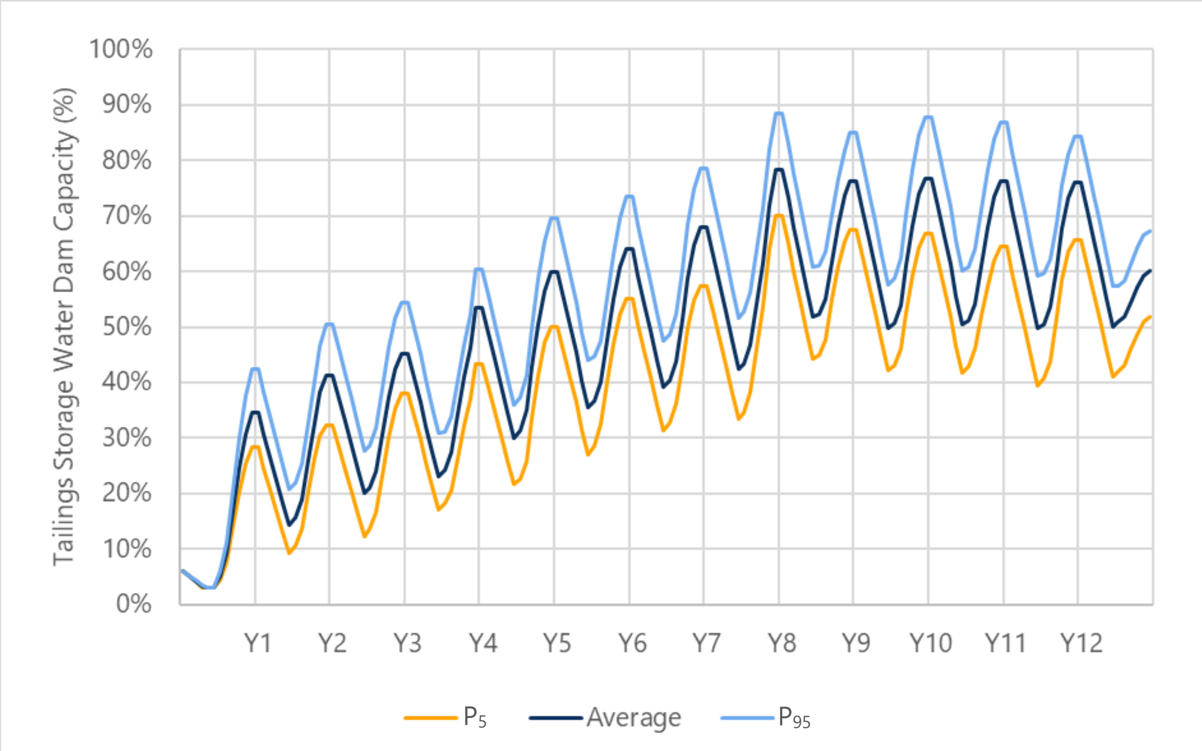

Figure 18.3: TSF Water Balance Probabilistic Analysis Results

In addition, on closure, the water storage facility will stabilise to between 30% and 50% of rainwater across the seasons which could be utilised by the local communities following monitoring to ensure water quality. This could provide a sustainable water supply of approximately 1 ML per day based on average rainfall year which would be sufficient to supply water to approximately 7,000 households. If this were a desirable outcome, the catchment and therefore sustainable water availability could be increased.

18.8 Surface Water Management

At its closest, the Project is located approximately 1.2 km north of the Niger River (southern end of the BC Pit) with the Niger catchment area approximating 17,120 km^2 at this point. The Niger is a major river and is highly seasonal, becoming impassable in the wet season, with limited flow in the dry season.

There are two dominant tributaries intersecting the site, both of which have headwaters within or near the mine site:

- Bankan Creek to the west which will be intersected by BC pit.
- An unnamed river north of NEB pit.

While the Niger River is perennial, the rivers intersecting the site are likely non-perennial given the well-defined wet and dry season. Additional minor streams (also likely non-perennial) are present within the site.

18.8.1 Climate

18.8.1.1 Design Rainfall

For the estimation of flooding and stormwater management, design rainfall is considered an important variable and the driver behind peak flows. To provide design rainfall estimates for the site the station at Kankan, approximately 75 km to the southeast was used.

A frequency analysis of the recorded daily rainfall was performed, and the generalised extreme value (GEV) maximum distribution was selected for the frequency analysis. In considering the annual maximum series (AMS) for Kankan, 31 years of data were available and a 1:62-year, return interval (RI) rainfall could be estimated with reasonable confidence. The 1:100-year RI was extrapolated from this and the 24-hour rainfall was estimated at 198.8 mm.

Considering the rainfall distribution over 24-hours, a review of rainfall trends did not conclude with any clear distribution of relevance to the region and site. The more intense (conservative with regards to flooding) SCS-SA Type III storm was thus adopted.

18.8.1.2 Monthly Rainfall

The synthetic CRU 4.09 dataset was selected for the long-term rainfall dataset compared to other datasets as it had the highest mean annual rainfall, being 1% higher than the Kankan dataset, 16% higher than CHIRPS and 15% higher than IMERG.

18.8.1.3 Evapotranspiration

Estimates of mean annual reference evapotranspiration were evaluated with estimates ranging between 1,648mm to 2,090mm. As with rainfall, the average of the estimates was used to define the relevant dataset. The AgERA5 was selected based on its being nearest to the annual average.

18.8.1.4 Niger River Peak Flows

For the Niger River, a highly seasonal flow from peak to near zero flow is evident in most years. Using the composite record of the Global Runoff Data Centre and Niger-HYCOS datasets, an AMS for relevant water years was subsequently extracted containing 79 years of data which allowed a reasonable estimation of the 1:100 year RI peak flow which was estimated at 2,061 m^3/s.

18.8.1.5 Climate Change

The analysis of climate change considered the outcome of the sixth Coupled Model Intercomparison Project (CMIP6) by Seneviratne et.al (2021) with a shared socio-economic pathways (SSP) SSP5 assumed (being a scenario where fossil fuel consumption continues to rise, with a focus on economic growth and technological innovation and limited attention to environmental sustainability) due to its increased influence on flooding and rainfall.

As the mine continues operations into the early 2040s, the medium-term projection according to the CMIP6 becomes relevant (ranging from 2041 to 2060), although the near-term projection (2021-2040) predominates for the majority of the Project life. The 1-day design rainfall (and consequently the previously outlined estimates for design rainfall) is projected to increase by 11.3% and 17.7% for the near and medium terms, respectively and the total annual rainfall is projected to increase by 3.9% based on the near term predictions and 1.1% based on the medium term predictions.

In the case of the Niger River, increased peak flows are anticipated with increased extreme rainfall. Given the large area of hydrological relevance of approximately 17,120 km^2 of Niger catchment, an area weighted increase in streamflow of 9.6% and 14% is estimated for the near and medium term respectively.

18.8.2 Regulatory Framework

The International Finance Corporation's (IFC) Environmental, Health and Safety Guidelines for Mining (2007) are the primary relevant guidelines, given the absence of any clear national guidance on mining (in relation to hydrology) for Guinea. The following points of guidance are relevant:

- Surface runoff from process areas or potential sources of contamination should be prevented.
- Separation of clean and dirty water areas is required, while minimising runoff, avoiding erosion of exposed ground surfaces, avoiding sedimentation of drainage systems and minimising exposure of polluted areas to stormwater.
- Dirty water areas include beneficiation plants, workshops (where oil and fuel is handled), residue disposal facilities, haul roads, opencast pits, and pollution control dams.
- Temporary drainage installations should be designed, constructed and maintained for RIs of at least a 25-year/24-hour event, while permanent drainage installations should be designed for a 1:100-year/24-hour event. Design requirements for temporary drainage structures should be defined on a risk basis considering the intended life of diversion structures and RI of any structures that drain into them.
- Reducing or preventing off-site sediment transport (e.g. use of settlement ponds, silt fences).
- Facilities should be designed for the full hydraulic load, including contributions from upstream catchments and non-mined areas.
- Where possible, maintain, restore or establish riparian zones to protect water courses.

18.8.3 Flooding Assessment

Potential flooding at the site was assessed for the current (status quo) scenario, using the aforementioned 1:25 and 1:100 RI events to define the design events of interest. In addition, with the relevance of climate change, simulations which considered the influence of climate change were included.

HEC-RAS 6.7 was utilised for the modelling of flooding, with two approaches adopted:

- Fluvial flood model, used for the Niger River based on the results of the frequency analysis (of recorded streamflow).
- Rain-on-Mesh (Pluvial) model which simulated the rainfall events over the area of hydrological relevance influencing the site (beyond the Niger River).

The most significant flooding event simulated was the 1:100 RI plus climate change event. The results of this simulation are presented in Figure 18.4.

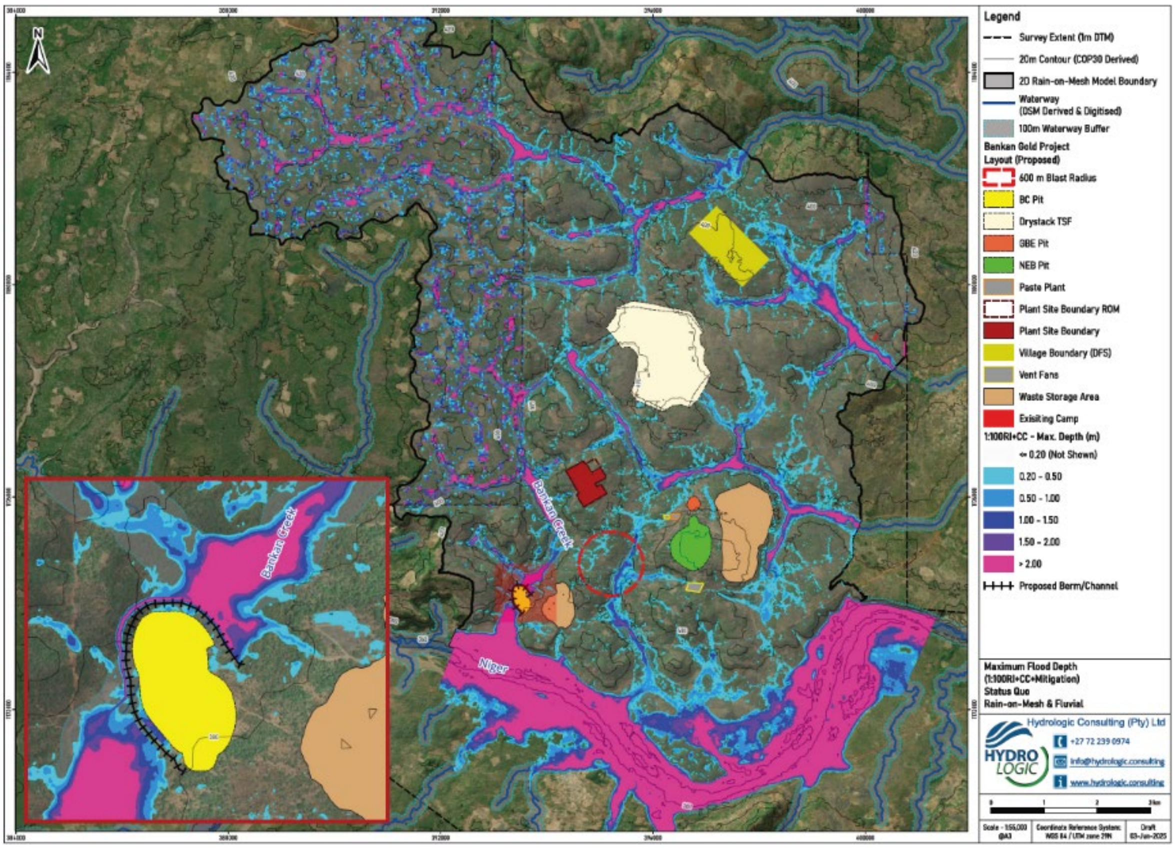

Figure 18.4: Flood Modelling Outcomes 1:100 RI plus Climate Change Event

In considering the results of this simulation, a minor river associated with the position of the TSF was noted as exhibiting flooding above 0.5m in depth. The proposed TSF will, however, replace the contributing catchment to this river, and this flood risk will consequently be mitigated through TSF development.

The only clear location of flooding on the site which will require mitigation is the BC pit, due to its position within Bankan Creek and potential, albeit minor, backflow from the Niger River. Mitigation of this flooding is proposed via a river diversion that involves a cut to the west of the BC pit over a distance of approximately 350 m and with a depth of up to 13 m (relative to existing terrain at maximum). Flood protection berms will prevent flood waters from entering the BC pit.

Secondary areas that will need to be considered for flooding are road crossings. These are, however, evaluated as part of the stormwater management plan, which has considered up to the 1:25 year RI (plus climate change) event in crossing design.

The majority of the site is otherwise, without significant flooding potential due to the position of proposed infrastructure away from rivers, and towards natural watersheds that limit the accumulation of floodwaters towards works or infrastructure. Implementation of the proposed stormwater management plan will help to mitigate residual flooding.

18.8.4 Surface Water Management

A conceptual stormwater management plan (SWMP) was developed, by which clean and dirty water-generating areas were first identified and then managed appropriately. Guidance applicable to the consideration of stormwater management has utilised IFC standards, given the absence of equivalent guidance in Guinea.

The geochemical assessment of the 111 samples tested across the PFS and DFS work programs concluded "*that only high-level controls will be required to manage the risk of acid generation in the waste rock. Additionally, it is recommended that waste dump leachate be contained and monitored to determine if it can be released into the environment.*"

Based on the above, the focus of stormwater management is sediment control (and not the management of leachate). Sediment control alone allows the consideration of lower design events than those outlined by the IFC guidance; however, for the purposes of the DFS, the IFC's minimum design event (1:25-year RI, 24-hour) has been adopted for the development of the SWMP. An allowance for climate change (an increase of 17.7% on design rainfall) has been included.

While not expressly defining catchment areas as dirty (given their potential to qualify as sediment control areas), the term 'dirty water areas' has nevertheless been used to align with IFC guidance and distinguish between areas requiring containment of runoff (dirty) and areas requiring routing around dirty areas (clean).

Areas requiring dirty water management include the pits (NEB, GBE, BC), the plant and associated infrastructure area, run of mine (ROM) pad and the WRDs (east of NEB pit and east of BC pit).

The plant and associated infrastructure area has been separately designed with internal bunding, to contain any process spillage and three sediment basins are included.

Beyond the process plant, three pollution control dams (PCD) are proposed to manage the dirty water areas on the site.

Clean trapezoidal channels without lining and dirty trapezoidal channels with lining have been conceptualised as routing runoff.

Runoff from haul roads and other access roads was not diverted to PCDs on the basis of the geochemical assessment results, and the challenge present in managing stormwater associated with linear infrastructure. Runoff from roads will be managed through localised removal of silt using typical road and drainage design. In addition, management approaches will include the clean-up of any spillage from these roads.

The proposed TSF, while identified as a dirty area, is designed as a zero-discharge system and considered separately to the surface water management design.

18.8.5 Site Water Balance

An overall operational water balance was developed based on the site information and the separate TSF water balance, which demonstrates the TSF as a zero-discharge system.

The water balance was developed as a static average annual model with three scenarios considered in the water balance. The key inputs are the pit/underground dewatering, peripheral borehole abstraction and runoff collection from WRDs and stockpiles. From this annual summary, the years with the highest, average and lowest volumes were extracted and used in the development of the three static average annual water balance models.

The water balance demonstrates a surplus of water, and as such, the groundwater abstractions (clean borehole water) will be diverted to the clean water pond for settlement of any sediment and reclaim for use where raw water is required, with excess discharged to the environment. The runoff from WRDs and stockpiles will be collected in pollution control dams for settlement of any sediment prior to discharge to the environment.

The pit/underground dewatering will be pumped to the dirty water pond for removal of any hydrocarbons and settlement and reused in mining operations where possible, however there will be a small surplus which will be discharged to the environment.

18.9 Water Supply and Site Water Management

Raw water supply to the Project will be via a combination of ground dewatering bores installed around the NEB and GBE pits as well two bores dedicated to water supply at the accommodation village.

Water from accommodation village bores, located adjacent to accommodation village infrastructure, will pump water directly to raw water tanks at the accommodation village. This water will then be treated via the accommodation village's water treatment plant.

Water from the 16 NEB and GBE dewatering bores will be pumped from bore headworks via above ground rising mains transferring water to a centralised HDPE lined clean water pond between the NEB and GBE pits.

Water required for potable use will be pumped from the clean water pond to raw water tanks at the water treatment plant which will be located adjacent the process plant. Given the primary source of water for processing will come from the TSF, there will be an excess of water supplied from this system. The excess water will be designed to overflow the clean water pond via a controlled discharge directly to the adjacent creek system.

A similar system will be installed at the BC pit whereby groundwater from dewatering bores will be pumped to a HDPE lined cleaned water pond. If required, water from this pond will be pumped to the water treatment plant or alternatively discharged to the adjacent creek system.

Mine dewatering, collected in pit sumps and underground sumps, will be pumped to surface from in mine dewatering systems and collected in a dirty water pond. This water will be used for dust suppression activities within the mining operation. When insufficient water is available for dust suppression from mine dewatering, make-up water will be pumped from the TSF water storage dam.

18.10 Mining Infrastructure

The bulk of the mining support infrastructure will be provided as part of the mining contracts by the open pit and underground mining contractors. The mining infrastructure required for the Project, including delineate of supply, includes:

- Earthworks, including hardstand, drainage, fencing and surface water management, installed by PDI.
- Facilities provided by the open pit and underground mining contractors, including:
  - Contractors offices.

- Contractors ablutions can changehouses.
- Contractors crib buildings.
- Heavy and light vehicle workshops.
- Warehouses.
- Construction of any go lines and maintenance bays in the area prepared by PDI.

- Heavy and light vehicle washdown bay and systems, including water cannons, hose reels, silt trap and oily water separator, installed by PDI for use by both open pit and underground mining contractors.
- High-flow and low-flow fuel dispensing bowsers installed by PDI for the use of both open pit and underground mining contractors.
- Services provided and installed to a common point for each mining contractor by PDI, including:
  - Power supply.
  - Potable water supply.
  - Wastewater collection and treatment in the main process plant wastewater treatment plant.
  - Communications via optical fibre.

18.11 Fuel Storage and Supply

Fuel storage for the Project will include bulk diesel storage, used in the mining and other mobile fleet, process plant heaters, regeneration kiln and power station, and HFO used for power generation. These fuels will be stored and distributed in two separate systems.

The diesel fuel storage facility will include up to 36, 71,720 L capacity self-bunded tanks connected in a master-slave arrangement for a total storage of up to approximately 2.6 ML, representing in excess of 30 days consumption. Since the diesel fuel storage facility is modular, the number tanks will be increased and decreased to match the long-term fuel consumption trends over the life of mine.

Fuel from the storage will be pumped to the process plant, power station and dispensed to the mining area heavy and light vehicle refuelling bowsers. The fuel storage will be filled from road tanker trucks at a dedicated diesel fuel unloading area which also incorporates a low-flow refuelling bowser for non-mining light vehicle use. Waste water from the diesel unloading area will report to the mining oily-water separator.

HFO will be stored in two, one million litre capacity tanks installed in a bunded area, equivalent to approximately 7 days storage, at peak power consumption, or 10 days storage with the solar PV operating at anticipated capacity. From the storage the HFO will be pumped to a day-tank in the power station for use. Heating of the HFO will be electrical tracing on piping and tanks. HFO will be filled from road tanker trucks at a dedicated HFO unloading area. Waste water from the HFO unloading and tank bunded area will report to the mining oily-water separator. The HFO storage will be designed, supplied and installed as part of the BOO power station contract.

18.12 Paste Plant

The paste plant is designed to take deslimed and filtered tailings from the processing plant, which will be delivered to the paste plant area and stockpiled by the tailings rehandling contractor, and produce a paste backfill suitable for use in the underground mine. The key design criteria for the paste plant is outlined in Table 18.2.

Table 18.2: Paste Plant Design Criteria

Parameter	Units	Value
Operating Hours (maximum)	h/y	6,238
Solids Feed Rate (nominal)	t/h	127
Binder Type		General Purpose Cement
Binder Addition Rate (nominal)	% Solids	7.7%
Binder Addition Rate (max)	% Solids	20%
Tonnes of Tails per Unit Volume of Paste	t/m^3	1.4
Peak Paste Usage	m^3/month	47,159
Average Paste Usage (Over Life of Underground Stoping)	m^3/month	30,741

18.13 Other

Other infrastructure, services and utilities required for the Project include:

- Surface water management infrastructure around the pits, WRDs, roads, process plant area and other infrastructure to divert clean water around the infrastructure and to collect any potentially contaminated water from inside the infrastructure areas and divert it into pollution control dams.
- Communications infrastructure consisting of connection to the national fibre optic backbone network for data and voice communication, UHF radio repeater tower and portable UHF radios.
- Fencing around the facilities, including:
  - Single 0.9 m high, fencing around the perimeter of the entire facilities encompassing the facilities, mine fly rock areas, and explosives magazine exclusion zone. The fencing also includes a patrol track around the perimeter.
  - Single 1.8 m high fencing around the process plant and administration area, explosives magazine, accommodation village and other remote infrastructure (such as the accommodation village's bores).
  - Double 1.8 m high fencing around the process plant controlled area.
- Security gatehouse at the main entrance to the site.
- Access control and security monitoring systems.

- Watse disposal facilities (landfill) for putrescible and non-recyclable waste, with recyclable waste to be sent off-site for recycling.

18.14 Conclusions

The Qualified Person considers that the infrastructure described is appropriately designed and specified to support the mining and processing operation previously outlined.

19 MARKET STUDIES AND CONTRACTS

19.1 Markets

PDI intends to enter into a contract for refining doré produced at Project. Under the intended contract, the refiner will arrange transport for the doré bars to its refinery with transfer of custody for the product taking place in the secure gold room located within the processing plant facility. The refiner will refine the gold to the purity accepted in international markets (generally 99.8% to 99.95%). The refined gold will be credited to the Company's gold account with its bank, to enable it to be sold on international markets or delivered into forward sales contracts. The refiner will invoice PDI for costs relating to transportation, insurance, refining and other charges.

Key terms for the transportation, insurance and refining include:

- Charges of US$8.45 per ounce of gold made up of:
  - Transport costs of US$4.60 per ounce.
  - Refining fee of US$2.50 per ounce.
  - Financing costs of $US1.35 per ounce.
- Payment of 95% of gold value on transfer of custody.
- Gold turnout expected within 7 to 9 days following transfer of custody resulting in final full payment.

As gold bullion is freely traded at prices that are widely known, a market study for the sale of gold doré was not undertaken. Given the freely traded nature of gold, the prospects for sale of any production are virtually assured.

19.2 Gold Price

The historical gold price over the last four years is shown in Figure 19.1 (The Perth Mint, 2025). This shows relatively stable prices from June 2021 to February 2024 with an average of US$1,870/oz, after which the gold price has increased relatively steadily to current prices, which have averaged US$3,284/oz over the month of May 2025.

Figure 19.1: Four Year Historical Gold Price

In order to forecast the long-term future gold price a consensus report was obtained from BMO Capital Markets (Chen, 2025). This forecast includes 34 individual forecasts from various sources. The median, low and high data from these individual forecasts over the period from 2025 to 2030 is presented in Figure 19.2.

The median of the long-term forecasts, also shown in Figure 19.2, is US$2,400/oz with the maximum of the long-term forecasts being US$3,477/oz and the minimum being US$1,740/oz.

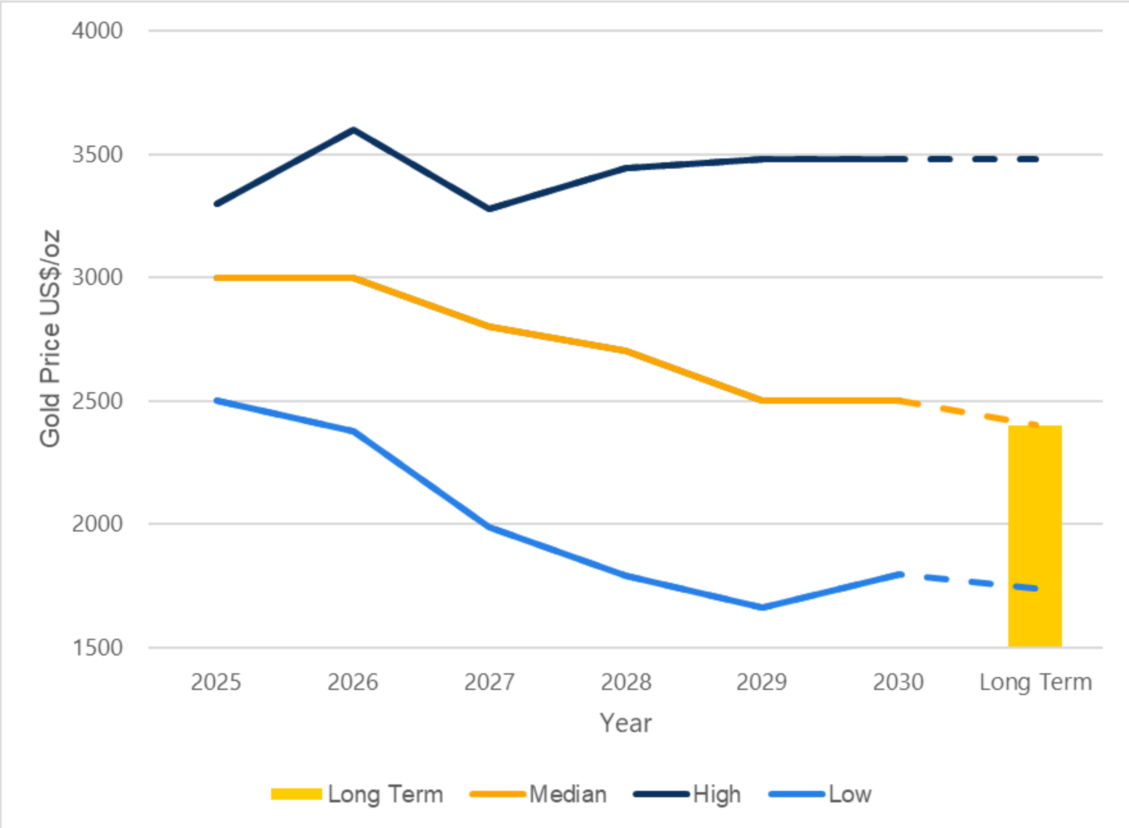

Figure 19.2: Consensus Gold Price Forecast

A gold price of US$1,800/oz has been used for cut-off calculations and the calculation of Mineral Resources and Mineral Reserves. The base case financial evaluation has been carried out at US$2,400/oz and the upside, "spot", financial evaluation has been carried out at US$3,300/oz.

The Qualified Persons consider that the pricing assumptions are reasonable given the current state of the gold market and forward looking forecasts.

19.3 Contracts

In addition to the contracts outlined in Section 24.1 relating to the delivery of the Project, PDI also proposes to enter into a number of contracts which will extend into the operational phase of the Project. These will include:

- Open pit mining services.
- Underground mining services.
- Explosives supply.
- Power purchase agreement.
- Analytical laboratory services.
- Fuel and lubricants supply.
- Catering and maintenance.
- Security services.

- Medical support services.
- Gold transport and refining.
- Various regent and consumable supply contracts.

None of these contracts have commenced being tendered, negotiated or finalised, however the Qualified Person believes that suitable contracts in line with industry standards will be able to be negotiated with suitable contractors and service providers.

20 ENVIRONMENTAL STUDIES, PERMITTING AND COMMUNITY IMPACT

20.1 Introduction

An environmental and social impact assessment (ESIA), including an environmental and social management and monitoring plan (ESMMP), was developed for the Project which aligns with the national laws and regulations, and international regulations. The ESIA (ERM, 2024), including baseline studies and identifying the potential risks and impacts which may occur due to the Project, was undertaken in March 2024 by Environmental Resource Management (ERM) and the environmental compliance certificate (the *Certificat de Conformité Environnementale*) was approved on the 17 January 2025 (CCE/00070). The Company is committed to complying with all relevant Guinean national laws and regulations, international standards, such as the International Financial Corporation's (IFC) performance standards on environmental and social sustainability (IFC-PS) (International Finance Corporation, 2012), and best practice standards in the environmental, health and safety (EHS) guidelines (International Finance Corporation, 2007) as well as human rights standards applicable to the project. The company has developed a corporate governance framework aimed at aligning with the principles and recommendations of the ASX Corporate Governance Council and the World Gold Council's responsible gold mining principles (World Gold Council, 2019). Based on the results of the ESIA and ESMMP report, the Company will develop an environmental management system, which will include an environmental policy statement.

20.2 Policies and Regulations

20.2.1 Corporate Values – Policy and Governance Commitments

The Company fosters a culture rooted in lawful, ethical and responsible conduct. It seeks to operate in line with its core corporate values and ensure directors, senior executives and employees uphold and reinforce these values. These core corporate values are entrenched in relevant policies, processes and procedures – established, under development and to be developed – across all aspects of the Project, *inter alia*, finance, procurement, anti-bribery and anti-corruption, diversity, human resources, environment, occupational health and safety, and corporate social responsibility.

Statement of Corporate Values:

- Our primary objective is to deliver maximum shareholder value whilst acting lawfully, ethically and responsibly.
- The Company will pursue operational and commercial excellence by using best practice approaches in our decision-making process focusing on continuous development, accountability and teamwork in all aspects of our business.
- In order to achieve these goals, we will ensure our employees and business partners have the appropriate skills and resources to perform their work effectively and efficiently and that all stakeholders (including investors, suppliers and regulators) are aware of the Company's values and our intention to uphold them. We will foster an open and supportive environment in all activities and relationships, and make sure that our senior executives demonstrate and reinforce our values in all aspects of our business and in all interactions with staff.

- We believe that our pursuit of these goals will cement a positive reputation for the Company in the community as a reliable, responsible and ethical organisation.

20.2.2 Environmental and Social Management System

The Company will implement an integrated environmental and social management system (ESMS) to provide a clear understanding of the expected standards and requirements for managing the Company's integrated management process. The ESMS will be developed with regards to international standards and practices such as:

- ISO 14001:2015 – Environmental Management Systems (International Organization for Standardization, 2015).
- ISO 26000:2010 – Social Responsibility (International Organization for Standardization, 2010).
- ISO 45001:2018 - Occupational Health and Safety Management (International Organization for Standardization, 2018).
- IFC Performance Standards Performance Standards on Environmental and Social Sustainability (International Finance Corporation, 2012).
- World Gold Council's Responsible Gold Mining Principles (RGMPs) (World Gold Council, 2019).

The ESMS will include all relevant requirements associated with the EHS, community and quality. The structure of the integrated ESMS is presented in Figure 20.1.

As part of the ESIA process, an ESMMP was developed which will form the basis of the ESMS. The ESMMP provides a framework for the environmental and social management of the Project as informed by the identified potential risks and impacts. The objectives of the ESMMP are aligned with the IFC-PS 1 approach to "*…manage environmental and social performance throughout the life of the project"* through the implementation of an ESMS. The ESMMP provides practical and effective mitigation measures to address these risks and impacts through minimising or reducing the negative impacts and enhancing the positive impacts during the construction, operational and closure and rehabilitation phases of the Project. To this end, the Company considers the ESMMP an essential tool, laying the initial foundation to establishing an ESMS during the next phases of the Project.

The key ESMMP objectives are:

- Identification of environmental, social and other relevant impacts, risks and opportunities.
- Mitigation of environmental, social and other relevant impacts, risks and opportunities.
- Implementing an integrated management system across the Project.
- Establishing and maintaining organisational capacity and competence.
- Proactive, transparent and continuous stakeholder identification and engagement throughout the Project's life.
- Implementing and maintaining robust audit and review mechanisms.
- Implementing and maintaining management of change processes.

- Defining and implement effective environmental and social monitoring programs as tools to continuously monitor and measure compliance.

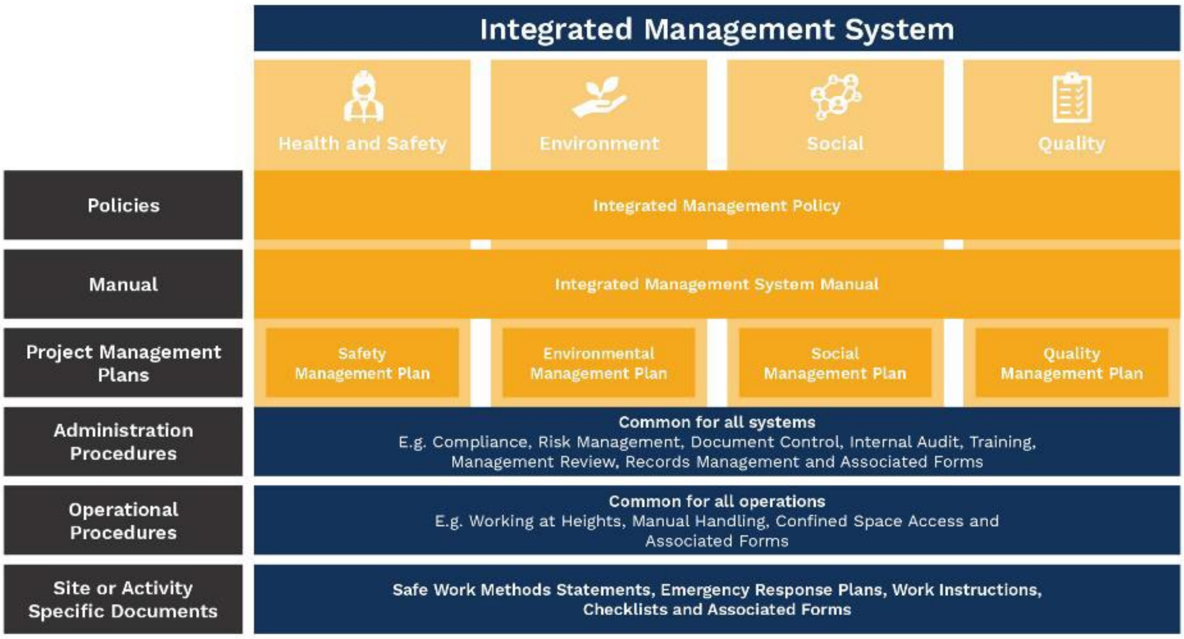

Figure 20.1: Integrated Management System (PDI 2024)

20.2.3 Statutory Regulations and Approvals

The ESIA was developed aligning to the applicable national regulations and international standards as outlined. Where Guinean and international guidelines differ, the Project adopted, and will continue to adopt, whichever is most stringent.

20.2.3.1 National Regulations

The decree D/2019/221/PRG/SGG of 4 July 2019 on the Environmental Code of the Republic of Guinea (the "Environmental Code") establishes the fundamental principles for promoting sustainable development and for managing and protecting the environment and natural capital. It establishes the administrative and legal framework enabling the Guinean state to deliver on its constitutional obligation to provide for a clean and healthy environment to every person in Guinea.

In accordance with Article 28 of the Environment Code, "*any project for the development or realisation of works or operations that may affect the environment is subject to a prior environmental and social impact assessment*", the contents and requirements of which are dictated by Article 24 of Arrêté A/2023/1595/MEDD/CAB/SGG Modifying Arrêté A/2022/1646/MEDD/CAB/SGG of 25 July 2022 on the Administrative Procedure for Environmental Assessments ("Order 1595").

Other relevant regulations and national standards applicable to the Project are:

- Mining Code (Law L/2011/006/CNT of September 2011, Amended by Law L/2013/053/CNT OF 8 April 2013).
- Water Code (Law L/ 94/ 005 /CTRN of 15 February 1994).

- Code for the Protection of Wildlife and Regulation of Hunting (Ordinary Law N° 2018/0049/AN of June 2018).
- Guinean Standards: NG 09-01-010:2012 / CNQ:2004 relating to new standards for wastewater discharges.
- Guinean Standards: NG 09-01-011:2012 / CNQ:2004 relating to new standards for air pollutant emissions.
- Guinean Standards: NG 09-01-012:2012 / CNQ:2004 relating to new standards on exposition to chemicals at work.
- Guinean Standards: NG 09-01-013:2012 / CNQ:2004 relating to new procedures for environmental inspection of industrial and commercial facilities.
- Guinean Standards on noise emissions.
- Local Government Code (1.12017/040/AN of 24 February 2017).
- The Labour Code (L/2014/072/CNT of 10 January 2014).
- Local Content Law (L/2022/0010/CNT on Local Content of 22 September 2022).
- Social Security Code (L/94/006/CTRN of 14 February 1994).
- Social Societies Law (L/2021/0017/AN of 30 April 2021).
- Law on the Protection, Conservation and Enhancement of National Cultural Heritage (L/2016/063/AN of 9 November 2016).
- Land and forestry legislation, including:
  - Land and Public Estate Code "*Code Foncier et Domanial*" (Ordonnance 0/92/019).
  - Urban Planning Code (Law L/98/017/98 of 13 July 1998).
  - Declaration of a Rural Land Tenure Policy (2001).
  - Forestry Code (Ordinary Law L/2017/ N°0038/AN of 24 April 2017).
  - Pastoral Code (Law L/95/51/CTRN of 29 August 1995).
  - Specific Laws and Regulations on Natural Protected Areas and Mining Activities (National Haute Niger).

20.2.3.2 Authorisations and Approvals

The Company has, or will, obtain all relevant environmental and social authorisations, licenses and sectorial permits required by Guinean law to undertake Project activities. The permits already obtained or applied for by the Company include:

- *Permis de Recherhce Industrielle (Or) (*exploration permits) are in place for ongoing exploration work outside of the Project Area, these were obtained from October 2018 through to September 2020 from the *Ministère de l'Environnement et du Développement Durable* (Ministry of Environment and Sustainable Development) (MEDD). These permits are currently under application for renewal and Article 78 of the Mining Code allows for these permits to be extended automatically until the date of renewal following until granted or denied.

- *Permis d'Exploitation* (exploitation permit or mining licence) to undertake exploitation activities within the Project Area, with the application submitted and under review.

The additional permits which will be required for the Project and will be obtained by the Company include:

- Electricity Generation Licence from *Autorité de Régulation de l'Électricité en Guinée* (AREG).
- *Autorisation de Prélèvement d'Eau* (water abstraction permit) required if water is drawn from a natural source.
- *Autorisation de Prélèvement d'Eau Souterraine* (groundwater abstraction authorisation*)* from the MEDD, in coordination with the Ministry of Energy, Hydraulics and Hydrocarbons.
- *Autorisation de défrichement* (vegetation Clearing) permit authorised by MEDD as a condition for approving the ESIA.
- *Autorisation d'Installation Classée pour la Protection de l'Environnement (ICPE)* (environmental authorisation for classified facilities) required under the Environmental Code for:
  - Waste/landfill facility and to manage solid and hazardous waste (including tailings).
  - Effluent treatment facility and to discharge effluent.
  - Water treatment facility and building permit for a sewage system.
  - Water transport pipeline.
- *Autorisation d'Occupation du Domaine Public ou Privé* (land use authorisation) and an *Accord de Développement Local* (ADL) (land development agreement).
- Authorisation to build fuel storage facilities within the ICPE permit which covers the storage of diesel, gasoline and other hydrocarbons.
- *Autorisation de Construire* (construction authorisation) within the Urban Planning Code and Environmental Code, which also includes the *Autorisation d'Occupation du Sol*. This is required for the mine village and other infrastructure.
- *Autorisation de Construction et d'Exploitation de Lignes Électrique* (authorisation to construct power transmission lines within the lease boundary) from *MEDD.*
- *Autorisation de Détention et d'Utilisation d'Explosifs* (explosive storage and use authorisation) from the Ministry of Mines and Ministry of Security and Civil Protection. This is also governed by the Mining Code and the National Security Regulations.

20.2.4 International Guidelines

The Project will align with international standards and best practices including:

- IFC-PS on Environmental and Social Sustainability (International Finance Corporation, 2012).
- World Bank Group (WBG) EHS Guidelines: General (International Finance Corporation, 2007).

- International Council on Mining and Metals (ICMM) Performance Expectations (International Council on Mining and Metals, 2022).
- International Cyanide Management Code for the Manufacture, Transport and Use of Cyanide in the Production of Gold (International Cyanide Management Institute, 2005).
- Other applicable good international industry practices (GIIP).
- Other international treaties, agreements or conventions relevant to the Project.

20.3 Environmental and Social Risks

As part of the ESIA process, various baseline environmental and social studies were undertaken, during the wet and dry seasons where required, to characterise the conditions of the Project areas. These baseline studies were used to determine the potential risks and impacts of the Project. The impacts were assessed before mitigation/management measures and assessed with the implementation of these measures.

20.3.1 Environmental and Social Baseline Studies

The existing physical, biological and social conditions of the area were assessed and classified as an area of influence (AoI) from the Projects infrastructure and are summarised in this section, focusing on the resource to receptors that may be impacted by the Project. The baseline studies for the ESIA were conducted through a desktop review of publicly available information, data collected by the Company and Australian Groundwater and Environmental Consultants (AGE), and ERM in May to December 2023.

The following environmental and social studies were undertaken across 2021 to 2024 to determine the existing baseline conditions, and the results are summarised in Table 20.1:

- Ambient air quality.
- Noise.
- Surface water.
- Groundwater.
- Soils, soil quality and geology.
- Biodiversity and ecosystem services.
- Socio-economic.
- Cultural heritage.
- Landscape and visual.
- Traffic and transport.
- Human rights assessment.

Table 20.1: Summary of the Environmental and Social Baseline of the Project Area

Receptor	Description	Standards/Guidelines
Topography	The Project's AoI is characterised mostly by hills and vast plains. The elevation ranges from 370 – 436 m above sea level. Valley slopes are generally gentle, with the remaining areas being flat, except to the west of the Project, where slopes are steeper and hills more pronounced.	NA
Soils, Soil Quality and Geology	The site consists of highly degraded wooded savannah, bare soil, shrubland and grasslands with four small villages on the periphery and traditional artisanal gold mining activities in the area. Cropland does not form the majority of the land cover however it is dispersed throughout the Project area. Artisanal mining and deforestation are likely to have already impacted soils from their natural state. The soils observed in the Project area were either loamy gravels or to a lesser extent gravelly sandy clays. Chemical sampling indicated no evidence of pollution from organic industrial, or commercial wastes, but heavy organic compounds were detected at sampled locations. In terms of agricultural quality, the soils are highly leached of nutrients, such as nitrate, phosphate, and most trace metals, although there is some variability, and some areas are more fertile than others.	NA
Climate and General Meteorology	During the dry season (November to April), the Siguiri basin is exposed to dry dust-laden continental winds (the Harmattan). These winds raise large amounts of dust into the atmosphere which can be carried for hundreds of kilometres before settling. The dominant wind direction is southwest, influencing emissions from the mine towards the northwest, Kouroussa Town. The mean annual precipitation (MAP), with wet seasons May to October, for the Bankan region between 1950 and 2020 was 1493 mm.	NA

Receptor	Description	Standards/Guidelines
Ambient Air Quality	SO_2 and NO_2 did not exceed the air quality standards. Dust deposition exceeded at six locations out of 30, indicating an exceedance of more than 50% of the standard at all locations. PM_{10} exceeded the annual IFC standards at three locations and the Guinean at one location over three-months. $PM_{2.5}$ exceeded the annual IFC and Guinean standards at one location. These exceedances of dust and PM_{10} from the mine will vary depending on the weather meteorological conditions and type of activity. Dust deposition during the dry season is more likely particularly during the Harmattan.	Guinean Standards *"Norme Guineene NG 09-01-011 :2012/CNQ :2004"* IFC (2007) General EHS Guidelines for Air Emissions and Ambient Air Quality.
Ambient Noise	The existing noise levels exceeded the IFC and Guinean's criteria for the daytime noise (07:00-22:00) in residential areas at seven locations. Exceedances were also observed for the nighttime noise (22:00-7:00) criteria at all monitoring locations. The likely noise sources are attributed to the nearby villages and National Roads (N1 and N31.7) at four locations and possible surrounding anthropogenic activities.	Guinea Standard NG 09-01-012:2012 / CNQ:200410 *"Limites Maximales d'exposition a Quelques Produits Chimiques et au Bruit Dans Les Lieux de Travail"* IFC and EHS Guidelines on Environmental Noise Management (Section 1.7)
Surface Water	The samples collected (upstream and downstream) within the surrounding watercourses in the study area failed to meet the IFC's and WHO standards for water quality. Surface water samples collected in the perennial Niger River showed a general decrease in water quality from December 2022- November 2023. The highest concentrations of heavy metals were during May 2023 Key parameters that failed to meet the IFC's, WHO's, and/or Guinean standards were pH, heavy metals (i.e arsenic, lead and mercury), oil and grease, total coliforms, and faecal coliforms. High observed faecal readings in the early wet season could be explained by a 'flushing' of pollutants into streams from the first rains.	World Health Organisation (WHO) Guidelines for Drinking Water Quality (2022). IFC Guidelines for Mining Effluent (2007). Guinea's Discharge of Wastewater Standards (GDWS) (2012).

Receptor	Description	Standards/Guidelines
Floodline Assessment	The Project is located on the north bank of the Niger River (1.2 km away) and the BC pit transversing the eastern portion of the Bankan Creek. A second tributary, the Komonida River drains northeast along the NEB pit area and enters the northwest of the tailings storage facility (TSF). The flood risk assessment for a 1 in 100-year event along the Bankan Creek and backflow from Niger river shows flood depths could affect the southern boundary of the BC pit, with flood depths reaching 2.55 m (maximum for the current scenario). Possible flooding from the Bankan Creek diversion around the project infrastructure, a flood peak of 149 m^3/s was estimated for the 1 in 100-year, 24-hour storm event. Water flow of smaller rivers in the Project's AoI may dry up during the dry season while valleys and plateaus can flood during the wet season.	NA
Groundwater	A total of 23 boreholes were identified during a hydrocensus. The groundwater depth ranged between 2.97 and 12.81 meters below ground level (mbgl) representing that the groundwater levels are representative of one homogenous aquifer. In this region, recharge to the aquifers takes place from rainfall or from streams. There are no artificial recharge activities in the region. There is a 98.5% correlation between the groundwater elevation and topographical elevation and flows from the high lying areas towards the low-lying Niger River. The water quality in March-June 2023 was generally good, except for exceedances in pH level, metals (sulphate, iron, manganese, arsenic, zinc and aluminium). The results indicated a calcium-magnesium-bicarbonate signature indicating recently recharged groundwater. Aluminium and iron exceedances were in the areas of the mine site's ore body and in surrounding villages.	WHO drinking water quality guidelines. Guinean wastewater discharge quality guidelines.
Geochemistry	Waste rock was classified as non-acid-forming (NAF) while material from the volcanic materials of Bankan Creek was classified as potentially acid-forming (PAF). Leachate anticipated is likely to comply with IFC guidelines, except high levels of copper concentration from the Bankan Creek. Sulphates will likely be stored in the waste rock dumps (WRD) and TSF, and the arsenic from the WRD and ore to the TSF are likely elements of concern.	IFC Mining Effluent Guidelines (2007).

Receptor	Description	Standards/Guidelines
Landscape values and visual amenity	It is assumed that the largest horizontal component of the project that will cause visual impacts is the waste-storage area, which would be a maximum of 2 km wide. Calculations suggest that the impact of a 2 km area would reduce to insignificant at a distance of about 45.8 km, though it is reasonable to assume that it will be hidden beyond 5km due to topography, vegetation and non-Project buildings. The highest vertical components are the processing plant and waste-storage area which would be a maximum of 40 m tall which would be considered significant up to a distance of 4.6 km.	NA

Receptor	Description	Standards/Guidelines
Biodiversity and ecosystem services	Field data was collected during 2022 to 2024 to understand the important biodiversity values in the study area. This included ecological assessment (April 2022), a wet season survey (November 2022–February 2023), a dry season surveys (March 2023 and January 2024), Ecosystem services and bushmeat surveys (February 2023). A chimpanzee survey was undertaken during the wet season in August- November 2023). The Project's area is in a mosaic of evergreen forest, savannah and grassland habitats. Most habitats are highly fragmented and degraded in this region, especially in areas of high human population density. 39% of habitats in the Project's area of influence consists of crops and fallows non-flooded land, largely attributed to forest fires, timber and firewood harvest, agriculture, gold panning and sand mining. The area has been locally deforested, altering the biodiversity of the vegetation, watercourses and habitats. 35% (19.17 km^2) of the AoI consists of natural wooded savannah and trees. A total of 281 floral species were identified during wet and dry season surveys; zero are Critically Endangered (CR) (0%), two are Endangered (EN) (0%), eight are vulnerable (VU) (3%) and five are near threatened (NT) (2%). A total of 344 faunal species were identified (2022 to 2024), 139 birds, 91 freshwater fish, 33 reptiles, 32 amphibians, 18 bats and 49 mammals. The significant fauna species of conservation concern (SCC), include the giant ground pangolin (EN), the white bellied pangolin (EN), common hippopotamus (VU) and the leopard (VU) as well as the hooded vulture (CR) and two ball python (NT) among other SCC. Western chimpanzees are CR but have significant populations in Guinea. During the wet season of 2023, 80 chimpanzee nests were recorded in the Moussava area. Most of these were just outside of the Project's AoI, in gallery forest. No observations of chimpanzees in the Project's AoI and surrounding area were recorded in the dry season, suggesting that the chimpanzees may inhabit this area seasonally between July and November. It is likely that chimpanzees move further west/southwest towards the Mafou Classified Forest, a core area of HNNP, during the dry season to avoid heightened human activity. The project is located within 20 km of one national park, two classified forests, one key biodiversity area (KBA), important bird areas (IBAs), two Ramsar sites and one biosphere reserve. Areas containing SCC should be considered as critical habitats.	Forest Code (Ordinary Law L/2017/ N°0038/AN of 24 April 2017). Code for the Protection of Wildlife and Regulation of Hunting (Ordinary Law No 2018/0049/AN of June 2018). The Framework Law on Freshwater Fishing (*Loi-cadre sur les activités de la pêche en eau douce;* L/96/067/AN of 22 July 1996). Wildlife order on protecting wild species of fauna and flora, Republic of Guinea (Arrete A/2020/1591/MEEF/CAB/SGG). IFC-PS 6; Biodiversity Conservation and Sustainable Management of Living Natural Resources

Receptor	Description	Standards/Guidelines
Ecosystem Services	Of the households surveyed, 72% reported using boreholes as the predominant water sources and 5% used wells and/or water towers. Hunting within the Kouroussa area is socially and historically linked to the status if Donso hunting and used for bush meat. Logging for firewood and timber, beekeeping for honey, harvesting of wild fish, use of the area for livestock and crops are among other ecosystem services in the Project area.	
Population and demographics	The Projects AoI was set using a 5 km buffer including ten settlements which were surveyed, Farakoun, Wouloukin, Souloukoudo, Tambiko, Bankan, Sokoro, Kignédouba, Diaragbéla, Menindji and Banako. The Project's AoI is situated in Kankan Administrative Region, specifically in Kouroussa Prefecture. The Kouroussa Prefecture has a population of 23 habitants/km^2, the population of the ten settlements was 64,375 with an average household size of 7.2 members which is lower than the Guinean average and Kankan's regional average (10.6). The household distribution of the social AoI is 50.1% male and 49.9% female. The main ethnicity in the social AoI is Malinké, an ethnicity commonly found in the Upper Guinea Region. Of the population in the social AoI, 99.5% speak Malinké, the primary language in Kouroussa Prefecture. French is spoken by 29% of the population, followed by Soussou (5%) and Pular (4%).	Administrative Procedure for Environmental Assessments (Arrêté A/2022/1646/MEDD/CAB/SGG) IFC-PS 4 – Community Health and Safety. IFC-PS 5 – Land Acquisition and Involuntary Resettlement.

Receptor	Description	Standards/Guidelines
Livelihoods	There is a significant seasonal immigration of young people in the social AoI linked to artisanal small-scale mining (ASM), known locally as artisanal gold panning (AGP). Competition has become locally significant for gold-mining areas between artisanal miners and industrial companies in recent years due to the arrival of three industrial mining/exploration projects on the outskirts of Kouroussa town: Kouroussa Gold Mine (KGM) to the northeast, Sycamore to the south and Predictive Discovery to the southwest. In the social AoI, AGP represents the main income source in 32% of households in Kouroussa town and 52% of households in rural areas. In the social AoI, 79% of households have at least one member who engages in AGP (47% in Kouroussa town and 100% in rural areas). Agriculture and gardening are the most important sector of activity in Kouroussa Prefecture, it accounts for 57.0% of rural households income and employment for 52% of the workforce. The soils of Kouroussa Prefecture are rich and diversified, making it possible to cultivate many crops and obtain good yields. Villages far from the Niger River, such as Kignédouba or Farakoun, fish in the river's tributaries. 61% of households reported their involvement in some form of forestry or use of forest products. Of the households, 12% reported harvesting honey.	
Education, health and infrastructure	The Kankan Administrative Region is the least literate in the country both among young people (25.8%) and adults (18.2%). It displays the national trend of lower literacy rates in rural areas compared to urban areas, as well as women having lower literacy rates than men. 76% of residents in Kankan Administrative Region aged 3 and over have never attended school, compared to the national average of 62%. The region has the country's highest number of school dropouts between the ages of 10-24. Only 7% of the population over the age of 15 in the social AoI has undertaken vocational training. The most widely attended vocational training sectors are liberal professions, teaching, medicine, mechanics, welders and other laborers and construction trades. In the social AoI, most households prefer to use personal transportation with 94%, across both rural and urban areas driving motorcycles. In total 95% of households in Kouroussa town use motorcycles daily and 57.0% of households in rural areas, indicating a greater need for or availability of mobility in urban areas. 80.8% of households in Kankan Administrative Region had access to drinking water; however, as little as 3.7% of households had a faucet. The predominant water source for households is boreholes with pumps, reported by 72% of the surveyed households. A smaller proportion, around 18.5%, use improved wells, while 5.5% rely on pumped wells	

Receptor	Description	Standards/Guidelines
Public infrastructure and utilities	There are 17 primary schools in Kouroussa town including public, private and Franco-Arabic institutions. There is no educational infrastructure in other villages in the social AoI, which forces students to make expensive and complicated trips, resulting in lower enrolment. Kouroussa town has one public and one private middle school, as well as one public and one private high school. The distribution of healthcare facilities in Kouroussa Prefecture is uneven, with most facilities located in Kouroussa town and thus not easily accessible for rural parts of the social AoI. Only 14% would choose Kouroussa's hospital as their primary healthcare facility. Instead, most rural households surveyed (68%) rely on health posts closer to their village. Guinea suffers from major deficits in energy production and electricity. Access to energy is much more limited in rural areas compared to Kouroussa town, with only 24% of households lacking access to electricity in Kouroussa town compared to 59% in rural areas.	
Cultural Heritage	The baseline study identified a total of 21 non-designated cultural heritage resources within the AoI (1,000 m from the Project infrastructure). Eight cultural heritage sites identified will be directly impacted by the Project footprint. Six cultural heritage sites are considered as high sensitivity, seven are medium sensitivity and six are low sensitivity. In other cultural heritage sites with religious or cultural value in the form of sacred trees, groves, ponds, rocks, as well as spirit dwellings (Insuco socioeconomic baseline study, September 2022) which require conservation for the wellbeing of the community. Artisanal mining is associated with people's connection to land and has been carried out in the AoI for approximately 400 years by nomadic Bambara gold miners from Mali across northwest Guinea.	IFC-PS8 – Cultural Heritage

20.3.1.1 Climate Change and Greenhouse Gas Emissions

An assessment of the potential greenhouse gas (GHG) emissions was undertaken for the Project. The assessment involved quantifying the likely GHG emissions from the hybrid power plant, process plant, open pit and underground vehicles and equipment and combining that with those generated by vegetation clearance activities and use of explosives. The emissions were quantified using Scope 1 (emissions from the mines sources and directly controls), Scope 2 (indirect, such as electricity) and Scope 3 (mines overall carbon footprint through its lifecycle across the value chain such as the fuel used).

The Project, as reviewed as part of the ESIA, is likely to emit a total of 3.91 million tonnes per carbon dioxide equivalent (tCO_2e) over its lifetime of 12 years. The average annual emissions over the 12 years are estimated to be 326,000 tCO_2e. The sources of emissions are the use of explosives (1,440 tCO2e/year), vehicles and equipment (28,500 tCO2e to 252,000 tCO2e a year), the processing plant (47,400 tCO2e over the 12 years). The hybrid power plant which will be used, incorporating solar PV, will save 28.6% of emissions.

Final emissions and savings estimates have not been updated as part of the DFS and will only be re-estimated during detailed design.

20.3.2 Risk Assessment Framework

The impact assessment included several steps to collectively evaluate how the Project will interact with the elements of the physical, biological and social environments to produce impacts on resources and surrounding receptors. The methodology used during the ESIA process followed the international best practice approach illustrated below in Figure 20.2.

The applied methodology considered the specific issues typically related to the development of a gold mining project and associated infrastructure, in order to present a mechanism for identifying and assessing impacts specific to this type of development, and to achieve a targeted and refined evaluation.

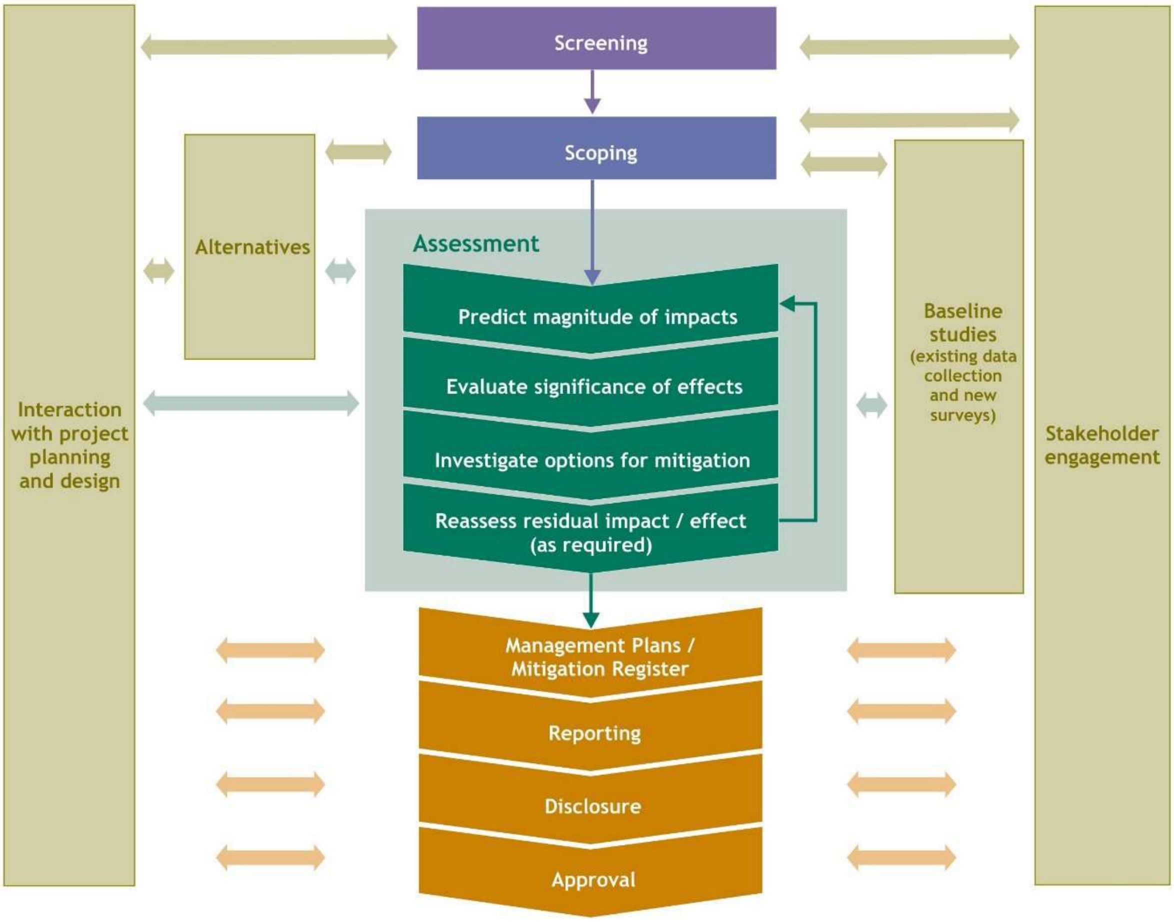

Figure 20.2: ESIA Process (PDI 2024)

20.3.3 Key Identified Risks

The assessment of the potential environmental and social risks and impacts attributable to the phases of the Project included qualitative and quantitative (where relevant) assessments. The significance of each potential impact was identified, and mitigation measures to minimise and reduce the impacts were recommended. Cumulative impacts, particularly on communities' health and safety and on biodiversity, were also assessed.

The risks and impacts were quantified and classified as having a negligible, minor, moderate or major significance. The complete impact assessment is available in Volume 3 of the ESIA (2024). The main significant negative environmental and social risks and impacts that were identified through the ESIA process are as follows:

- Air Quality – Mining activities could result in the deterioration of ambient air quality due to the emission of pollutants and dust deposition. This impact was assessed to have a major significance.
- Surface Water – Infrastructure such as the BC pit is as risk of flooding during 1-in-100 year events due to it intersecting the Bankan Creek and its proximately to the Niger river. This impact is assessed to have a moderate significance in all phases.

- Surface Water – Surface water quality is likely to be impacted particularly at the Bankan Creek, Bankan Village Stream and Bankan NE Creek (headwater tributaries of Kolomida River). The seasonal flow patterns/hydrological alteration of these creeks will be altered and impacted due to the presence of mine infrastructure, and possible discharge of excess mine water during the dry season. These impacts are considered to be a major risk during the dry season and minor risk during the wet season.
- Groundwater – Groundwater levels and/or water quality will be impacted due to dewatering and seepage/leachate from the TSF, WRDs, ore body and mine pits through acid rock drainage and metal leaching resulting in a contamination plume. This impact is considered as having a moderate significance in all phases of the Project.
- Biodiversity – Loss of habitats due to the establishment of the project, including natural habitats such as the gallery forests, bowal, woodland savannahs, wetlands and watercourses habitats which are considered as threatened and retain SCC such as hippopotamus, slender-snouted crocodile, vultures, amphibians and freshwater fish. This impact is considered to have a moderate to major significance from throughout all phases of the Project from pre-construction to closure.
- Biodiversity – Several nest of Western chimpanzees (which are considered as critically endangered) were identified within the Project AoI and will be negatively impacted through the loss of habitat, habitat fragmentation, loss of food security, disturbance from mining activities and increased human-wildlife conflict due to the population influx. This impact is considered as having moderate to major significance throughout all the phases of the Project.
- Biodiversity – The Project will potentially impact the surrounding legally protected and nationally recognised Haut Niger National Park, Tamba Classified Forest and L'Amana Classified Forest. Potential impacts include habitat loss and fragmentation, disturbance, hydrological impacts from change of flow and drainage due to infrastructure, and wildlife mortality from increased road collisions. These impacts are considered as having a moderately significance throughout all phases of the project.
- Landscape and visual - Alteration of the baseline landscape and visual intrusion from the infrastructure will cause negative visual impacts to the surrounding communities. These impacts are considered as having a be moderate significance during construction, however, will be major during the operational phase and minor during closure.
- Traffic and Transport – The projects activities will impact the surrounding road function, potentially degrading the roads (paved and unpaved) and increasing risks of traffic accidents. This impact is considered as having minor to major significance throughout all phases of the Project.
- Social – Use of child labour or use of people ages 16-18 in hazardous work within the supply chain remains a possibility. If there are incidences of child labour, the magnitude of the effect to the individual affected will remain unchanged regardless of phase of the Project. This impact is considered as having a major significance in all phases of the Project.

- Social - Impacts associated with loss of land and livelihoods during pre-construction will result in economic displacement. These impacts are considered as having a major significance.
- Social – Surrounding communities expressed a level of expectations from the Project such as employment, business opportunities, compensation packages, infrastructure development and livelihood support among other benefits. If these expectations are unmet this will result in upset and tension from the community towards the Company. This impact is considered as major impacts in all phases of the Project.
- Social – Impacts to community health and safety, include road safety, nuisance impacts (noise, air quality, dust, vibration,), trespassing and increased spread of diseases. These impacts are considered as having a moderate to major significance impacts in all phases of the project.
- Ecosystem Services – The loss of ecosystem services for the communities due to the establishment of the Project, will include loss of access to the services, modification and fragmentation of habitats, water and soil contamination, introduction of alien invasive plant species, loss or pollution of wetlands. This impact is considered as having a major significance in all phases of the Project.
- Cultural Heritage – Direct impact to cultural heritage sites identified in the Project infrastructure footprint and indirect impacts (loss of access, intrusive visual, auditory or dust elements) to site which are within the Project's social AoI. Chance finds of unknown cultural heritage sites during construction and operation which will lead to the site's destruction or damage. This impact is considered as having a major significance in the pre-construction and construction phase.

20.4 Health, Safety, Environmental and Social Management Plans

The management plans incorporate measures and procedures for the short- and long-term health and safety, environmental and social management. The management and monitoring plans were developed as a tool to be used by the Project throughout construction, operation and closure and rehabilitation. An integrated ESMS is being developed as part of the Project to guide design and manage construction, operation and the closure and rehabilitation phases.

20.4.1 Community Health and Safety Management Plan

The Project may increase risks and impacts for neighbouring communities, which may experience negative impacts on their natural resources, health, and overall infrastructure, it is imperative to develop and implement a specific community health and safety plan, which will aim to:

- Avoid or minimise risks and impacts on the health and safety of the local community during all phases of the Project; and
- Protect personnel and property in a legitimate manner that avoids or minimises risks to the community's safety and security.

The following summarised management measures will be implemented within the community health and safety plan:

- Community road safety awareness campaigns will be conducted especially in communities where heavy mobile equipment will be most active. Awareness training should be repeated in villages as construction extends into their territory. The campaign should also be deployed in schools to raise awareness among children, who often travel long distances along roads to get to school.
- In the event of an accident involving Project activities, where a community member is injured, the Project will arrange transportation of the injured person to an appropriate health centre capable of treating the injuries and facilitating access to medical treatment.
- Accident reporting and investigation procedures will be developed to determine root causes and identify corrective measures to reduce the risk of recurrence.
- Road safety features are incorporated into the design, including speed restrictions in more populated areas, signage warning drivers of population centres, livestock, schools, etc., speed bumps at the entrance to populated areas, and bus stops/areas where vehicles can park, etc.
- The Project will consult with other organisations, including the police, to ensure they are informed about how the Project will use public roads.
- Monitoring will be conducted during the construction, operational, and closure phases to evaluate the mitigation measures and residual impacts on the health, safety, and security of the population.
- All direct employees and subcontractors will undergo an induction training that includes the Company policies and project procedures

20.4.2 Occupational Health and Safety Management Plan

The Company acknowledges that mining is a high-risk industry, creating a complex working environment that required rigorous health and safety procedures. Failure to manage these high-risk environments can result in injuries, loss of life, or property destruction. An occupational health and safety management plan (OHSMP) has been developed and aims to implement a structured approach to occupational health and safety to achieve a high and consistent level of safety performance.

The following management measures have been developed in the OHSMP:

- The OHSMP applies to all the Company's projects, including all employees, subcontractors, and visitors associated with operations.
- The OHSMP will be based on the Plan-Do-Check-Act (PDCA) cycle principle.
- Within the OHSMP is a health, safety and well-being policy where the Company sets out commitments to take full responsibility for the health, safety and well-being of its personnel through conducting regular audits, establishing engagement platforms, promoting well-being and healthy lifestyles and comply with applicable laws and regulations etc.
- The Company uses various processes to identify safety risks related to its activities. This includes recognizing common industrial risks, which the OHSMP largely focuses on, and continuously evaluating emerging risks.

20.4.2.1 *Emergency Preparedness and Response Plan*

The emergency response framework is designed to ensure that the Company can effectively manage emergencies and minimise their impact on personnel, operations, and the environment. The Company is committed to ensuring the safety and well-being of its personnel and recognizes the potential challenges and risks associated with operating in high-risk industries and remote, isolated locations. While all efforts are made to prevent emergency situations, it is acknowledged that there may be cases where preventive measures fail, or an unforeseen event occurs. In such cases, it is essential to have the necessary resources and capabilities to protect personnel, assets, and the environment by mitigating the consequences of the incident.

The Project will be equipped with all required and relevant emergency response means, such as:

- Fire equipment (cameras and sensors, extinguishers, fire hoses, automatic sprinklers, fire blankets, and a fire smoke detectors, fire truck), first aid kits, cyanide emergency response kits, spill kits, medical facility, first-aid centre, and ambulance.
- Trained workforce (general emergency response training and area-specific training).
- The Project will consult with other organizations, including the police and wildlife authorities.
- Hazardous materials storage will be in line with all regulations and international best practices and the Project will develop a detailed hazardous material response plan.
- Establish a first aid centre staffed with trained personnel equipped with a fire truck, an ambulance, sufficient first aid supplies, a first aid kit, and medications.
- Establish a clinic staffed with a medical doctor and trained nursing staff and equipped appropriately to manage any reasonable risks.
- Develop and communicate clear evacuation plans in high-risk areas, indicating designated evacuation routes and emergency shelters.
- Develop an emergency response team and training regime to ensure that any emergency can be adequately addressed.

20.4.3 Environmental Management Plan

The environmental management/mitigation measures were informed by the identified environmental risks and impacts. The ESMMP and the integrated ESMS were developed and will be implemented during all phases of the Project. The ESMMP will provide the basis for compliance management, operational procedures and practices to be defined and implemented across all project activities undertaken by the Company, its contractors and suppliers. The following management plans will be developed as part of the ESMS and are detailed in the ESIA (2024):

- Air quality and GHG management plan.
- Dust management plan.
- Noise and vibration management plan.
- Biodiversity management plan (BMP) and biodiversity action plan (BAP).

- Sustainable Water management plan.
- Stormwater Erosion and sedimentation control management plan.
- Waste management plan (including general waste and hazardous waste).
- Traffic management plan (TMP).
- Socio-economic management plan.
- Cultural heritage management plan (CHMP).
- Emergency prevention, preparedness and response plan.

The sections below summarise the mitigation measures to address the most significant impacts identified during the ESIA.

20.4.3.1 Air Quality and GHG Emissions Management

Table 20.2 presents the key management/ mitigation measures which form part of the ESMMP for air quality and GHG as it was identified as a significant impact.

Table 20.2: Key Management Measures for Air Quality and GHG

Impact	Management Measures	Key Managing Documentation
Dust Management	Develop and implement an Air Quality Management Plan (AQMP), which will include dust management measures and measures to control other air emissions. As far as practicable, plan site layout so that machinery and dust causing activities are located away from receptors. Where practicable and safe, use dust generating equipment fitted or in conjunction with suitable dust suppression techniques such as water sprays or local extraction, e.g. suitable local exhaust ventilation systems. Traffic speed control: limit traffic speed on un-paved roads, haul roads and restrict off-road travel and define Project routes and transport procedures to avoid dust emissions in sensitive areas. Use suitable stabilising materials where it is not possible to re-vegetate or cover with topsoil, as soon as practicable to stabilise surfaces. Impose and signpost maximum speed limits for project light and heavy vehicles and mobile equipment on surfaced and local roads, unsurfaced haul roads, community roads. Specific fugitive emissions suppression programs (operational procedure level) will be developed to limit dust emissions from Project activities such as earth movement during construction, mining and blasting, minerals processing, transport and loading and traffic in general.	Air Quality Management Plan (AQMP) Traffic Management Plan (TMP) / Site Audits Sustainability Report to reduce climate change emissions
GHG Emissions	Systematic substitution principal implementation - Use or replace where practicable fuels and refrigerants with lower global warming potential (GWP) types. Energy efficiency enhancement - Consider/introduce technology appropriate to local conditions for engines, vehicles, exhaust filtration systems, etc. Efficient waste management - Enhance recycling and reuse to reduce and avoid incineration where practicable. Transport efficiency - Avoid unnecessary transport and use low carbon intensive transportation, develop a travel and transport policy with the objective to reduce GHG emissions.	

Impact	Management Measures	Key Managing Documentation
	Continuous awareness training and good practices implementation to limit energy consumptions and building cooling by Project employees at all activities levels. Monitor and manage emissions by: – Continuous air quality monitoring at emission points and receptors. – Adapt and/or update procedures, as necessary. – Define resources and responsibilities to implement these procedures and implement necessary changes based on monitoring results. – Implement a Grievance Mechanism allowing for stakeholders to file air quality related grievances and enabling the Project to identify problematic areas and seek resolution.	

20.4.3.2 Water Management

Table 20.3 presents the key management/ mitigation measures which form part of the ESMMP for the Sustainable Water Supply Management Plan and the Stormwater Erosion and Sediment Control Management Plan as they were identified as a significant impact. A sustainable water supply management plan is required to mitigate the negative surface water and groundwater impacts that are expected to downstream users, such as hydrological alternations to watercourses, ensuring any discharge of water meets the effluent standards, and ensuring the groundwater drawdown does to surrounding rivers/creeks and other users is significantly reduced.

A preliminary static annual water balance was completed to evaluate the expected mining operation water use on an annual average basis, which will be updated pre-construction.

Table 20.3: Key Management Measures for Water Management

Impacts	Management Measures	Key Managing Documentation
Impacts to groundwater quality water quality, seepage/leachate from PAF material resulting in a contamination plume.	Where practicable, create an impermeable base and walls to contain any PAF rock identified, internal of the WRDs main landform. This will reduce the impact of poor-quality leachate seeping from the WRDs into the underling soils and eventually joining the saturated zone. Treatment of contaminated water will include first holding the water in settling ponds to settle out any suspended solids. If needed, the water will then be treated further in the proposed water treatment plant. Construct an engineered dam which could spill and contaminate the groundwater resource. Implement a sustainable water supply management plan. Blasting will be controlled to minimise residual nitrogen in the waste rock. Controls can be used to limit wastage to 1% or less. Any spills of ammonium nitrate will be cleaned up prior to blasting. Extensive use of emulsion or heavy ANFO will limit possibilities of spillage. The projected level of wastage will not result in nitrate levels in waste rock runoff or seepage that will exceed the Project's design criteria. Efficient oil and grease traps or sumps will be installed and maintained at refuelling facilities, workshops, fuel storage depots and containment areas, and spill kits should be available with emergency response plans. All dangerous and hazardous material and fuels stores and handling areas should be provided with secondary containment capable of containing the larger of 110 percent of the largest tank or 25% percent of the combined tank volumes in areas with above-ground tanks with a total storage volume equal or greater than 1,000 litres. Reduction of water consumption to minimise drawdown of the groundwater levels: – Collection, treatment, recycling, reuse of wastewater used to wash equipment, surfaces, used for dust suppression purpose. – Monitoring of water consumption and identification of excessive consumption,	• Sustainable Water Supply Management Plan. • Stormwater Erosion and Sediment Control Management Plan. • Groundwater and surface water monitoring plan. • Water Balance.

Impacts	Management Measures	Key Managing Documentation
	– Inspection and maintenance of water infrastructure to detect leakages, inefficiencies, conditioning failures, etc.	
Infrastructure such as the BC pit is as risk of flooding during 1 in 100 year events due to it intersecting the Bankan Creek and its proximately to the Niger river. Surface water quality may also be impacted through hazardous chemical spills or seepage.	A preliminary static annual water balance was completed to evaluate the expected mining operation water use on an annual average basis, which will be updated pre-construction. This is to be implemented into the Sustainable Water Supply Management Plan. Implementation of temporary drainage and sediment control measures prior to construction to ensure that any potentially sediment laden water is treated prior to discharge into existing rivers and creeks and that the volume, peak flow rate and velocity are attenuated to existing greenfield rates, as far as practicable. Develop a storm water management plan: – Collection and recycling of rainwater that falls on non-contaminated areas (such as roofs and undeveloped surfaces within the mine infrastructure area). – Design water runoff management systems to limit erosion, storm water flows and discharge and enable abatement of suspended solid. – Use oil and water separation treatment systems (e.g. vegetative filters and nonvegetative covers including mulches and stone aggregates, limit slopes, runoff control structures, first flush settling ponds, oil/water separator, mine sewage treatment system, etc.) Management measures for hazardous products, effluent and waste products to prevent water pollution: – Implement systematic substitution approach: replace any hazardous products with less or non-hazardous equivalent, where practicable. – Storage and handling of products: use of retentions with recuperation of drips and spillages for the storage and unloading of hazardous products as well as for washing and maintenance activities. – Aboveground tanks for hydrocarbon or other hazardous products' storage preferred to underground tanks.	

Impacts	Management Measures	Key Managing Documentation
	– Storage of hazardous products will be designed and built to allow adequate confinement and protection against leakages. – Storages and networks of hazardous products, i.e. tanks, pipelines and connecting pipes will be regularly inspected by qualified staff and an inspection report will be drawn up. Staff will be trained in good practices in terms of the storage and handling of products and in maintenance, to prevent any risks linked to hazardous products. Regular sampling and chemical analysis of mine effluents to ensure compliance with national and international environmental standards, whichever is the more stringent.	

20.4.3.3 Biodiversity Management

Table 20.4 presents the key management/ mitigation measures which form part of the ESMMP for biodiversity, through the biodiversity management plan (BMP) and the biodiversity action plan (BAP) as it was identified as a significant impact.

Several SCC were identified within the Project AoI and will be impacted, with a particular concern for the Western Chimpanzees (CR on the IUCN Red List). Western chimpanzees were evident through 120 nests identified in the wet season (November 2022) and the camera trap survey (August to September 2023).

The BMP will be written to support the Project's compliance with the IFC PS6: Biodiversity Conservation and Sustainable Management of Living Natural Resources. As part of the mitigation hierarchy, Project offsetting aspects will be set out in the separate biodiversity offset strategy (BOS). A biodiversity action plan (BAP) will also be prepared to assist the Project to comply with the requirements of the IFC PS6: Biodiversity Conservation and Sustainable Management of Living Natural Resources, and the associated WBG Environmental, Health and Safety (EHS) guidelines.

Potential impacts arising from the Project identified through the ESIA and critical habitat assessment (CHA) will be referenced in the BAP.

Table 20.4: Key Management Measures for Impacts to Biodiversity

Aspect	Management Measures	Key Managing Documentation
Loss of habitats (natural habitats such as the gallery forests, bowal, woodland savannahs, wetlands and watercourses) which are threatened and retain SCC) Impacts to the habitats and nesting areas for the Western chimpanzees (CR) e.g. loss of habitat, habitat fragmentation, loss of food security, disturbance and increased human-wildlife conflict. Impact to surrounding protected areas (Haut Niger National Park, Tamba Classified Forest and L'Amana Classified Forest) e.g. habitat loss and fragmentation, disturbance, hydrological impacts from change of flow and drainage and wildlife mortality.	Avoid construction and activities within natural habitats, areas with SCC and critical habitats. Where practicable, construction and excavation activities will avoid peak chimpanzee activity periods within the AoI and surrounding area, which is likely to be during the wet season (May to October) based on baseline survey findings. Avoid areas where chimpanzee nests have been identified. Continue stakeholder engagement to the surrounding communities and with Chimpanzee Conservation Centre and Wild Chimpanzee Foundation. Where avoidance is not possible, suitable measures will be implemented to minimise disturbance, develop and implement a site-specific BMP and BAP, which includes budget and timings as well as progressive rehabilitation. Undertake a critical habitats assessment and implement the findings and mitigation measures into the BMP/BAP. Undertake stakeholder and consultation program to achieve a transparent dialogue and ensure input from stakeholders with an interest in conservation are incorporated into the plan and the design of the mitigation measures. The BMP/BAP should include: – Control of access to areas of importance for biodiversity. – Detail actions for habitat protection, species-specific actions where required, habitat and ecosystem services enhancement, monitoring and reporting protocols. – The BMP shall provide for the closure on acknowledge gaps and limitations currently identified, to allow for the determination, implementation and monitoring of No Net Loss and Net Gain requirements. – Include opportunities to identify the need for any translocation of species including conservation priority flora and fauna in accordance with best practice. Where working in legally protected and internationally recognised areas measure to protect and enhance habitats will be explored and any translocation would be discussed through engagement.	Biodiversity Action Plan (BAP) Biodiversity Management Plan (BMP)

Aspect	Management Measures	Key Managing Documentation
	– Clear demarcation of all protected areas to avoid inadvertent destruction through ignorance or carelessness. – Specification of controls on how vegetation (and associated fauna) is removed. – Management of pest or alien invasive plants and animals. – Management of community biodiversity uses and other ecosystem services. – Research and development programs as deemed necessary, including revegetation trials. – Included in the BAP should be the framework BAP, Biodiversity Offset Strategy and Biodiversity Offset Management Plan etc. Develop a detailed site clearance guideline which includes measures such as no loss of wetland connectivity, minimal loss of ecosystem services, avoid clearing indigenous flora minimal loss of natural habitat and vegetation etc. Harvest areas to be cleared of all smaller indigenous plants, e.g. orchids, that can be maintained in nurseries, harvest seeds and underground storage organs for cultivation and subsequent rehabilitation, build an inventory of desirable grass and shrub/tree species from surrounding areas. Design and implement wildlife corridors where access road will be developed in areas with residing fauna to allow wildlife to migrate and travel, reducing habitat fragmentation. Implement buffer zones around areas of biodiversity significance such as 1 km from the Niger River or wetlands.	

20.4.3.4 Traffic and Transportation Management

Table 20.5 presents the key management/ mitigation measures which form part of the ESMMP for traffic and transport as it was identified as a significant impact. A traffic management plan (TMP) will be developed as part of the ESMS to address the impacts.

Table 20.5: Key Management Measures for Traffic and Transport

Impact	Management Measures	Key Managing Documentation
Impact to road function, degradation to the roads (paved and unpaved) and increased risks of traffic accidents.	Transport routes will be designed to support heavy vehicles and mining equipment. Road safety features are incorporated into the design, including speed restrictions in more populated areas, signage warning drivers of population centres, livestock, schools, etc., speed bumps at the entrance to populated areas, and bus stops/areas where vehicles can park, etc. On transport routes and access roads to the Project site: – Identify pedestrian routes within and near the Project site and roads. – Establish safe zones and crossing procedures with stakeholders and villagers near the Project site. – Improve roads and intersections near the Project to handle increased traffic. – Install signage and, where possible, erect barriers along road sections to discourage pedestrian use. – All construction sites must be appropriately marked with high-visibility signs, cones, and barriers to minimize unintentional or intentional intrusions and keep community members and outsiders away from construction areas. – Collaborate with local authorities to schedule truck deliveries, especially heavy truck deliveries, to reduce impacts on road operation and safety. Where possible and safe, schedule deliveries to minimize impact on other road users, based on local conditions and stakeholder engagement results. – Repair road damage caused by traffic before and during construction, either immediately (for significant damage that prevents or greatly hinders future public use) or at the end of the construction phase, in collaboration with national and local road authorities. Conduct appropriate risk assessments before exceptional load deliveries to evaluate routes for large delivery trucks, potential obstacles, or necessary road modifications, and determine risk mitigation measures for structures or properties along the routes. Obtain necessary permits for public road use.	Traffic Management Plan (TMP) Stakeholder Engagement Plan (SEP)

Impact	Management Measures	Key Managing Documentation
	Plan delivery truck routes using roads with geometry and load capacity that ensure safe passage: – Training and accreditation of Project drivers, including subcontractors. – Standards for driver fitness, including mandatory rest periods and prohibition of drug or alcohol use. – Onboard monitoring systems to control vehicle speed and location (Project and subcontractor vehicles). – Project and subcontractor vehicle safety and maintenance standards. – Safety intervention in case of vehicle incidents. – Load stability standards. As part of a public consultation program related to the Project, inform, educate, and regularly update stakeholders and communities near transport routes about Project traffic, especially safety issues and schedules associated with heavy and large delivery trucks on public roads. A TMP (including vehicle movements, frequency/times of day, probable routes, and associated risk assessments) will be developed and implemented, considering the following: – Safe workplace arrangements. – Vehicle safety equipment standards (e.g., seat belts and first aid kits). – Driving rules (e.g., speed limits, driving hours, mandatory breaks, passenger transport, and use of mobile phones/radios). – Driver qualification and selection (e.g., defensive driving courses, accident history, and practical interviews to test skills). – Driver education and training (awareness, information on required standards, and incident review). – Vehicle inspection and maintenance (in accordance with international vehicle inspection standards). – Onboard monitoring systems to control vehicle speed and location (Project and subcontractor vehicles).	

Impact	Management Measures	Key Managing Documentation
	– Site layout plans including traffic routes, pedestrian crossings, rights of way, signage, etc. – Accident/incident reporting and investigation. – Disciplinary procedures.	

20.4.3.5 Waste Management

Table 20.6 presents the key management/ mitigation measures which form part of the ESMMP for Waste Management, mineralised and non-mineralised waste, as they were identified as a significant impact due to the possibility of contamination.

Non-mineralised waste includes general waste such as industrial waste, domestic waste or garden waste that can generally be recycled or re-used, and hazardous waste such as oils, grease, medical waste, sewage waste and other chemicals used. Mineralised waste refers to tailings which will be stored and managed at the TSF with the filtered-stack method, aligning with the IFC-EHS Guidelines.

Table 20.6: Key Management Measures for Waste Management

Aspect	Management Measures	Key Managing Documentation
Non-Mineralised Waste		
General Waste and Hazardous Waste	Procurement – selection of materials and products that generate the least possible waste, where practicable, incorporate least waste generation potential product options in selection and procurement criteria. Inventory management: – Maintain inventory management system with a view to identifying product consumption, ensuring the traceability of waste, and identifying any wastage and over-consumption. – Maintain an inventory of all waste generated and eliminated (type and volumes). – Develop objectives for reductions in the amounts of waste generated, based on a periodic review of inventories. Staff training and management: – Staff will be trained in correct applicable waste management procedures and subjected to waste risk identification and mitigation training. – Waste will be handled and stored according to type and risk classification, in compliance with health and safety rules and hazardous materials management and spill prevention protocols. Satellite waste collection and temporary transfer (SWCTT) facilities shall be managed as follows: – SWCTT facilities shall be selected and approved in functional areas which shall be fenced, and access controlled. – Designated and suitably trained operational personnel shall have overall responsibility to maintain these areas in acceptable condition. – Hazardous waste storage shall be minimised but where unavoidable, areas shall be fitted with suitable protection features/infrastructure to prevent soil and groundwater contamination.	Waste Management Plan (WMP)

Aspect	Management Measures	Key Managing Documentation
	– Compatible waste materials will be stored together. Final disposal of waste: – Recyclable waste will be collected by suitably qualified and government certified companies. Waste collection contracts with these companies will be confirmed upon successful commercial, social, environmental, health and safety management due diligence assessment. – All hazardous and non-combustible waste will be processed appropriately in the country or exported abroad for processing and final discharge. Any export of waste for elimination outside the borders of Guinea will meet the demands of the Basel Convention on the control of transboundary movements of hazardous wastes and their disposal as well as the Bamako Convention on the ban of the import into Africa and the control of transboundary movements and management of hazardous wastes within Africa. – Medical, and potentially infectious waste will be place in dedicated, labelled receptacles, for disposal to a dedicated incinerator. – No waste will be burned in the open air. Transport of waste off site: – When waste materials are sent off site, suitable transport vehicles will be used to ensure loads are safe, properly labelled, and traceable. – Transport vehicles used will be fitted with means with which to act in case of any accidental spillage.	
Spillages and leaks of hydrocarbon and other hazardous waste	Spillages of all hazardous chemicals and large hydrocarbon spills need to be contained with urgency as they are extremely toxic to biodiversity and treated in accordance with the hazardous waste management plan to prevent such chemicals spreading into surrounding habitats. The above spillage control also needs to be implemented in the case of an unforeseen accident or breakdown of machinery, vehicles or trucks (also along transport routes outside the AoI), especially where trucks transport hydrocarbons and chemicals.	Emergency prevention, preparedness and response plan. Waste Management Plan (WMP).

Aspect	Management Measures	Key Managing Documentation
	Areas where spillage of soil contaminants occurs, must be excavated (to the depth of contamination) and suitably rehabilitated. If any other minor spillage occurs the spillage must be cleaned as soon as possible, but within the same shift and the contaminated area must be reinstated. All contaminated material should be suitably treated and cleaned, preferably by bio-piles where such is feasible. Parking and operational areas must be regularly inspected for oil spills and covered with an impermeable or absorbent layer or grease pans (with the necessary storm water control) if oil and fuel spillages are highly likely to occur. Vehicles and machinery should be regularly checked and serviced to ensure there is minimum risk of leaks. Follow the waste management plan for proper disposal of hazardous waste, such as ensuring that hazardous waste is disposed of in a separate and labelled bin (or container) in ana rea which is closed off before incineration or collection an appropriate waste disposal company.	
Mineralised Waste		
Tailings	The TSF will be lined with four-layer impermeable liners with drainage channels around the perimeter to collect runoff from the tailings and channel any surface water around the storage area. As the tailings are stacked up to their designed height, based on geotechnical design constraints, they can be rehabilitated with fresh rock waste and vegetation, similar to the waste rock dumps. The design of the TSF will take into account the specific risks/hazards associated with geotechnical stability or hydraulic failure and the associated risks to downstream economic assets, ecosystems and human health and safety. Environmental considerations will also consider emergency preparedness and response planning and containment/mitigation measures in case of a catastrophic release of tailings or supernatant waters. Any diversion drains, ditches, and stream channels to divert water from surrounding catchment areas away from the tailings structure will be built to the flood-event-recurrence interval standards outlined elsewhere in this Section.	Environmental Management Plan (EMP). Tailings Designs.

Aspect	Management Measures	Key Managing Documentation
	Seepage management and related stability analysis will be a key consideration in the design and operation of the TSF. This will include a piezometer-based monitoring system for seepage water levels within the structure wall and downstream of it, which will be maintained throughout its life cycle. A zero-discharge tailings facility will be considered as part of this project, and the completion of a full water balance and risk assessment for the mine process circuit, including storage reservoirs and tailings dams, will be undertaken. Natural or synthetic liners will also be considered. Design specification will take into consideration the probable maximum flood event and the required freeboard to safely contain it (depending on site specific risks) across the planned life of the tailings dam, including its decommissioned phase. Where potential liquefaction risks exist, including risks associated with seismic behaviour, the design specification will take into consideration the maximum design earthquake. On-land disposal in a system that can isolate acid leachate-generating material from oxidation or percolating water, such as a tailings impoundment with a dam and subsequent dewatering and capping, will be considered. On-land disposal alternatives will be designed, constructed and operated according to internationally recognized geotechnical safety standards. Thickening or formation of paste for backfilling open pits on surface and underground workings during mine progression, where feasible.	

20.4.4 Socio-Economic Management

The ESMMP includes specific socio-economic management measures to address the negative and positive impacts identified. The following management plans will be developed as part of the ESMMP and ESMS (refer to ESIA for detailed plans):

- Livelihood Restoration Plan.
- Stakeholder Engagement Plan.
- Community Grievance Mechanism.
- Artisanal and Small-Scale Gold Mining Management Plan.
- Community Health and Safety Management Plan.
- Community Development Plan.
- Project-Induced Influx and In-Migration Management Plan.
- Labour and Workers Accommodation Management Plan.

Table 20.7 presents the key management/ mitigation measures which form part of the ESMMP for traffic and transport as it was identified as a significant impact.

Table 20.7: Key Management Measures for Socio-Economic Impacts

Impact	Management Measures	Key Managing Documentation
Project Related In-flux	A grievance mechanism will be developed, whereby affected people can raise issues and concerns associated with vehicle movements, driver behaviours and report accidents or damage to property they feel are caused by Project. Conduct awareness programs informing relevant stakeholders about employment opportunities to manage expectations. Ensure that information on employment and procurement strategies is communicated in all settlements located within the social AoI; Provide clear information on the number and timing of employment opportunities at each phase of the Project. Strengthen the capacity of national suppliers through a comprehensive supply and demand analysis, phased capacity-building programs, and targeted training agreed upon with local authorities and other local organizations. Implement a phased capacity-building program (sector by sector) enabling local businesses to obtain qualifications and possibly certification in accordance with relevant standards and requirements well in advance of the tendering process. Collaborate with local authorities and other local organizations to define targeted training opportunities (all selected potential suppliers must meet the required hygiene, safety, and quality standards). After selecting direct service providers, train them on the Project's HSE and socio-economic policies before the start of construction. Training of providers and employees on the Project's hygiene and safety requirements (aligned with internal policies) and socio-economic policies before the start of activities.	Livelihood Restoration Plan. Stakeholder Engagement Plan. Community Grievance Mechanism. Artisanal and Small-Scale Gold Mining Management Plan. Community Health and Safety Management Plan. Community Development Plan. Project-Induced Influx and In-Migration Management Plan. Labour And Workers Accommodation Management Plan.
Impacts associated with loss of land and livelihoods resulting in economic displacement.	Develop an artisanal management plan and livelihood restoration plan.	

Impact	Management Measures	Key Managing Documentation
Surrounding communities have a level of expectations for employment, livelihood support etc which if unmet can result in upset and tension.	Engagement with stakeholders in the artisanal mining sector to establish conditions for coexistence, seek constructive long-term relationships, potentially improve practices in the sector, and prevent or minimize economic shocks to local livelihoods. Promoting diversification through participatory alternative livelihood programs for ASM. Establish a community development plan which includes capacity building and training for employment opportunities and business potentials. The Project adheres to the hiring and employment standards contained in the Mining Code, the Local Content Law, and the Labor Code (hiring quotas). Capacity building and knowledge transfer to local providers and their employees through the development of formal training programs and, where possible, formalising on-the-job training with learning objectives and performance monitoring. Implement awareness and training in alternative livelihoods which are sustainable particularly for the project-affected person (PAP) and economically displaced persons. Implementation a procurement strategy to have a local workforce (skilled and unskilled), if unskilled then provide the necessary training. Future local economic development initiatives should aim to strengthen or create: – A network of local and regional subcontractors meeting the mine's needs. – Capacity building for communities and subcontractors. – Development of value chains between Kouroussa and Conakry, and possibly other cities in Guinea. – Creation of indirect local/regional jobs. – An increase in food production and strengthening of food security. – Improvement of access to financing mechanisms. The Stakeholder Engagement Plan will be implemented to establishing strong relationships between the Project and its stakeholders, creating an atmosphere of mutual understanding, respect, trust, and collaboration.	

Impact	Management Measures	Key Managing Documentation
	The Community Development Plan will be developed to maximise the local economy's benefit from the presence of the Project in the area and will also include community development indicators to assess the Project's participation in local economic and social benefits. The Community Development Plan will ensure a sustainable future for the communities surrounding the Project through community development and community empowerment in the following ways: – Meeting the basic needs of the community. – Constructing infrastructure (health, water, roads, etc.) independent of the mine. – Promoting renewable energy. – Local economic development through SME development initiatives. – Strategies to support local procurement, including strengthening the management capacities of local subcontractors. – Recruitment supported by the development of the local workforce and the enhancement of employees' technical and managerial capacities. – Capacity building for local government and communities. The Community Development Plant is defined based on the following axes and types of actions: – Population health. – Water and sanitation. – Food security and the fight against malnutrition. – Education and training. – Local economic development and income generating activities.	
Community health and safety impacts, include road safety, nuisance impacts (noise, air quality, dust, vibration,),	In the event of an accident in which a community member is harmed, the Project should organise transport for the injured person to an appropriate health facility capable of dealing with the injuries and facilitate access to medical treatment.	

Impact	Management Measures	Key Managing Documentation
trespassing and increased spread of diseases.	Accident reporting and investigation procedures should be developed to determine root causes and identify corrective measures to reduce the risk of the accident happening again. During operation there is potential for road traffic accidents along public roads – as per the TMP. A grievance mechanism will be developed, whereby affected people can raise issues and concerns associated with vehicle movements, driver behaviours and report accidents or damage to property they feel are caused by Project vehicles. The grievance management mechanism will be fully accessible to all stakeholders affected by the Project, allowing them to submit their grievances conveniently. There will be multiple appropriate channels through which stakeholders can submit their complaints free of charge and without retribution to the party causing the issue or concern. Undertake stakeholder engagement and health programmes to educate and inform the community members of potential diseases, how they are spread and how to prevent them.	

20.4.5 Cultural Heritage Management Plan

Table 20.8 presents the key management/ mitigation measures for the identified impacts to cultural heritage. A CHMP will also be developed and implemented as recommended in the ESMMP and align with the objectives of the IFC-PS 8: Cultural Heritage, which stipulate the following objectives:

- Protect cultural heritage from the adverse impacts of Project activities and support its preservation; and
- Promote the equitable sharing of benefits derived from the use of cultural heritage.

Table 20.8: Key Management Measures for Cultural Heritage

Impact	Management Measures	Key Managing Documentation
Direct impact to cultural heritage sites within project infrastructure footprint and indirect impacts (loss of access, intrusive visual, auditory or dust elements). Chance finds of unknown cultural heritage sites during construction and operation which will lead to the site's destruction or damage.	Regulator engagement with the National Directorate of Historical Heritage to agree site-specific mitigation measures. A comprehensive Cultural Heritage Management Plan (CHMP) will be developed pre-construction to adequately manage cultural heritage. A Chance Finds Procedure will be designed and implemented to manage any unexpected discovery of archaeological material in-line with international requirements and guidelines IFC PS8. Access arrangements will be made to the satisfaction of identified stakeholders through a memorandum of understanding (MOU) agreed to by authorities and identified stakeholders, which will allow unrestricted access to cultural heritage resources. This memorandum should be in place before construction begins. The implementation of the mitigation hierarchy should be implemented where cultural heritage sites should be avoided, where possible re-design infrastructure to avoid these sites. Where avoidance is not feasible, the Project shall apply a mitigation hierarchy that minimises adverse impacts on cultural heritage, as far as practicable. Detailed site-specific archaeological mitigation, such as pre-construction investigations, archaeological excavations etc. will be undertaken. Ensure cultural heritage is incorporated explicitly as a topic in the community grievance mechanism. Sacred sites and grave relocation plan should be developed in collaboration with the Social Performance Team and engagement with community representatives, if required.	Cultural Heritage Management Plan (CHMP). Chance Finds Procedure. Community Grievance Mechanism. Grave Relocation Plan (if required).

20.4.6 Closure and Rehabilitation Management Plan

The Project's ESIA contains a conceptual plan for the rehabilitation, decommissioning and closure of the Project. The overriding intent of mine closure and rehabilitation is to return the land as close as is reasonably practical to its pre-disturbance condition that is pastoral and agricultural activities. This will be achieved through establishment of a safe and stable post-mining land surface which supports vegetation growth and is erosion resistant over the long-term. Closure and rehabilitation activities will be undertaken during operations (progressive closure) or after operation (final closure).

The preliminary closure objectives have been proposed as follows:

- Physical stability - The landscape and any retained infrastructure must remain structurally and geotechnically stable.
- Geochemical Stability - The closure plan will prioritize geochemical stability of its components to maintain native vegetation and compensation areas.
- Enhancing Biodiversity - The closure plan will be designed to enhance ecosystem services, sustainably integrating them into existing economic activities in the region in a post-mining land use scenario, ensuring compatibility with the objectives and requirements of the PNHN.
- Health and Safety - The closure plan will be designed to ensure the environmental and health safety of employees and communities and will include technical training on economic activities that may continue during the Project's operational and closure phases.
- Water Quality - The use of surface and groundwater will be aligned with Technical and Planned Use of Water Resources (*Utilisation Technique et Planifiée des Ressources en Eau*) (UTPM)), and water quality and quantity must meet legal requirements and remain stable post-closure. Pits will likely be filled with water as part of restoration, and UTPM options will seek to use them as socio-economic assets.
- Socio-Economic Transition - A socio-economic transition plan should be developed to accommodate alternative economic activities in the region, in line with community expectations and needs. This plan will prioritize current economic activities in the region as long as they comply with the defined UTPM. It will also seek to enhance the socio-economic value of the region by exploring other options for economic land use.
- Regulation - The Company will identify and ensure compliance with all its internal and external commitments, including national and international legislation and standards.

The closure and rehabilitation plan includes procedures if the mine experiences temporary or sudden closure. The plan also includes maintenance and monitoring post-closure covering physical aspects of the Project such as landforms (geotechnical stability), water and biodiversity. The objective of long-term monitoring to validate compliance with closure success criteria, which includes the safety of landforms, maintaining appropriate water quantity and quality, and establishing self-sustaining ecosystems and habitats.

As described in the ICMM and the IFC-EHS guidelines for mining (International Council on Mining and Metals, 2022) (International Finance Corporation, 2007), an effective mine closure and planning and implementation, take into consideration the view, concerns, aspirations, efforts and knowledge of internal and external stakeholders to identify mutually beneficial closure outcomes for the company and its host communities. Therefore, stakeholder engagement and participation will be undertaken and is essential in the success of the mine's closure and social transition. Stakeholder engagement will be undertaken throughout the mine's lifecycle with an approach that empowers the local community in the decision-making process.

20.4.6.1 Closure and Rehabilitation Activities per Infrastructure

A summary of the closure and rehabilitation activities recommended per infrastructure are provides in Table 20.9.

Table 20.9: Closure and Rehabilitation Plan for the Project Infrastructure

Infrastructure	Closure Activities
Open Pits (BC pit and NEB pit)	Upon cessation, the two pit areas; BC pit and NEB pit will naturally flood. For the NEB pit, water will come from the overflow from the upper slope of the pit, in conjunction with overflow from the NEB WRD. For the BC pit, if natural recharge is insufficient, Bankan Creek can be redirected to help recharge the pit if necessary. Safety measures such as berm elevation, fencing, and signage will also be installed around the pit perimeter. Updated hydrogeological studies will be required during the Project's lifespan to estimate the time required to flood both pits and the water level they will reach. The lakes/watercourses that will form in the pits will create a new ecosystem that could be connected to surface watercourses to promote biodiversity. Geochemical studies will need to be updated during the Project's lifespan to define water quality projections for the closure and post-closure phases. This information is important to decide on necessary actions to prevent the water from becoming acidic, if possible.
Underground Mine	The proposed base scenario is that underground voids will be backfilled with cement paste to ensure appropriate stability and subsistence, with portal backfilling requiring more detailed study to ensure proper stability. All equipment will be removed, and an audit of materials, products, and installations remaining underground will need to be conducted before closure. Developing a detailed groundwater model to better understand potential impacts on underground workings will be necessary. This model must be supplemented with data on the quantity and quality of groundwater generated during the Project's lifespan to understand the risks associated with the conceptual closure plan.

Infrastructure	Closure Activities
Waste Rock Dumps (WRD)	The two WRDs are located next to their corresponding pit. The BC WRD, will be available later in the Project's life for closure, while the NEB WRD will complete its operational life in parallel with the Project's lifespan. Partial closure can be developed in areas where the WRD has reached its final configuration. Both WRDs will follow the same closure process. The WRDs will be fully or partially regarded as safe, stable, long-term slopes. The final slope should be 22 degrees, with 10 m benches and 15 m wide berms to minimise erosion, creating an overall slope of 14.1 degrees. When closing a WRD, environmental considerations may include mitigation measures such as recontouring buffer zones to restore habitat and wildlife corridors or reworking drainage patterns where water quality impacts could occur. As designed, potentially acid-generating waste will be encapsulated with appropriate material to avoid potential impacts from acid and metalliferous drainage. A single layer of enriched topsoil is therefore envisaged for the base case of closure, the required volume will be defined in the next stages of the Project, and a mass balance will be developed. The existing drainage subsystem will be maintained and monitored throughout the closure period. If the water meets legal quality requirements, it will be redirected to a discharge point; otherwise, it will be redirected for treatment. After closure, the focus will be on the presence of natural drainage systems and the absence of water requiring active treatment systems.
Water Storage Facilities (WSF)	The WSF configuration will be designed considering closure slopes, therefore, no regrading will be necessary. The WSF design could incorporate a progressive cell closure strategy to avoid water infiltration into the dry stack during the operational phase. The cover system (designed using layered materials with specific properties for specific reclamation objectives) will consider two layers: a drainage layer to capture and divert any infiltration over the deposits, and an enriched topsoil layer to facilitate reclamation. Saprolite has been identified as an appropriate choice of waste rock to use as a drainage layer due to its chemical composition and low permeability. Topsoil will come from stocks developed during the construction phase and later phases; the required volume will be defined in the next stages of the Project, and a mass balance will be developed. Soil amendments or growth supports may be necessary to supplement or cover the topsoil demand.

Infrastructure	Closure Activities
Processing Plant and Support Infrastructure	Several infrastructure elements will remain available for use within the UTPM framework. Any other buildings without a UTPM will be demolished down to the foundation level. The closure plan provides a preliminary list of buildings and their post-mining land status, while recognizing that this status may change as the closure plan evolves. The infrastructure that will remain after closure will be reviewed with stakeholders and relevant authorities to establish a clear transfer process. For the buildings that will be demolished, the proposed base scenario for their closure is as follows: – Foundations, footings, and concrete curbs will be completely removed; voids will be backfilled with inert materials up to ground level and covered with topsoil. – Decontaminate concrete slabs. – Previously removed overburden/site soil will be used to cover the slabs. – All surface pipelines will be removed unless they are needed for post-closure water management. – All recoverable materials (e.g., steel, piping, copper, stainless steel, and aluminium) will be removed before demolition and offered for reuse or to benefit local communities. Any remaining materials will be disposed of at a recycling facility. – All non-recyclable residual waste will be disposed of at an approved disposal facility, either on-site or off-site. – Any potential hazards identified during the closure process will be removed and disposed of at an approved disposal facility. – Final grading will be designed to be compatible with the surrounding topography. – Vegetation will be designed as part of the biodiversity management plan. – In cases where roads are not part of the post-mining land use plan, they will be dismantled and replaced with a 300 mm subsoil layer, followed by 300 mm of topsoil and, if applicable, appropriate vegetation.

Infrastructure	Closure Activities
Water Management Infrastructure	As part of the closure process, a comprehensive reassessment of existing water management structures will be conducted to ensure that closure plans are based on the most recent information, minimise contact water, and that long-term maintenance will not be necessary based on available geochemical information on local water. This will include the following aspects: – The drainage system will be reviewed as part of the closure design. The infrastructure must support a 100-year, 24-hour rainfall event while incorporating a long-term maintenance design. – Sedimentation basins will be adapted to become wetlands if water quality and flow allow. If water quality does not permit, sedimentation basins will remain in place until the final closure stage. Once water quality meets legal requirements, they will be covered with reclaimed soil, and the basin footprints will be revegetated using native species. – Water Treatment Plant (WTP): The freshwater treatment plant should be demolished at the end of the closure period. – The treated water storage tank should remain operational for use within the UTPM framework. – The wastewater treatment plant, if required, should remain operational for use within the UTPM framework. – Any potentially hazardous material identified during the closure process will be removed and disposed of at a contaminated site in an approved disposal facility.

Infrastructure	Closure Activities
Contaminated Sites	On-site studies should be conducted to monitor potentially contaminated sites for their dismantling and progressive restoration during operations and closure, including areas such as: – Processing plant and reagent mixing areas. – Cyanide facility and support infrastructure. – Power supply and distribution. – Fuel storage areas. – Explosives storage. – Processing plant workshop. – Processing plant warehouse. – Processing plant laboratory. – Temporary hazardous waste storage areas.

20.4.6.2 Closure Liability Estimate

The closure liability estimate was developed from first principles based on the requirements outlined in the ESIA and updated from the ESIA to reflect the current closure costs in 2025 with costs as follows:

- Waste dump profiling, included in mining costs and carried out progressively over the life of mine.
- Abandonment bunds and plugging of the underground portal, included in the mining costs.
- Topsoil re-spreading on mining areas and waste rock dumps, included in the mining costs.
- Water management structure rehabilitation, US$675,000.
- Removal of fencing, overhead powerlines and pipelines, US$160,000
- Process plant removal, removal of concrete and rehabilitation, US$24,840,000.
- TSF capping of final stage and rehabilitation, US$1,965,000.
- TSF water storage pond closure and rehabilitation, US$1,815,000
- Accommodation village dismantling and rehabilitation, US$845,000.
- Rehabilitation of roads, US$455,000.
- Closure project management and monitoring, US$5,975,000.
- Contingency, US$3,640,000.

The total closure cost, excluding closure and rehabilitation activities included in the mining costs and post closure monitoring, is US$40,010,000.

20.4.6.3 Post Closure Monitoring

The objective of long-term post closure monitoring is to validate the fulfilment of closure success criteria, which include ensuring the safety of landforms, maintaining appropriate water quantity and quality, and establishing self-sustaining ecosystems and habitats. The monitoring and maintenance program will be evaluated further during future study and operation of the mine.

The monitoring and maintenance program will include the below:

- Geotechnical stability – A geotechnical monitoring program will include visual inspections, annual site inspections and use of instrumentation tools. If a geotechnical concern is identified, it will need study by qualified engineers, and mitigation measures must be implemented to maintain long-term stability.
- Reclamation monitoring – Monitoring the success of reclamation during and after closure including visual inspections to identify signs of erosion and soil loss, soils samples to be collected. Due to the low quality of the soil, all revegetated areas will be sampled to check that the soil’s restored properties are maintained over time and the ecosystem is self-sustaining.

- Water monitoring – Groundwater and surface water should be monitored during closure and post closure, particularly as the tailings could be PAF/ML, however it is assumed that the dry-stack tailings won't need water treatment. Groundwater monitoring is to be conducted on the boreholes, while surface water monitoring should be conducted upstream and downstream (as per the baseline sampling locations).
- Biodiversity – The monitoring and maintenance program will be developed as part of the Biodiversity Management Plan, with the program defining monitoring points, parameters and monitoring frequency. This is to begin as progressive closure.

Based on the program outlined above, an estimate has been made for the post-closure monitoring that will be undertaken. This cost has been estimated at US$3.2m and will take place over a period of up to five years post closure.

20.5 Land Acquisition and Resettlement

The Project requires the acquisition of approximately 2,000 ha of land for the establishment of infrastructure, consequently this will result in economic displacement of the landowners and occupant, leading to the loss of livelihoods. Land that is being used for agriculture, livestock (grazing), ASM sites and areas used for ecosystem services will be acquired by the Project. Additionally, the establishment of the Project will result in change of access to land and areas which have not been acquired. ASM is one of the main forms of income for the surrounding households surveyed, the majority of the ASM sites are located within the mining permits thus eviction and restriction of these areas will result in reduced income. The loss of agricultural land and livelihoods could lead to increased food insecurity and reduced forms of income.

In particular, the communities in Bankan and Kignédouba will lose access to land in preparation for the construction of the Project. Therefore, the impact of economic displacement will require careful and proper management pre-construction to minimise the significance of this impact through an economic resettlement action plan (RAP), livelihood restoration plan (LRP), ASM management framework and stakeholder engagement framework.

20.5.1 Land Ownership Access Requirements

The Project will align with the relevant national and international legislations /frameworks for the acquisition of land and land access. The *Code Foncier et Domanial (Ordonnance 0/92/019)*, or land and public estate code, is a legal framework which governs the land tenure and property rights in Guinea. It defines two procedures for land registration:

- Cadastral plan, which an administrative document not a title in itself, which is kept at the municipal level in the cities and in the community for rural development in rural areas.
- Registration of land ownership leading to the issuance of land title.

In practice, these procedures of land registration have not been fully implemented in rural areas, where customary rights predominate; in the absence of formal private property, the land is essentially the State's property. The Project will expect that the terms of access to and occupation of the land required for the Project would be provided for in the mining title itself or it in the mining convention.

Article 123 of the Guinea Mining Code states that the granting of a mining title and the authorisation to occupy the land within the Project's footprint does not affect the rights of the owners, usufructuaries and occupants of the land as well as those of their beneficiaries. Consequently, if the State grants the holder of a mining title the possibility of occupying all the land necessary for its activities within its perimeter, provided that it is authorised to do so by its title or by order of the Minister in charge of mines, this must be done by paying the legitimate occupants of this land compensation intended to cover the disturbance of use that they suffer. Such compensation must be in an amount reasonable enough not to compromise the viability of the Project and proportionate to the disturbances caused by mining operations (Article 124). In the absence of the consent of the landowner or his successors, the State may impose on the latter, for reasons of public utility, either an expropriation, which then gives rise to compensation which may not be less than that provided for the aforementioned occupation, or the obligation, subject to adequate and prior compensation fixed as in the case of expropriation, to allow the work to be carried out on his property and not to hinder it (Article 125).

20.5.2 Resettlement Action Plan and Livelihood Restoration

The Project has established a Resettlement and Compensation Policy Framework (RCPF) which will serve as the basis for the development of the economic resettlement action plan and the LRP. The RCPF follows the national legislations and standards as well as international standards.

The economic resettlement action plan and LRP will aim to integrate all aspects of the planned economic resettlement related to the Project and the concurrent livelihood restoration activities into the ESMMP and, ultimately, into the Project's ESMS. This plan will serve as the basis for defining and implementing operational compliance management procedures and practices in all Project activities undertaken by the Company, its contractors, and suppliers. All aspects of this plan will be integrated into the Company's activities during the construction, operation, and closure phases of the Project. The RCPF specifically aims to:

- Define the principles and procedures governing land acquisition, displacement, and involuntary resettlement caused by the Project.
- Enable the Company to define future human, technical, and financial resource needs.
- Plan future relevant engagement activities with stakeholders.

The economic resettlement action plan and LRP will aim to include the following elements:

- Avoid or minimize, as much as possible, involuntary resettlement and land acquisition by exploring all viable alternatives during the Project design.
- Improve, or at least restore, the livelihoods and living standards of people affected by displacement.
- Ensure that affected persons are consulted and can participate in all key stages of the development and implementation process of involuntary resettlement and compensation activities.
- Ensure that compensation is proportional to the impacts suffered, to verify that no person affected by the Project is disproportionately penalised.

- Ensure that affected persons, including those identified as vulnerable, are assisted in their efforts to improve their livelihoods and living standards, or at least to maintain them at their pre-resettlement level or their pre-Project level, whichever is more advantageous for them.
- Preparation of databases for stock and complaint management.
- Establish institutional framework, particularly the establishment of monitoring committees with framework agreements.
- Creation of a schedule linking the economic resettlement action plan to the mining schedule and the schedules of other stakeholders (e.g., the agricultural schedules of farmers affected by the Project).
- The compensation strategy is defined by the resettlement principles outlined in Guinean legislation and international standards, particularly IFC Performance Standard 5. Where the Guinea requirements differ to international, use the stricter standards.
- Compensation should be in kind, in land or in cash, where a person is able to choose between two or more compensation options according to their assets.
- The economic resettlement action plan process will require a grievance management mechanism, stakeholder engagement, and inclusion of non-land-related impacts from the Project.
- Livelihood Restoration - Initial Social Analysis: During the economic resettlement action plan inventory operations, affected households will undergo initial socio-economic surveys to define the main demographic, economic, health, and social characteristics of the PAPs and their households. The socio-economic surveys will include the following elements necessary for the development of the LRP:
  - Livelihood Impact Assessment – Depending on the nature of the impacts and the necessary displacements, the LRP will include a qualitative and quantitative assessment of the impacts on the livelihoods of the PAPs.
  - Development of a Livelihood Restoration Strategy - Consultations with the PAPs in the form of focus groups and with key stakeholders (technical services, NGOs, microfinance organizations) to develop a livelihood restoration strategy.

Once the LRP has been developed and validated by the PAPs and stakeholders, the Company can proceed with its implementation.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimate

Capital costs for Project have been developed based on the designs developed and described in this report. In addition, the costs have been developed in line with the implementation methodology and contracting strategy presented in Section 24.1. The estimate has generally been compiled based on the mechanical equipment list, specifications and material take offs (MTOs) produced for the DFS, and include:

- Purchase and installation of permanent plant, equipment and materials.
- Construction labour.
- Construction plant and equipment.
- Contractors' preliminaries, overheads and profit.

The capital cost estimate for the Project has been developed to be generally consistent with the requirements of an AACE Class 3 estimate with an accuracy of ±15%.

In addition, operational costs incurred prior to the commencement of ore processing, particularly the underground mine development and other pre-production mining, have been included in the capital cost estimate, however, have been estimated in the same manner as operating costs in those areas.

21.1.1 Basis of Estimate

The capital cost estimate was compiled on the following basis:

- Earthworks quantities have been derived from designs developed from 3D modelling of earthworks based on the requirements identified by specialist studies and pricing has been developed based on detailed quotations from earthworks contractors active in Guinea.
- Bulk materials and equipment quantities have been developed based on the engineering completed during this study including allowances for growth and wastage.
- Major equipment supply has been included based on budget quotes sourced during the study.
- Bulk materials supply and installation pricing has been compiled based on budget quotations from fabricators and contractors based on preliminary scopes of work, which were used to develop unit rates to be applied to final quantities.
- Construction and installation costs have been based primarily on submissions from construction contractors based on preliminary scopes of work which was utilised to develop preliminary and general costs as well as installation norms and labour rates.
- The estimate has been compiled on the execution basis outlined in Section 24.1 and utilising traditional field installation with equipment and bulk materials brought to the site in the largest possible items able to be transported via standard gauge road transport.
- Project indirect costs have been included based on detailed estimates and aligned with the execution plan and schedule.

- Owners costs including:
  - Owners project management team and indirect costs.
  - Mining costs, including detailed mine design, carried out by consultants, and owner's mining equipment costs.
  - Environmental consulting for construction clearances, auditing and operational planning reviews.
  - First fills and spares inventory.
  - Project insurances.
  - Site security costs during construction.
  - Site medical services costs during construction.
  - Pre-production labour costs.
  - Operational readiness.
- Pre-production mining costs have been estimated in line with operating cost estimates and included in capital costs where prior to commencement of ore processing.
- Additional site investigation costs for plant site and TSF geotechnical conditions, underground mine geotechnical conditions and grade control drilling of the GBE pit.
- Contingency has been included at the P_{80} based on the quantitative risk analysis (QRA).

21.1.2 Estimate Methodology

The methodology for the development of the estimate is outlined in Table 21.1 and the required deliverables basis and project data required to support the estimate is outlined in Table 21.2. The table shows the minimum standard adopted by the Qualified Person for a Class 3 estimate with a comparison to the standard applied in the DFS and outlines the reasoning behind evaluation of the estimate accuracy.

Table 21.1: Capital Cost Estimate Methodology

Aspect	Minimum Standard	Method Used	Comments
Level of Definition (expressed as a percentage of complete engineering using appropriate indicators ie % of EPCM, % of Engineering Cost)	15% to 25% of full project engineering definition	Approximately 16% of final engineering costs expended in DFS engineering	Meets standard
Typical Accuracy Range based on the P10 and P90 levels	10 to 15%	±15%	Majority of estimate methodology meets minimum standard with only minor areas not meeting the standard and some exceeding the standard.
Quotations / Tenders – Supporting the Estimates	Multiple firm equipment quotes. Multiple material supply and construction quotes and rates checked. Other contracts tendered and evaluated or near heads of agreements/final agreements specific to the business case.	Multiple quotations for all major equipment with only minor equipment from database pricing. Construction contracts tendered to multiple parties, including fabrication. Earthworks tendered to multiple parties. Mining contracts tendered to multiple parties. All contracts tendered with project specific term sheet developed for the DFS.	Meets standard
Mining Costs			
Mineral Resource Classification	Measured (preferable), Indicated and Inferred	Over 90% of metal in Indicated category	Meets standard
Geotechnical Conditions	Defined	Defined – extensive geotechnical drilling, logging and testing program undertaken	Meets standard

Aspect	Minimum Standard	Method Used	Comments
Hydrological Conditions	Defined	Defined – detailed hydrogeological and hydrological modelling undertaken	Meets standard
Site Layout	Defined – Generally optimised	Defined – layout takes into account all aspects and it optimised	Meets standard
Mine Design Criteria	Defined – Generally optimised	Defined	Meets standard
Waste Dump Design Criteria	Defined – Generally optimised	Defined	Meets standard
Mine Schedule	Optimised or Preliminary - Matched to Fleet	Preliminary and matched to fleet	Meets standard
Mine Equipment	Calculated or Detailed	Calculated	Meets standard
Mine Services	Calculated or Detailed - full outlines	Calculated	Meets standard
Mine Environmental Compliance	Defined – Generally optimised	Defined	Meets standard
Mineral Reserve Classification	Required, Generally payback x 1.5	Mineral-reserve achieved payback x 5	Exceeds standard
Plant and Infrastructure Costs			
Equipment Quotes	Detailed - Multiple quotes	Multiple quotations obtained for majority of equipment with only minor equipment priced from database	Meets standard
Civil / Structural	Calculated or Detailed - MTO & multiple quotes for supply costs, benchmarked hours to install	Detailed MTOs developed from 3D model of facilities. Multiple quotations for supply with hours for installation provided by contractors as part of the quotation.	Meets standard
Mechanical / Piping	Calculated or Detailed - MTO & multiple quotes – Benchmarked to	Detailed MTOs developed from equipment list, preliminary P&IDs	Meets standard

Aspect	Minimum Standard	Method Used	Comments
	similar plus hours to install data. Small bore piping may be factorised.	3D model of facilities. Multiple quotations for supply with hours for installation provided by contractors as part of the quotation. Small bore piping factored.	
Electrical / Instruments	Calculated - MTO & Hours with benchmarked or budget quotes	Detailed MTOs developed from electrical equipment list, preliminary P&IDs and 3D model of facilities. Multiple quotations for supply with hours for installation provided by contractors as part of the quotation.	Meets standard
Information Systems / Control Systems	Calculated - Mix of calculated and multiple quotes	Calculated based on assessment of requirements with quotation for systems.	Meets standard
Labour Rates	Budget Priced, by contractors or equivalent and benchmarked.	Labour rates provided by multiple contractors across all packages.	Meets standard
Labour Productivity	Calculated	Labour hours provided by multiple contractors across all packages which included contractors assessment of productivity.	Exceeds standard
Construction Equipment	Calculated - $ / Hr on Labour Rates – quoted or calculated for large cranes and special equipment costs	Construction equipment requirements and pricing detailed by contractors with only minor and small tools factored into the labour rates.	Exceeds standard
Indirect Costs			

Aspect	Minimum Standard	Method Used	Comments
Temporary Facilities	Calculated based on assessment of requirements	Priced by multiple contractors based on their assessment of the requirements, requirements and pricing for EPCM and other supervision assessed by the engineer.	Meets standard
Construction Support	Calculated based on assessment of requirements	Calculated by the construction contractors, and included in the quotation, or the engineer based on assessment of requirements	Meets standard
EPCM Services or Engineering	Calculated in detail – Benchmarks used to verify	Calculated in detail and reviewed against benchmarks.	Meets standard

Table 21.2: Basis of Deliverables and General Project Data Requirements

Aspect	Minimum Standard	Method Used	Comments
General Project			
Baseline Reports (Climate, Soils, Geotech, Hydrology, Wind, Wave, etc.)	Defined or Complete	Complete	Meets standard
Environment and Community Reports	Defined - Specific constraints, issues and commitments declared	ESIA complete and approved outlining Project constraints and commitments	Exceeds Standard
Project Scope Description	Defined	Defined	Meets standard
Integrated Project Execution Plan	Defined - Specific	Defined	Meets standard
Contracting Strategy – Implementation	Defined and generally optimised	Defined	Meets standard
Project Master Schedule – Implementation	Defined and resourced (site labour) to Level 4 and the critical path fully	Defined to level 4 including critical path detail. Resourcing not loaded	Schedule is not resource loaded, however the critical path is not

Aspect	Minimum Standard	Method Used	Comments
	detailed to activity Level 4. Resourced at Level 3 or lower.	into the schedule, however assessed by the contractors.	through the construction activities, rather around mine development, as such impact on accuracy is considered minor
Project Master Schedule – Commissioning and Ramp-up	Defined to Level 4 and the critical path fully detailed to activity Level 4	Defined to level 4.	Meets standard
Work Breakdown Structure	Defined to Level 4	Defined to level 4 across the whole project	Meets standard
Escalation Strategy	Defined and Detailed to source currency for individual items	Escalation excluded from the project	Escalation excluded
Foreign Exchange	Defined	Foreign exchange carried over into the financial model in native currency	Meets standard
Contingency Methodology	Detailed Calculation and Risk Analysis	Based on schedule risk assessment and quantitative risk assessment and then P_{80} used	Meets standard
Accuracy	Detailed Analysis – Monte Carlo	Monte Carlo simulation completed	Meets standard
Basis of Estimate and Methodology Statement	Complete	Complete	Meets standard
Engineering Deliverables			
Block Flow Diagrams	Complete	Complete	Meets standard
Process Flow Diagrams	Preliminary to Complete	Complete	Meets standard
P&ID's	Preliminary identifying major lines, piping materials and instrumentation requirements	Preliminary and detailed	Meets standard
Heat & Material Balances	Preliminary to Complete	Complete	Meets standard
Design Criteria	Complete	Complete	Meets standard

Aspect	Minimum Standard	Method Used	Comments
Control Narratives	Preliminary	Not started	
Overall Site Plan	Preliminary to Complete	Complete	Meets standard
Plot Plans	Preliminary to Complete	Complete	Meets standard
Process / Mechanical Equipment List	Preliminary to Complete	Complete	Meets standard
Electrical Equipment List	Preliminary to Complete	Complete	Meets standard
Specifications and Datasheets	Preliminary to Complete	Preliminary	Meets standard
GA Drawings by Facility / Area or 3D model	Preliminary to Complete	3D model developed	Meets standard
Mechanical / Piping Discipline Drawings	Preliminary – Major pipe routing	Major pipe routing in 3D model	Meets standard
Civil / Structural Discipline Drawings	Sketched to Preliminary (for major structures)	Developed from 3D model for structures	Meets standard
Earthworks Discipline Drawings	Preliminary earthworks designs	Preliminary design completed for all earthworks	Meets standard
Electrical Single Line Diagrams	Preliminary	Typical only	Not required for the Project due to its typical nature
Electrical Discipline Drawings	Started, major cable routing	Major cable routes considered in 3D model	Meets standard
Instrumentation & Control Discipline Drawings	Started	Not Started	Not required for the Project due to its typical nature
Process / System Capacity Simulations	Complete (as appropriate for level of complexity of project)	No assessment carried out	Not required for the Project due to its typical nature, availability of tailings filtration considered by the equipment vendor
Communications and Data Capture Systems	Started	Started	Meets standard
Information Systems Plan	Started	Considered	Only considered given typical nature of the Project

Aspect	Minimum Standard	Method Used	Comments
Spare Parts Listings	Started	Not Started	Factored from mechanical equipment cost, however recommendations provided by equipment vendors
Environmental Management	Defined	Defined	Meets standard
Cash Flow	Detailed	Assumed by cost area	Assumed cashflow is suitable for a typical project such as Bankan
Information Systems	Preliminary	Preliminary	Meets standard
Information Systems Plan	Preliminary	Preliminary by consultant familiar with country	Meets standard
Owners Deliverables			
Project Execution Plans and Procedures	Defined - Project Execution Plan	Defined	Meets standard
Operational Readiness Plan	Preliminary - included in Project Execution Plan or stand-alone document	Preliminary assessment	Meets standard
Permits and Approvals	Essentially complete with "Headline" Approval Document and Management Plans issued	Approvals in place other than exploitation permit which is submitted and being considered by government	Meets standard
Baseline Environmental Conditions	Complete (known basis)	Complete	Meets standard
HSEC Standards and Policies	Declared Policy and expanded to suit circumstances	Declare policy and management plans developed	Meets standard
Communications and Stakeholder Liaison	Preliminary / Complete	Complete	Meets standard
Human Resources Strategy	Preliminary	Started	Preliminary report for expatriate labour provided by human resources consultant.

Aspect	Minimum Standard	Method Used	Comments
Financing Plan and Strategy	Started – Potential Conditions Precedent Considered	Started	Meets standard
Marketing Plan and Strategy	Defined Plan and Offtake Negotiated (or forward pricing forecast considered in widely traded commodities)	Not required for gold project, long term pricing based on consensus	Meets standard
Purchasing Plan and Strategy	Defined Plan	Not defined	Not commenced as majority of purchasing will be carried out by EPCMs in the execution plan
Economic Modelling	Defined - Cash Flow Model with all cash flows (including financing and taxation), plus multiple scenario analysis and simulation	Detailed	Meets standard

21.1.3 Estimate Currency and Base Date

The capital cost estimate is in United States dollars (US$) at a base date is the 1st Quarter 2025.

The exchange rates in Table 21.3 have been used in the compilation of the capital cost estimate. All costs were carried through to the financial model in the native currency such that foreign exchange modelling can be carried out in the financial model.

Table 21.3: Foreign Exchange Rates

Currency	Exchange Rate	% of Capital Costs[1]
United States Dollar	1.000	92
Euro	0.900	0
Australian Dollar	1.550	4
South African Rand	18.0	4
Guinean Franc	8600	0

Notes:

1. Percentage of pre-production capital cost excluding pre-production mining

21.1.4 Contingency Estimate

In accordance with industry best practice a quantitative risk assessment (QRA) and schedule risk assessment (SRA) were completed to determine the capital cost and schedule risk profiles for the Project.

The QRA and SRA assessed the level of Project schedule and cost performance variability and risk to establish an appropriate level of contingency to be applied to the current schedule and cost estimates for this stage of the Project development.

Contingency is a provision for known project costs that will occur but cannot be defined in sufficient detail for estimating purposes due to the lack of complete, accurate and detailed information, as well as limited engineering which has been performed to date. The addition of contingency is required to determine the most likely cost of the Project. The project contingency does not cover scope changes, project exclusions or changes to the proposed execution strategy.

The schedule risk analysis results indicated a mean contingency of 3.0% of the Project duration and an accuracy for the 80% confidence interval of -2.4% to +3.5%.

A detailed Monte Carlo analysis was completed based on the outcomes of the QRA. Contingency has been included in the capital cost estimate based on the P_{80} probability estimate, which equates to 10.6% of the base estimate. The estimation of contingency excludes the pre-production mining cost.

21.1.5 Capital Cost Estimate

As summarised in Table 21.4 the total pre-production costs from the commencement of execution readiness (following completion of the DFS) through to the first processing of ore, which will be shortly followed by first production of gold have been estimated at US$463.0m, including a contingency of US$34.3m. The estimate includes all the infrastructure and services required to operate the Project, pre-production mining, project management, first fills and spares, owner's costs and execution readiness costs.

Table 21.4: Capital Cost Estimate

Area	US$m[1]
Mining	
Pre-production Open Pit Mining	42.9
Pre-production Underground Mining	62.8
Mining Infrastructure	5.1
Paste Plant	6.0
Construction Costs	
Earthworks	15.0
Process Plant	146.8
Non-Process Infrastructure	30.4
Plant Buildings	3.9
Accommodation Village	13.4
Tailings Storage Facility	18.3
Power Supply and Distribution	2.5
Construction Indirect Costs	31.3
Owners Costs	
Owners Project Management	14.2
First Fills & Spares	12.0
Vehicles & Cranage	7.8
Corporate Project Costs	9.6
Pre-Production Costs	6.6
Contingency	**34.3**
Total	**463.0**

Notes:

1. Totals may not compute due to rounding

The capital cost estimate presented excludes the following:

- Sunk costs including pre-FID costs outlined Section 21.1.6.
- Forward escalation from the estimate base date through to Project completion.
- Government levies and taxes.
- Working capital, sustaining capital and stay-in-business capital.
- Financing or other funding costs.

- Schedule acceleration costs, or schedule delays and associated costs such as those caused by labour disputes and/or force majeure.
- Exploration for further resources which may supplement the Project feedstock beyond the LOM production schedule identified in this study.

21.1.6 Execution Readiness Costs

The execution readiness costs outlined in Table 21.5 will be expended prior to FID from existing cash reserves and as such are not included in the capital costs for the project. The activities planned for the execution readiness phase are outlined in Section 24.1.5.

Table 21.5: Execution Readiness Costs

Area	US$m
Owners Team and Tendering Costs	2.9
Land Acquisition and Economic Resettlement Action Plan Implementation	5.9
Site Investigations to Support Detailed Design	1.9
Dewatering Bore Installation	1.1
Total	**11.8**

21.1.7 Sustaining and Deferred Capital

Various capital and sustaining costs will also be incurred during the operations phase, including:

- Underground development costs including decline and accessway development, main pump station development, ventilation, escape and drill drives, waste stripping and haulage, pump station service holes, vent raises and escape ladderways.
- Deferred capital cost for the refrigeration plant required for underground cooling, installed in three modules across years two to four of production.
- Deferred capital for the establishment for mining of the BC pit including the construction of the access road, creek diversion, flood mitigation bunds, surface water management drains and pollution control dams, dewatering bores and fencing.
- TSF expansion comprising:
  - Second lift in cell 1 (Year 2).
  - Development of cell 2 and closure of cell 1 (Year 3).
  - Second lift of cell 2 (Year 5).
  - Development of cell 3 and closure of cell 2 (Year 6/7).
  - Second lift of cell 2 (Year 9).
- General sustaining capital allowance over the life of the Project to replace equipment at the end of its useful life.

The life of mine sustaining and deferred capital estimate, detailed in Table 21.6, is US$164.3m.

Table 21.6: Life of Mine Sustaining and Deferred Capital Estimate

Area	US$m
Underground Development	91.7
Refrigeration Plant	4.6
BC Pit Establishment	5.7
TSF Expansion and Progressive Closure	58.5
General Sustaining Capital	3.9
Total	**164.3**

21.1.8 Closure Costs

Estimates have been made of mine closure costs as follows:

- Final TSF closure, US$3.8m (with progressive closure included in sustaining capital).
- Plant and infrastructure removal and demolition along with remediation of the process plant and infrastructure sites, US$28.0m.
- Mine closure, including closure of the underground mine, construction of abandonment bunds and removal and rehabilitation of WRDs, roads and other surface infrastructure, US$2.2 (with reprofiling of WRDs included in the mining cost estimate schedule).
- Closure planning and management, US$2.5m.

Salvage value of the plant and infrastructure has been included at the equivalent value as the cost of closure of the plant and infrastructure with the cashflow occurring at the completion of closure activities.

In addition, a program of post-closure monitoring over a period of up to five years following completion of closure, has been estimated at US$3.2m.

21.2 Operating Costs

The DFS operating cost estimate has been developed from first principles by category and cost type. This method is described in the subsequent sections.

The operating cost has been developed in line with a Class 3 estimate with a target accuracy of ±15%. The date of the estimate is the 1st quarter 2025, presented in US$. Table 21.7 presents the life of mine operating costs for the Project.

Table 21.7: Life of Mine Operating Costs

Area	LOM Cost US$m	Unit Costs	Unit Cost US$/oz
Open Pit Mining	604.3	US$4.97/t material mined	222.81
Underground Mining	442.1	US$57.39/t ore mined	163.00
Processing	873.7	US$17.08/t ore milled	322.13
Tailings Handling	174.6	US$4.54/t milled	64.38
General and Administration	164.8	US$14.3m per annum	60.77
Transport & Refining	22.9	US$8.45/oz	8.45
C1 Cash Costs	**2,282.4**		**841.55**
Royalties	390.6	6 % of Revenue	144.00
Sustaining Capital & Closure Costs	175.9	-	64.86
All-in Sustaining Costs	**2,849.9**		**1,050**

The basis for estimating these costs is outlined in the sections below.

21.2.1 Basis of Estimate and Methodology

The methodology for the development of the estimate is outlined in Table 21.8. The table shows the minimum standard adopted by the Qualified Person for a Class 3 estimate with a comparison to the standard applied in the DFS and outlines the reasoning behind evaluation of the estimate accuracy.

Table 21.8: Operating Cost Methodolgy

Aspect	Minimum Standard	Method Used	Comments
Typical Accuracy Range P_{10} to P_{90}	Based on known operations, ±5% to 10% or for new operations or novel technology 10 to 15%	±10%	Meets standard
Contingency on Operating	None unless special circumstances and only if defined	None	Meets standard
Basis of Estimate and Methodology Statement	Complete	Complete	Meets standard
Staff Levels	Detailed - Estimate	Detailed organisational chart developed	Meets standard
Consumables	Calculated - Estimate	Detailed calculations of reagents usage and consumables based on metallurgical test work were appropriate or based on information from equipment vendors	Meets standard
Maintenance	Calculated - Estimate	Calculated	Meets standard
Spares	Calculated - Estimate	Factored from mechanical and electrical equipment cost	Factoring of spares costs is appropriate for typical projects
Basis of Deliverables			
Labour Cost Rates	Detailed - calculations based on new contract basis	Expatriate cost rates based on Human Resources study from independent consultant, P_{50} used for majority, however P_{75} used for GM and UG mining staff. Local labour rates based on current employment contracts in place in country	Meets standard

Aspect	Minimum Standard	Method Used	Comments
Labour Burden Rates	Detailed - calculations based on new contract basis	Detailed first principles calculations	Meets standard
Labour Hours	Detailed calculations	Detailed calculation based on rosters	Meets standard
Labour Overheads / Management Costs	Detailed - calculations based on new contract basis	Detailed calculations	Meets standard
Power Costs – Fuel and Generating Costs	Detailed Calculations	Quotation from BOO power provider, including fuel usage, with fuel cost provided from fuel supplier	Meets standard
Water Costs (if applicable)	Detailed Calculations	Not required	Meets standard
Fuel Costs (Mobile Equipment)	Detailed - Quotes – firm, typical usage rates	Mining equipment usage from contractors estimates based on typical usage rates, typical usage rates used for other mobile equipment	Meets standard
Drill & Blast Costs	Detailed - Quotes - firm	Drill and blast cost from mining contractor submissions, explosives costs from multiple quotations in country	Meets standard
Supplies & Reagents	Detailed - calculations	Detailed calculations and quotations	Meets standard
Maintenance Materials	Detailed - calculations	Factored from mechanical and electrical equipment costs	Factoring of spares costs is appropriate for typical projects
Plant Hours	Detailed - calculations	Detailed	Meets standard
Transport & Logistics	Detailed - Calculations	Quotation from logistics provider	Meets standard
Other Operating Costs			

Aspect	Minimum Standard	Method Used	Comments
Business Systems (core and support systems, e-commerce, business management, office, etc)	Preliminary - detailed review	Preliminary	Meets standard
Training	Preliminary - detailed review	Calculated as a percentage of wages	Slightly below standard, however not a significant cost so not considered to be a significant impact on the accuracy
Ramp Ups including any loss of production from existing mine and plant	Preliminary - detailed review	Not included	Due to the typical nature of the Project and the contracting strategy it was assessed that this would be minimal
Insurances	Preliminary - written quotes from Broker, not Insurance Policy	Quotations from multiple brokers	Meets standard
Escalation	Preliminary - detailed calculations. Based on individual commodities.	Excluded	Escalation excluded
Foreign Exchange	Identify equipment and commodities to be imported, basis, values and likely currency. Provide forecasts of changes. Quantify to Level 3.	All costs estimated in native currency and carried through into the financial model	Meets standard

21.2.2 Estimate Breakdown

The operating cost estimate has been built from first principles using the following categories:

- Mining.
- Labour.
- Power.
- Reagents.
- Consumables.
- Mobile equipment.
- Maintenance.
- Transport and logistics.
- General and administration.
- Tailings handling.

The estimation of each of these categories is outlined in the following sections.

21.2.3 Mining Costs

21.2.3.1 Open Pit Mining Cost

Requests for budget pricing was issued for the open pit mining scope of work to four suitable mining contracting groups as part of the DFS. Each of the groups were requested to provide estimates for the full life of mine based on the following supply from PDI:

- Diesel fuel, with consumption estimated by the contractor, including fuel storage and dispensing infrastructure.
- Explosives, with the consumption estimated by the contractor, and explosives storage magazine.
- Heavy vehicle washdown bay and associated infrastructure.
- Mining infrastructure area, with services, for the establishment of the contractor's infrastructure.

Three submissions were received and reviewed with all being acceptably complete and in line with the stated requirements. The spread of LOM open pit mining costs was relatively close across the submissions varying by less than 5% once normalised for assumptions. On this basis the contractor submission that was closest to the average was selected for use in the DFS.

Following finalisation of the pit designs and mining schedule, the selected contractor was requested to update their submission based on the updated information and this pricing, and consumable (diesel, explosives etc.) usage was used as the basis for the cost estimate in the DFS.

In addition to the contractor's information the following aspects of mining costs were estimated from first principles:

- Grade control drilling and sampling costs.
- Assaying costs, which are included in overall laboratory costs.
- Owner's management and technical personnel, which are included in the overall labour estimate.

The open pit mining costs can be broken down into following categories:

- Unit rates, which are fixed rates applied to the physical quantities over the LOM.
- Variable unit rates, which are rates that vary over the LOM such as load and haul costs and are applied to physical quantities over the LOM.
- Fixed contractor costs, which vary over the LOM but are not applied to physical quantities directly. These also include some minor costs, such as pit dewatering, clearing and grubbing, topsoil management, road construction and WRD rehabilitation, which are determined by the overall mine plan rather than direct physical mining quantities.
- Owner supplied consumption, including diesel and explosives, which are driven by the physical quantities over the LOM.

These costs are outlined in Table 21.9, which provides the open pit unit mining rates, Figure 21.1 showing the fixed contractor costs over the LOM, Table 21.10 and Figure 21.2 which provide the fixed inputs into the owner supplied consumption and the variation in consumption over the LOM respectively.

Table 21.9: Open Pit Mining Unit Rates

Cost	Units	Rate
Pre-split	US$/m	16.90
Ore Re-handle to Crusher (FEL)	US$/t ore	1.15
Ore Stockpile Re-handle (Truck & FEL)	USD/BCM ore	3.35
GBE Waste Rehandle to ROM (Construction)	US$/t	1.50
UG Waste Rehandled to ROM (Construction)	US$/t	1.08
UG Waste Rehandled to Waste Dump	US$/t	1.08
UG Ore Rehandled to ROM Pad	US$/t	1.50
Grade Control Drilling	US$/m	35.25
Drill & Blast – Production	US$/t mined	0.07 – 0.42
Load & Haul – Ore to ROM	US$/t ore	1.59 – 4.42
Load & Haul – Ore to Stockpile[1]	US$/t ore	0.49 – 6.46
Load and Haul – Waste	US$/t waste	1.32 – 2.18

Notes:

1. Based on P_{10} and P_{90} due to high proportion of fixed costs and variable tonnage to stockpile

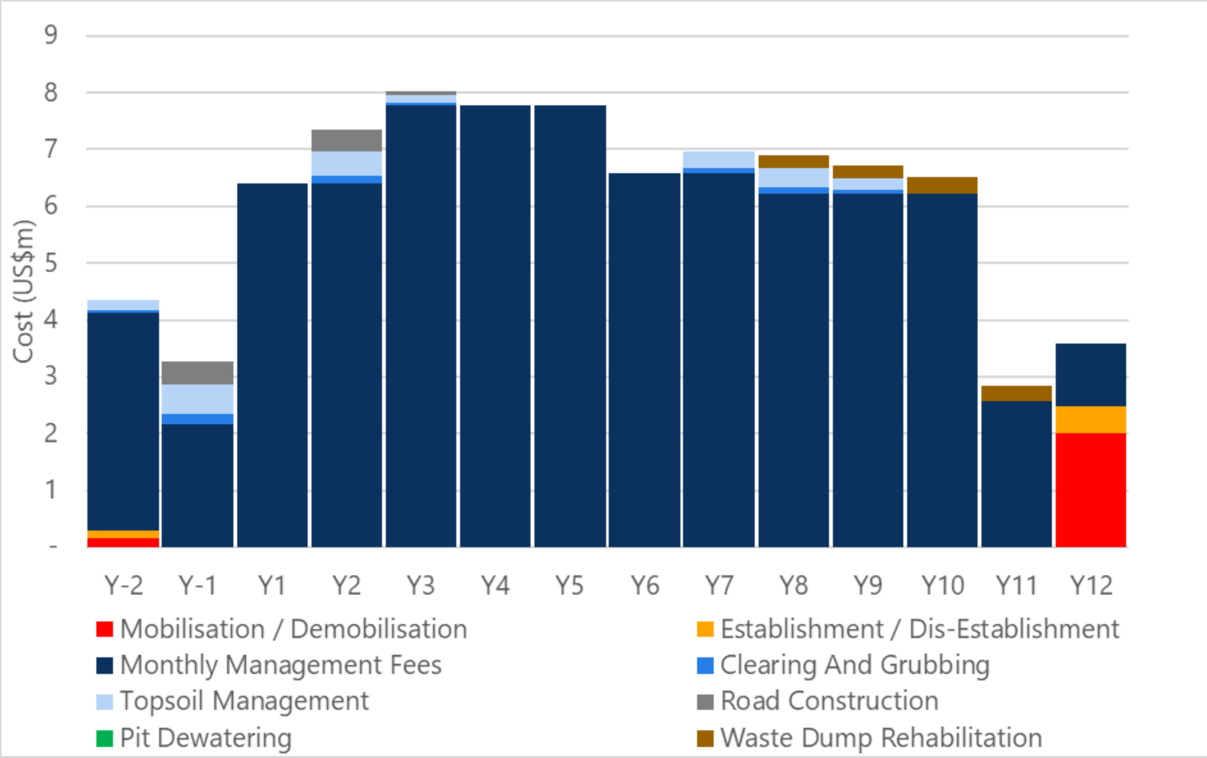

Figure 21.1: Fixed Open Pit Contractor Costs over LOM

Table 21.10: Open Pit Owner Supplied Rates

Aspect	Units	Rate
Explosives Usage	kg/m blasthole	1.10
Bulk Explosives Costs	US$/kg	1.15
Presplit Explosive Cost	US$/kg	10.00
Initiating Explosive Cost	% of Bulk Explosive Cost	12 – 17
Fixed Explosives Magazine Cost	US$/month	25,000
Diesel Cost	US$/L	1.10

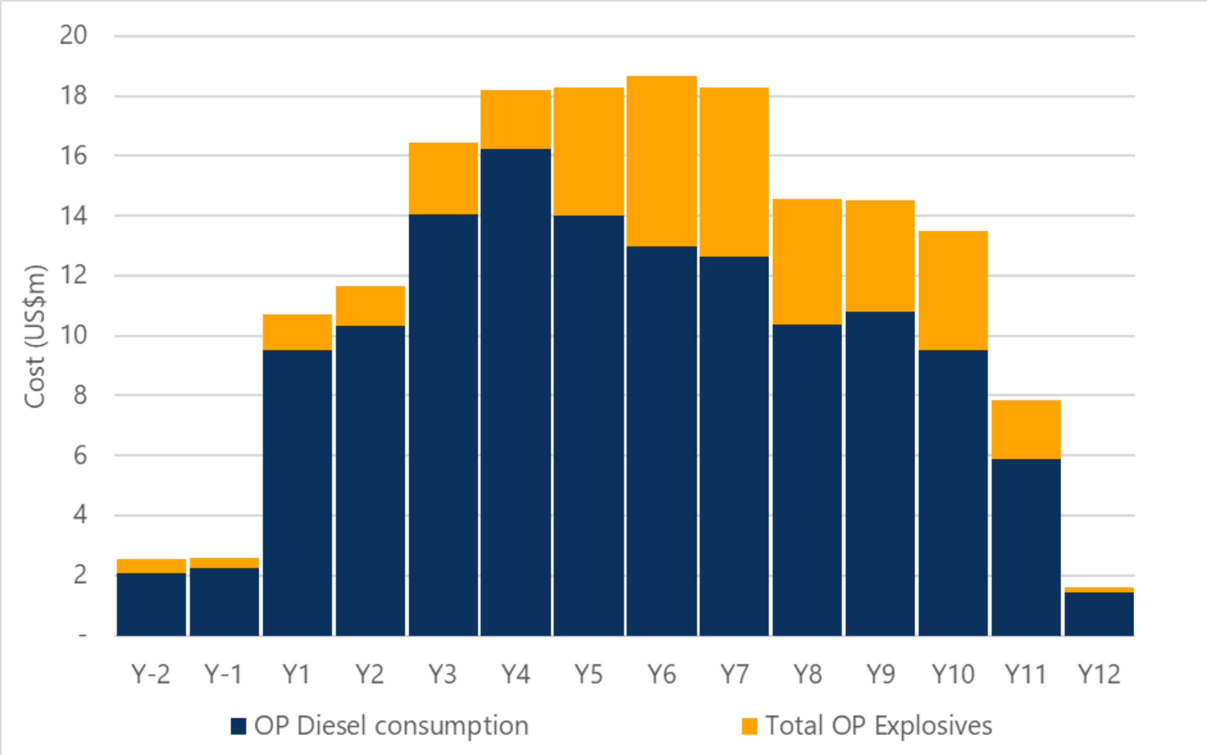

Figure 21.2: Open Pit Diesel and Explosives Usage over LOM

21.2.3.2 Underground Mining Costs

Following the completion of the underground mine design, Barminco (part of the Perenti Group of which Orelogy is also part) were provided with the design to provide cost and consumption inputs on a contract mining basis for the estimation of underground mining costs. As with the open pit mining costs, it was assumed that the following would be supplied by PDI:

- Diesel fuel, with consumption estimated by the contractor, including fuel storage and dispensing infrastructure.
- Explosives, with the consumption estimated by the contractor, and explosives storage magazine.
- Heavy vehicle washdown bay and associated infrastructure.
- Mining infrastructure area, with services, for the establishment of the contractor's infrastructure.

In addition to the contractor's information the following aspects of mining costs were estimated from first principles:

- Grade control drilling and sampling costs.
- Assaying costs, which are included in overall laboratory costs.
- Owner's management and technical personnel, which are included in the overall labour estimate.

The underground mining costs can be broken down into following categories:

- Variable unit rates, which are rates that vary over the LOM such as decline and lateral development, vertical development, production charging and grade control drilling which vary by the location in the mine and are applied to physical quantities over the LOM.
- Fixed contractor costs, which vary over the LOM but are not applied to physical quantities directly. These also include some minor costs, such as ground support, ventilation and backfill required which are determined by the overall mine design rather than direct physical mining quantities.
- Owner supplied consumption, including diesel, explosives, binder for paste and power, which are driven by the quantities estimated over the LOM.

These costs are outlined in Table 21.11, which provides the underground mining rates, Figure 21.3 showing the fixed contractor costs over the LOM and Table 21.12 and Figure 21.4 which provide the fixed inputs into the owner the variation in owners supplied consumption over the LOM respectively.

Table 21.11: Underground Mining Unit Rates

Cost	Units	Rate
Decline & Lateral Development[1]	US$/lateral m	2,022 – 2,573
Vertical Development (incl. Production Drilling)[1]	US$/vertical m	7,626 – 44,804
Production Charging[1]	US$/t material	0.53 – 1.22
Materials Handling[1]	US$/t material	2.33 – 9.75
Grade Control Drilling	US$/drill m	82

Notes:

1. Range presented as P_{10} to P_{90} due to impact of fixed costs.

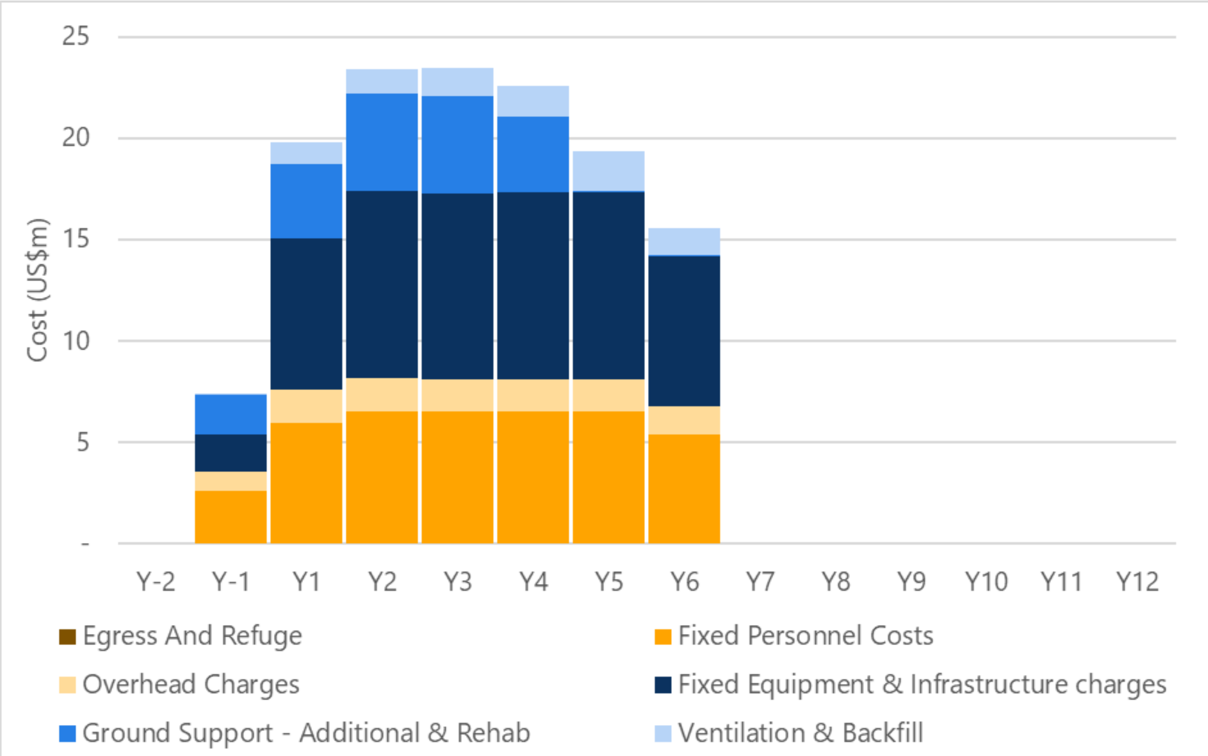

Figure 21.3: Fixed Underground Contractor Costs over LOM

Table 21.12: Underground Owner Supplied Rates

Aspect	Units	Rate
Diesel Cost	US$/L	1.10
Binder Cost	US$/t	244.19
Power Cost (average over LOM)	US$/MWh	225

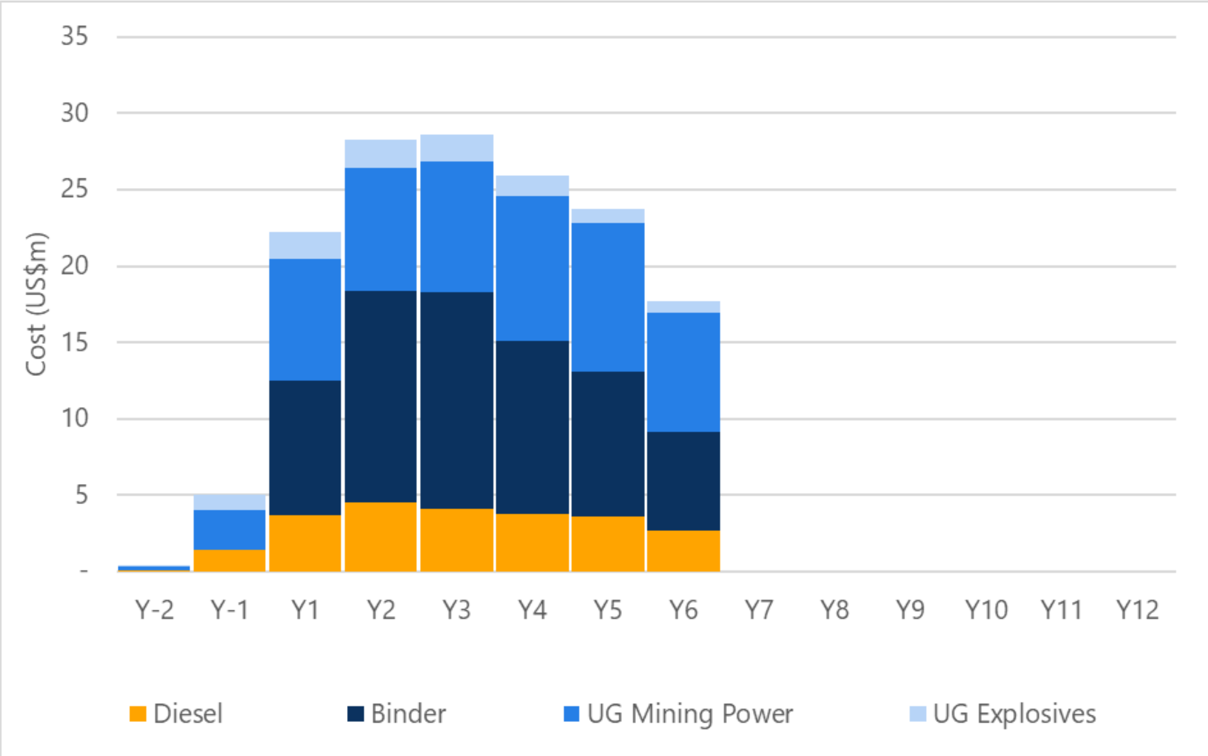

Figure 21.4: Underground Owner Supplied Diesel and Explosives Costs over LOM

Other costs associated with the operation of the paste plant such as labour, power and maintenance are estimated as part of the processing plant costs and then assigned in the financial model to underground mining cots.

21.2.4 Labour Costs

Labour requirements were developed from the organisational charts and rosters developed for the Project.

Labour costs were calculated from first principles based on the following sources:

- Average net salaries by job level for Guinean personnel were provided by PDI and applied by matching appropriate job levels for each operational role.
- For expatriate positions, a human resources consultant was engaged to provide recommended net salary costs for all anticipated expatriate positions (international and regional).
- Guinean tax rates were applied to estimate the company cost of employment for all roles.
- The annualised salaries for each position are inclusive of.
  - Salary.
  - Employee contributions to the National Social Security fund (CNSS).
  - Employer contributions to the National Social Security fund (CNSS).
  - Income tax (RTS).

For international expatriate roles and critical roles where there is expected scarce regional labour with the required skillset, P_{75} salary estimates were applied. For all other expatriate labour P_{50} salary estimates were applied.

Expatriates and Guinean fly-in/fly-out staff will work 6 weeks on and 3 weeks off roster where international flights are assumed to be business class, and in-continent flights assumed to be economy class. In addition, medical expenses and communication expenses (where applicable) were estimated by role but captured as a general and administration cost.

Local based staff will work eight hours per day or shift with four shift panels required to provide 24/7 coverage.

The total labour costs, excluding all contract labour, is provided in Table 21.13 based on operations with both open pit and underground mining underway.

Table 21.13: Labour Costs

Area	Units	Cost
Mining	US$/a	2,171,214
Processing	US$/a	1,371,654
Administration and Environmental	US$/a	2,517,321

21.2.5 Power

21.2.5.1 Power Demand

The fixed power for the operation has been defined by the electrical load list, typical utilisation for electrical equipment items and plant area availability.

The SAG and ball mill have been excluded from the fixed power consumption estimate because different ore types have been demonstrated to require varying amounts of power to achieve the design comminution circuit grind product size at design tonnage. Grinding power is estimated by applying the specific energy to the tonnes of the respective lithology per period. The lithologies and respective specific energies are presented below:

- Saprolitic (clay like), 5.5 kWh/t.
- Lateritic/Saprock (weathered but friable), 12.8 kWh/t.
- Fresh (mafic, tonalite, etc), 29.2 kWh/t.

21.2.5.2 Power Cost

Power costs for the site power station have been taken from tenders for the installation of a hybrid HFO power station on a BOO basis with power purchase agreement (PPA). The power cost has been calculated based on fixed and variable charges, HFO price and degree of renewable energy penetration, as outlined in Table 21.14.

Table 21.14: Power Station Power Costs

Aspect	Units	Cost
Fixed Capacity Charge – HFO Power Station	US$/a	12,917,040
Variable Maintenance Charge – HFO Power Station	US$/MWh	11.86
Diesel Usage – HFO Power Station	L/a	688,516
HFO Usage – HFO Power Station	kg/kWh	0.20
HFO Price	US$/kg	0.84
Diesel Price	US$/L	1.10
Fixed Capacity Chage – Solar PV/BESS	US$/a	5,831,000
Renewable Penetration	%	30.40
Average Power Cost	US$/MWh	225

21.2.6 Reagents

Reagent costs are generally considered variable with consumption varying due to plant throughput. However, when the operation is consistently operating at design capacity, the reagent cost effectively becomes fixed. The exception to this is lime where it was demonstrated that weathered ores (laterite, saprolite) consumed significantly more lime than fresh ores (mafic, tonalite).

Reagent costs are based on quoted prices delivered Conakry with transport costs estimated and added as appropriate.

The total reagent requirements for the processing plant are summarised in Table 21.15 for the average operating blend that includes 20% weathered and lime consumption consistent with that blend.

Table 21.15: Reagent Cost and Consumption

Reagent	Packaging	US$/t	Consumption Rate	Annual Consumption	Source
Cyanide	Briquettes, bulk box	2,150	0.62 kg/t	2,777	Database price, assumed CIF Conakry
Quicklime	Bulk tanker, local	503	0.68 kg/t	3,042	Local supplier, inclusive of transport
Activated Carbon	Bulka bag	3,050	-	180	CIF Conakry quote
Copper Sulphate	Bulka bag	3,000	0.10 kg/t	450	CIF Conakry quote
Sodium Metabisulfite	Bulka bag	480	0.56 kg/t	2,520	CIF Conakry quote
Flocculant	Bulka bag	2,200	30 g/t	135	CIF Conakry quote
Coagulant	Bulka bag	3,390	15 g/t	68	Database price, assumed CIF Conakry
Binder	Bulk supply, local	243	111.9 kg/m^3 paste	49,637	Local supplier, inclusive of transport
Hydrochloric acid 32%	IBC	405	90 g/t	417	Database price, assumed CIF Conakry
Sodium hydroxide	Pearl	880	30 g/t	89	Database price, assumed CIF Conakry
Sulphamic acid	IBC	3,343	-	22	Database price, assumed CIF Conakry
Antiscalant	IBC	4,396	-	89	Database price, assumed CIF Conakry
Borax	Bag, 25 kg	2,240	-	9	Database price, assumed CIF Conakry
Silica	Bag, 25 kg	906	-	9	Database price, assumed CIF Conakry
Soda ash	Bag, 25 kg	1,107	-	9	Database price, assumed CIF Conakry
Sodium nitrate	Bag, 25 kg	2,430	-	9	Database price, assumed CIF Conakry
Process Diesel (m^3)	Bulk tanker	1.10/L	-	2,001	Project Pricing

21.2.7 Consumables

Specialised consumables for mechanical equipment have been included as presented in Table 21.16. These items are considered variable costs as their replacement is variable to the throughput of the associated equipment. The consumable wear rates associated with the comminution circuit were estimated separately for fresh ore, and the softest blend considered by OMC. The consumption rates presented are consistent with a life of mine blend which is dominated by 80% fresh ore and only 20% of the softer ore types.

Table 21.16: Consumables

Consumable	$ / Unit	Unit	Annual Consumption	Basis of Estimate
SAG mill media	1,157	t	1,251	Database price, CIF Conakry
Ball mill media	1,127	t	3,340	Database price, CIF Conakry
Primary crusher fixed jaw	13,225	set	7.36	Supplier quote, plus 15% freight
Primary crusher moving jaw	10,925	set	5.5	Supplier quote, plus 15% freight
Sizer teeth/studs	45,384	set	2.0	Supplier quote, plus 15% freight
SAG mill liner & lifter set	1,901,295	set	2.3	Supplier quote, plus 15% freight
Ball mill liner & lifter set	1,063,290	set	0.7	Supplier quote, plus 15% freight
Pebble crusher liner set	6,325	set	8.5	Supplier quote, plus 15% freight
Tailings filter cloth set	88,424	set	12	Supplier quote, plus 15% freight
Stainless steel wool	805	kg	405	Database price, inclusive of freight
Crucibles	1,977	each	26	Database price, inclusive of freight
Oxygen sparge tip	990	each	12	Supplier quote, inclusive of freight

21.2.8 Mobile Equipment

The maintenance and consumables required for mobile equipment has been estimated and considered a fixed cost. Maintenance for this equipment has been included within the maintenance costs and diesel which is detailed in the consumables.

To estimate the maintenance per equipment, benchmarked annual estimates for tyres, drivetrain, brakes, lubricants and general maintenance costs were applied for each vehicle identified. The sum of these costs is captured in the maintenance cost estimate.

Similarly, for each piece of equipment identified, average daily runtime and diesel consumption rates were applied to estimate the mobile equipment diesel demand, which is presented as a fixed cost in the consumables.

21.2.9 Maintenance

Maintenance costs are considered fixed annual costs and include the cost of spare parts (other than those included as consumables), maintenance consumables and maintenance contracts to maintain the processing plan and non-process infrastructure.

Maintenance contracts include the labour, transport and messing and accommodation costs for shutdown and specialist maintenance activities.

Direct labour charges for routine maintenance are included in labour costs.

Maintenance costs were determined as a by the application of a factor to mechanical and electrical equipment capital costs, excluding installation. The factors were based on analysis of the equipment located in each area and typical factors.

Shutdown contract labour was calculated based on activity and frequency of required shutdowns.

21.2.10 Transport and Logistics

Transport charges to site have been constructed to include the following for a twenty-foot container equivalent (TFE) transporting a 25-tonne payload:

- Terminal handling and port services charges
- Customs clearance
- Hazardous goods surcharges, if appropriate
- Local shipping line charges
- Customs inspections
- Restitution costs of empty container to shipping line
- Customs IT tax.
- Forwarding fee
- DDI services
- Transport from port to site via road, including the return of empty containers.

This has been estimated at US$4,965 per tonne of transported goods.

Freight costs for consumables (crusher jaws and teeth, mill liners, filter cloth sets, etc) have been estimated at 15% of the supply cost.

Where applicable, the estimated transport charge has been applied to the goods supplied CIF to Conakry.

21.2.11 General and Administration

General and administration costs are considered fixed costs. A summary of the general and administration costs included are presented in Table 21.17.

Table 21.17: General and Administration

Category	US$ / Annum
Corporate (Guinean Only)	
Corporate Off-site Costs	989,191
Site Administration	447,610
Accounting	211,481
Government Charges	197,032
Insurances	
Property Damage and Business Interruption	1,250,000
Product/Public Liability	200,000
Other	448,676
Technical Consultants/Specialist Software	
Mining	475,000
Processing	265,806
Admin/Environmental	467,097
Contract Services	
Laboratory	1,800,472
Other Administration Contracts	308,359
Messing and Accommodation	1,290,629
Bussing	548,256
Domestic Air Travel	936,000
Medical	596,520
Equipment and Consumables	74,258
Security	1,618,668
HR and Personnel	1,152,496
Community Relations	645,533
General	423,173
Total per Annum	**14,346,256**

21.2.12 Tailings Handling

To determine the cost of tailings rehandle and placement in the TSF, as well as transport of tailings to for feed to the paste plant a first principles estimate was developed equipment and personnel requirements based on the transport and placement requirements, cycle times and operating efficiencies. Based on this the following equipment, and associated leasing costs, operating staff costs and fuel consumption, as shown in Table 21.18, was developed.

Table 21.18: Tailings Rehandle Fleet and Costs to TSF

Equipment	Number	Lease Rate US$/day	Operator Rate US$/day	Fuel Consumption L/hr
FEL CAT 980 or equivalent	1	150	20	40
ADT Truck, CAT 745 or equivalent	8	150	20	40
Grader, CAT 140G or equivalent	1	150	20	25
Dozer, CAT D6 LGP or equivalent	1	140	20	30
Water Cart	1	75	20	20

A similar process for determining fleet requirements etc. was undertaken for delivery of filtered tailings to the paste plant.

Using these costs for both the rehandle to the TSF and paste plant, tailings rehandle costs per day were developed, and from these daily costs unit rates, shown in Table 21.19, were back calculated for use in the cost estimation.

Table 21.19: Tailings Rehandle Unit Rates

Tailings Destination	Rehandle Cost US$/dry t
Tailings Storage Facility	3.60
Paste Plant	1.25

In addition to these costs additional owners labour was added for the management of the tailings rehandle and TSF operations and sustaining capital is included for TSF expansion and closure.

22 ECONOMIC ANALYSIS

The financial evaluation of the Project has been undertaken using a discounted cash flow (DCF) analysis in US$ (real Q1 2025 dollars). The evaluation includes only cash flows from the Project from FID and excludes potential cash flows from exploration activities or other assets held by PDI. Cashflows, net present value (NPV) and internal rate of return (IRR) for the Project have been calculated over the 12-year operational period.

The following key economic assumptions apply to the base case:

- Discount rate of 5%, comparable to other gold project assessments, applied to cashflows at the end of each period.
- NPV has been calculated at the Project commitment date, or FID, currently anticipated to be the beginning of the 2nd quarter of 2026.
- Project funding entirely through equity with no accounting for uplift that may result from any component of debt or other financing. However, it is possible that the Project will be at least partly funded through debt or other financing.

22.1 Key Assumptions

Physical assumptions are based on the LOM production schedule presented Section 16. Capital, sustaining capital, operating and closure costs are as per Section 21. Other key financial assumptions are set out in Table 22.1

Table 22.1: Key Financial Model Assumptions

	Units	Value
Gold Price	US$/oz	2,400
Discount Rate	%	5.0
Government Royalty	% of Revenue	5.0
Local Development Contribution	% of Revenue	1.0
Selling Costs	US$/oz	8.45
Corporate Tax Rate	%	30.0
Import Duties[1]	%	6.5
Import Charges/Taxes[1]	%	2.5
Withholding Tax[2]	%	15.0
Sunk Costs for Depreciation	US$m	142.0

	Units	Value
Working Capital Assumptions		
Payment on Shipment of Gold	%	100
Capital Expenses	month	1
Operating Expenses (excl. Labour)	month	1
Labour and Royalties	month	0
Tax	month	1
Diesel Fuel Price	US$/L	1.10
HFO Fuel Price	US$/kg	0.84

Notes:

1. Import duties/taxes apply to spares, supplies and consumables imported into Guinea and not supplies from Guinean suppliers, this is only applied to operating spares, supplies and consumables and it is assumed that an exemption will be granted by way of a "mining list" for the capital development phase for the duties only.
2. Withholding tax applies to services provided in Guinea by non-Guinean consultants and has been applied to consultant costs included in the operating cost estimate.

As outlined in Section 19, the gold price of US$2,400/oz is based on the median long term real consensus pricing from more than 30 institutions as of May 2025 which is notably lower than the May 2025 average spot price of approximately US$3,300/oz.

The tax and duties regime for the Project will ultimately be agreed upon with the Government during the negotiation of the *Convention de Base*, or mining convention, which may negotiate a more favourable tax treatment to support the development of the Project and the associated creation of government revenues, jobs and other social and economic benefits.

For the DFS, no negotiation outcome has been assumed on company tax, which are based on the full mining company tax rate of 30% with no tax holidays, or import duties which are assumed to be a combined 9% between import duties, charges and taxes or withholding tax on services, however it has been assumed that negotiations will be successful to secure an exemption from the fuel levy imposed on mining operations to provide for an assumed fuel price of US$1.10 per litre.

Prior spending on exploration and studies has not been assumed to be tax deductible at this stage and depreciation is calculated using a unit of production basis. Tax treatment is, therefore, conservative compared to what might be negotiated.

Foreign exchange rates in the DFS and the financial model are as per those provided in Table 21.3.

22.2 Key Financial Outcomes

The financial analysis was developed based on the LOM production schedule for the NEB open pit, NEB underground and BC open pit.

Production averages approximately 236,000 oz annually over 11.6 years at an AISC of US$1,050/oz. Financial metrics for the Project are robust, based on the consensus gold price of US$2,400/oz, with a post-tax $NPV_{5\%}$ of US$1,445m, IRR of 46% and a payback period of 1.8 years.

Financial outcomes improve significantly if the spot gold price of US$3,300/oz is adopted, with post-tax $NPV_{5\%}$ of US$2,571m, IRR of 74% and a payback period of 1.1 years.

The key Project outcomes for the Project are provided in Table 22.2.

Table 22.2: Key Project Outcomes

	Units	Base Case US$2,400/oz	Spot Price US$3,300/oz
Production			
Mine Life	years	11 years and 6 months	
Total Gold Production	koz	2,712	
Average Gold Production	koz/a	236	
Costs			
Pre-Production Capital Costs	US$m	463.0	
Sustaining Capital Costs	US$m	164.3	
Mine Closure Costs (excluding salvage)	US$m	39.6	
C1 Cash Costs	US$/oz	842	
All-in Sustaining Costs (AISC)	US$/oz	1,050	1,104
Financial			
Pre-Tax $NPV_{5\%}$	US$m	2,051	3,638
Pre-Tax IRR	%	57	90
Pre-Tax Payback Period	Years	1.5	1.0
Post-Tax $NPV_{5\%}$	US$m	1,445	2,571
Post-Tax IRR	%	46	74
Post-Tax Payback Period	Years	1.8	1.1

Annual production (Figure 22.1), operating costs (AISC) (Figure 22.2) and project cashflows (Figure 22.3) are presented graphically below and the annual production and cashflow information is provided in detail in Table 22.3.

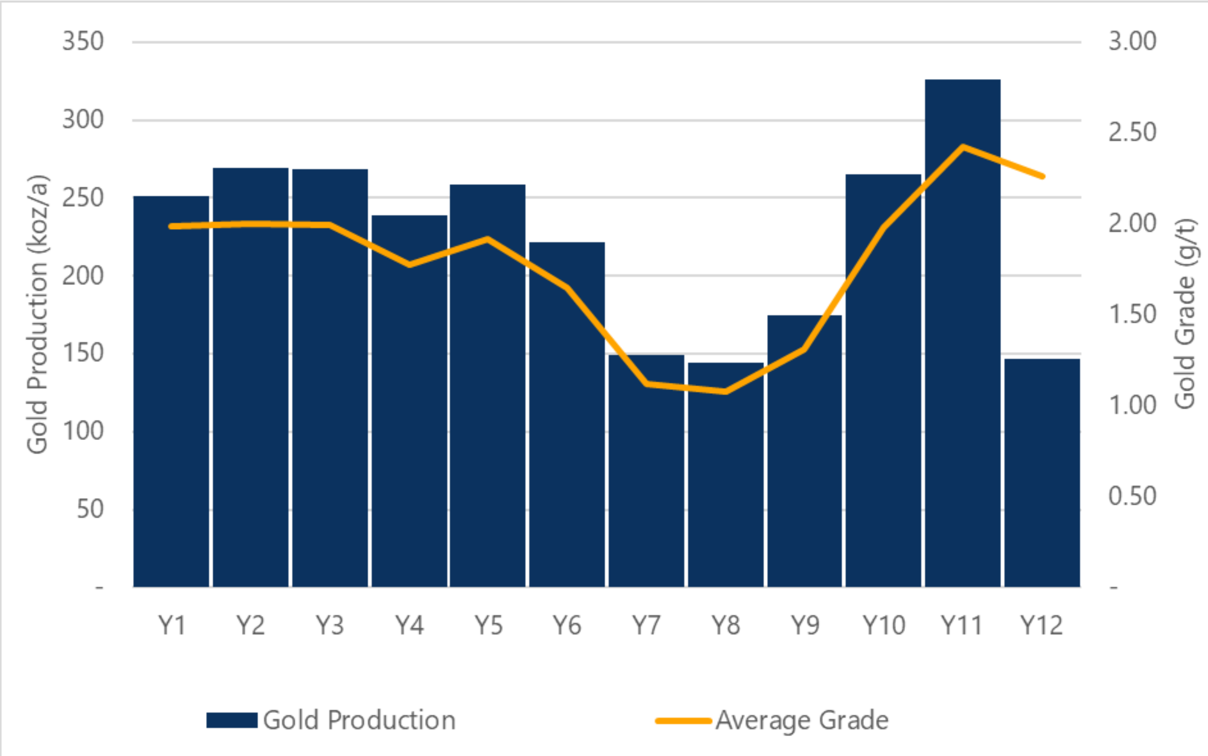

Figure 22.1: Gold Production and Grade

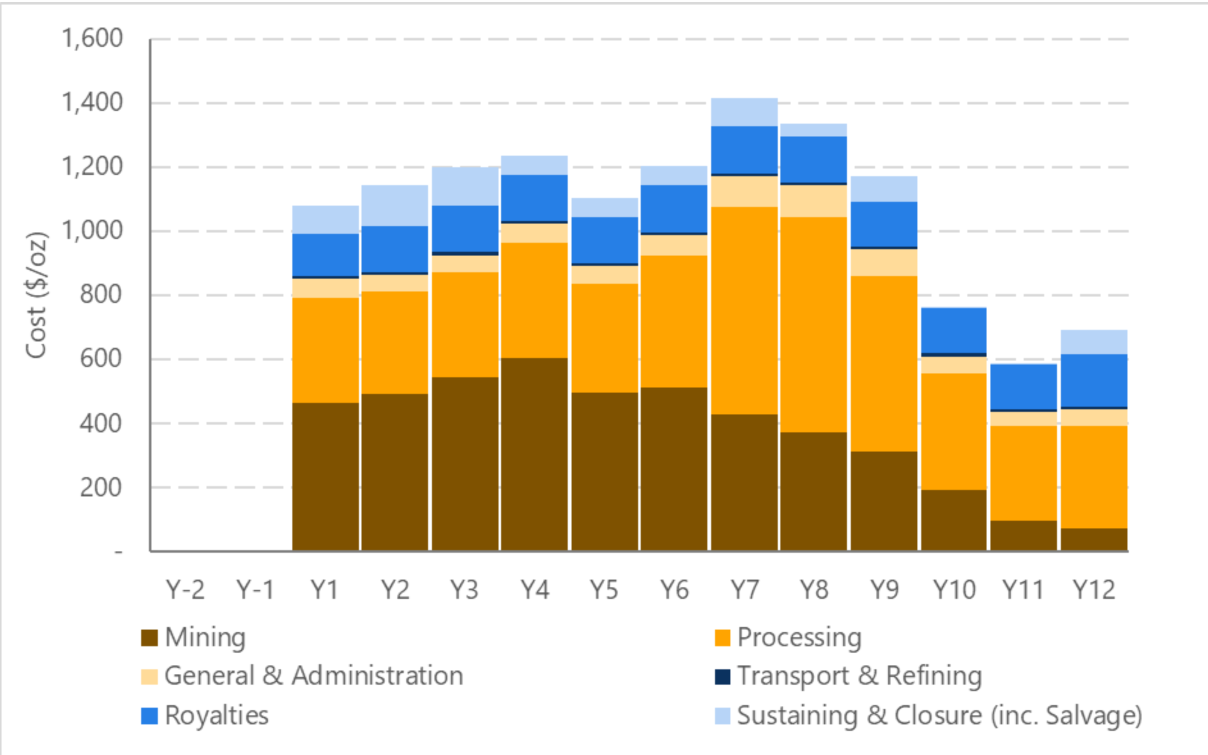

Figure 22.2: All-in Sustaining Cost per Oz

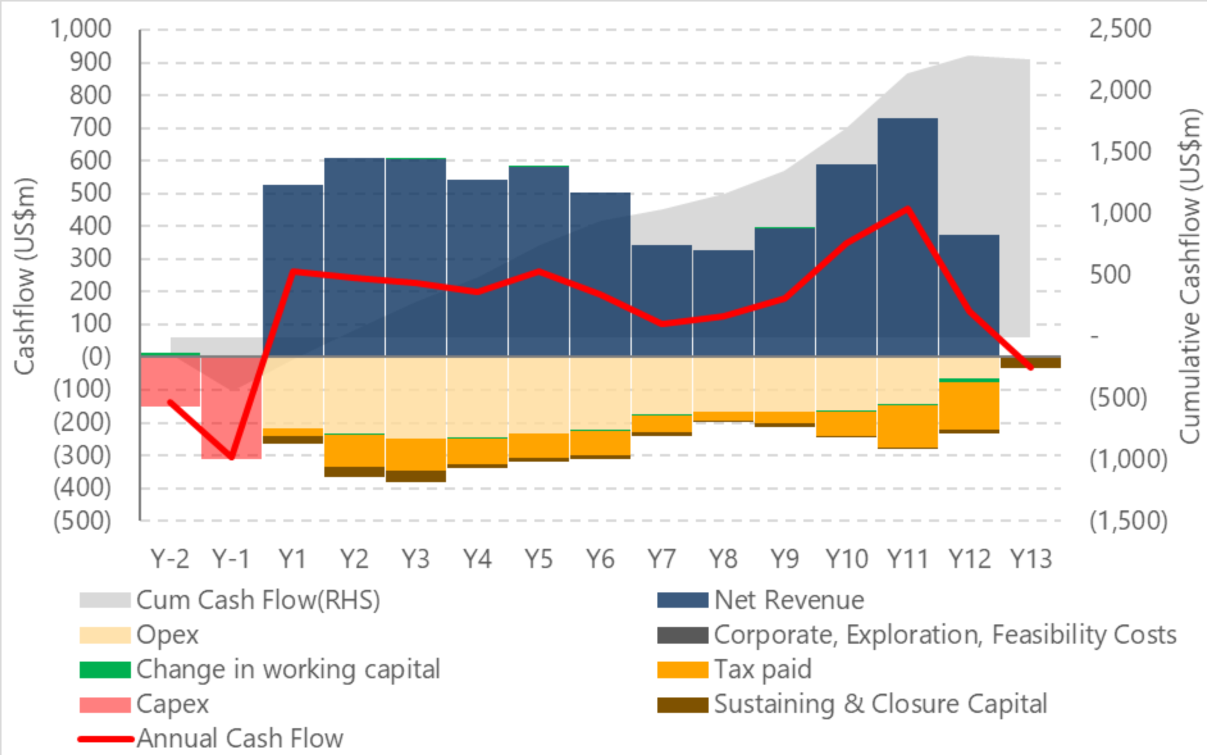

Figure 22.3: Project Cash Flow

Table 22.3: Detailed LOM Production and Cashflow

Unit	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14
Production																	
Mill Feed, Mt	51.39			4.22	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	2.17		
Head Grate, g/t	1.77			1.99	2.00	1.99	1.77	1.92	1.65	1.12	1.08	1.31	1.98	2.43	2.26		
Au Recovery (%)	92.8%			93.0%	93.0%	92.9%	93.0%	93.2%	92.9%	92.1%	92.1%	92.2%	92.6%	92.8%	92.7%		
Production, koz	2,712.2			251.0	269.0	268.1	238.5	258.4	221.7	149.0	143.9	174.9	265.0	325.8	146.8		
Cashflows (USD'000)																	
Gold Sales Revenue	6,509.2			559.0	649.1	644.3	575.5	618.4	534.6	364.9	346.0	416.6	627.1	775.9	397.9		
Capital Expenditure	(463.0)	(152.0)	(311.0)														
Mining	(1,046.4)			(116.8)	(132.8)	(146.4)	(143.7)	(128.7)	(113.5)	(63.6)	(53.6)	(54.3)	(51.0)	(31.1)	(10.8)		
Processing	(1,048.3)			(82.4)	(85.2)	(87.4)	(85.8)	(87.5)	(91.4)	(96.6)	(96.5)	(96.1)	(96.3)	(96.1)	(47.0)		
G & A	(164.8)			(14.3)	(14.3)	(14.3)	(14.3)	(14.3)	(14.3)	(14.3)	(14.3)	(14.3)	(14.3)	(14.3)	(7.2)		
Transport & Refining	(22.9)			(2.0)	(2.3)	(2.3)	(2.0)	(2.2)	(1.9)	(1.3)	(1.2)	(1.5)	(2.2)	(2.7)	(1.4)		
Royalties	(390.6)			(33.5)	(38.9)	(38.7)	(34.5)	(37.1)	(32.1)	(21.9)	(20.8)	(25.0)	(37.6)	(46.6)	(23.9)		
Sustaining & Closure	(203.9)			(21.7)	(34.3)	(32.5)	(13.8)	(14.9)	(13.7)	(13.0)	(6.0)	(13.5)	(0.3)	(0.3)	(11.1)	(28.7)	
Salvage Value	28.0															28.0	
Δ Working Capital	(0.0)	15.1	6.4	(0.2)	(1.3)	1.4	(2.2)	0.0	(3.2)	(2.2)	(0.8)	2.2	(2.4)	(1.6)	(8.3)	(3.3)	
Net VAT Paid	(0.0)			(2.6)	(0.1)	(0.1)	0.3	(0.1)	0.3	0.4	0.1	(0.3)	0.1	0.1	1.4	0.2	0.1
Pre-Tax Free Cashflow	**3,197.4**	**(136.9)**	**(304.6)**	**285.9**	**339.9**	**324.0**	**279.2**	**333.7**	**264.9**	**152.4**	**152.9**	**213.8**	**423.0**	**583.2**	**289.6**	**(3.7)**	**0.1**
Tax	(909.8)	-	-	(26.8)	(96.9)	(97.9)	(78.3)	(73.2)	(74.1)	(49.5)	(26.8)	(35.6)	(74.0)	(128.9)	(147.7)	-	-
Post-Tax Free Cashflow	**2,287.6**	**(136.9)**	**(304.6)**	**259.1**	**243.0**	**226.1**	**200.9**	**260.5**	**190.8**	**102.9**	**126.1**	**178.2**	**349.0**	**454.3**	**141.9**	**(3.7)**	**0.1**

22.3 Sensitivity Analysis

The sensitivity of the post-tax $NPV_{5\%}$ to changes in key assumptions are shown in Figure 22.4. As is typical for gold projects, the Project is most sensitive to changes in revenue linked assumptions such as gold price, grade and processing recovery (shown at ±5% below), followed by operating costs and capital costs.

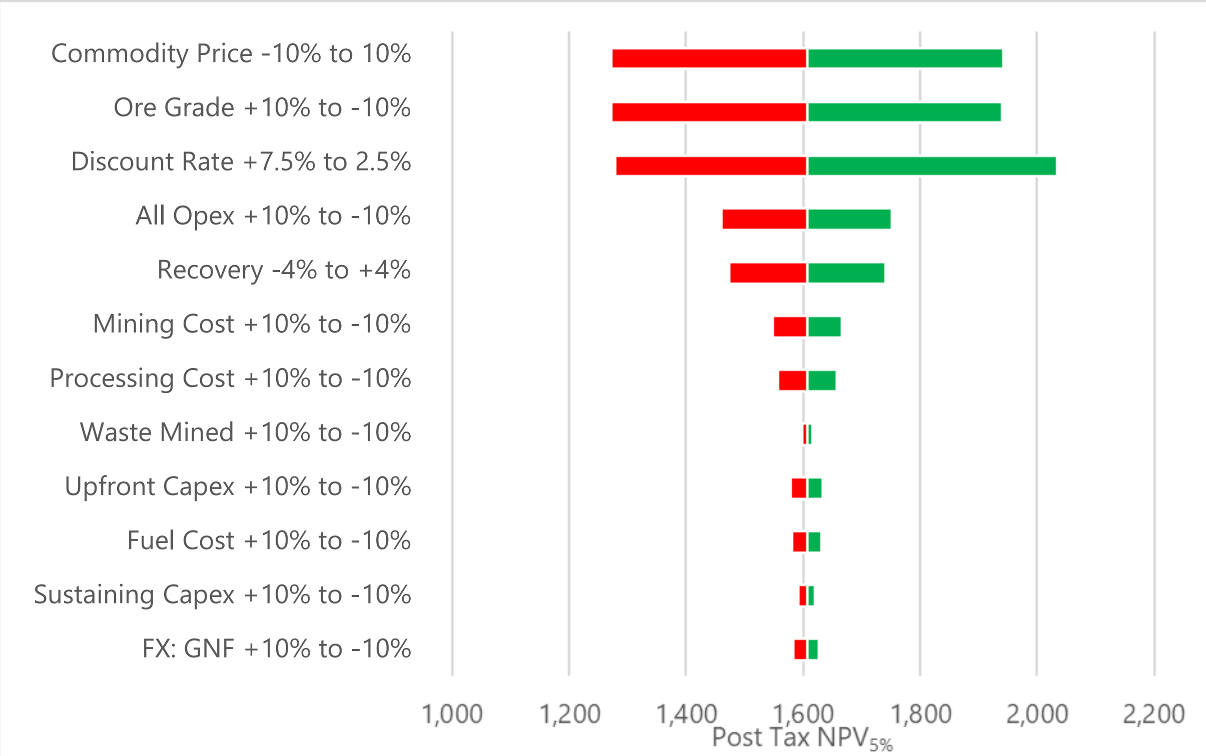

Figure 22.4: Post Tax $NPV_{5\%}$ Sensitivities

This sensitivity analysis demonstrates clearly that the Project is robust when considering typical variability in changes to key assumptions. The key sensitivities are to those assumptions impacting on revenue, such as gold price and ore grade, and cost of funding, indicated by the discount rate.

The Project is relatively insensitive to estimated capital and operating costs.

22.4 Funding Requirement and Strategy

The DFS confirms that the Project has strong financial fundamentals. The upfront capital requirement is estimated at US$463.0m. To secure this funding, the Company intends to pursue a combination of equity and debt which could include other alternative financing such as streams and royalties, with the aim of managing risk while optimising financial resources.

To support this strategy, the Company is working with Terrafranca Capital Partners (Terrafranca), a specialist financial advisory firm, which provides consulting services in relation to the Project's financing. Terrafranca's role includes:

- Conducting market sounding to assess appetite for gold projects in Guinea.
- Advising on benchmarking potential financing structures and financiers against PDI's objectives.

- Assisting, where appropriate, in soliciting expressions of interest from potential lenders.

Initial informal engagement with a broad range of potential financiers began in early 2023. This has included commercial banks, African and international development banks, debt funds, private equity firms, and royalty/streaming companies. Early feedback has been positive, and the Company has continued to provide updates through ongoing discussions with multiple interested parties. Following completion of the DFS, the Company will move to formal engagement with financiers to finalise the Project's financing structure.

PDI has a strong track record in raising equity capital, having secured approximately A$160m since 2023 to advance the Project. The Company is backed by a robust shareholder base, including large institutional investors and strategic mining groups, many of whom have a history of funding successful gold developments. With a current market capitalisation of approximately A$1,100m, the Company is well positioned to support the equity portion of the Project's funding requirements.

The Company believes there are reasonable grounds to expect that the required funding for the Project will be available when needed. These grounds include:

- The DFS demonstrates that the Project is economically robust.
- Feedback from early engagement with potential financiers has been positive.
- Continued availability of equity and debt financing for high-quality gold projects, supported by ongoing discussions.
- Proven track record of raising equity funding as needed to advance the Project.
- No debt and a current market capitalisation of approximately A$1,100m;
- Clean corporate and capital structure with the Company's Guinean subsidiaries holding the two exploration permits and the associated exploitation permit applications underlying the Project.

These factors are expected to be highly attractive to potential financiers.

The ability of the company to fund its future requirements will depend on, amongst other things, debt and equity market conditions at the time. Funding via additional equity issues may be dilutive to the Company's existing shareholders and, if available, debt financing will be subject to the Company agreeing to certain debt covenants and other terms and conditions.

23 ADJACENT PROPERTIES

23.1 Gold Mining in Guinea

Artisanal gold mining in Guinea dates to the 3rd Century, mainly in the Siguiri Basin in the northeast of the country; production reached its zenith during the Mali Empire (1235 CE – 1670 CE). Colonial era mining started in 1909 with dredging operations on the Tinkisso River, to the southwest of Siguiri, and continued intermittently until Guinea won its independence in 1958. Total French production in this period is estimated at 70 t of gold (Bolay, 2016).

Modern exploration dates from the 1930s with geological reconnaissance and mapping at 1:500,000 scale by the Bureau Minier de la France d'Outre-Mer (BUMIFOM); between 1943 and 1958 shaft sinking, trenching and limited diamond core drilling was conducted in the Kiniéro area (Bolay, 2016).

Between 1961 and 1963, a Russian geological expedition mapped the Siguiri Basin at a 1:200,000 scale and reviewed the dredging potential of the Tinkisso River and other river basins (Barradas, Siguiri mine, Guinea, 2016).

In 1988, the Société Aurifère de Guinée (SAG), (UMEX 25.5%, Pancontinental Mining Ltd 25.5% and the Government of Guinea 49%) restarted placer mining (Barradas, Siguiri mine, Guinea, 2016) at the Siguiri Mine; production ceased in 1992. The Project was acquired by Ashanti Goldfield Co. Ltd (now AngloGold Ashanti) and production recommenced open pit mining and heap leaching processing. An 8.5 Mtpa CIP process plant was commissioned in 2005 and was subsequently upgraded to a 12 Mtpa CIL plant. As at 31 December 2022, Mineral Reserves were Proved plus Probable of 90.88 Mt @ 0.80 g/t Au for 2.34 Moz Au (AngloGold Ashanti, 2025).

23.2 Regional Gold Mining

23.2.1 Artisanal Gold Mining

There are approximately 200,000 to 300,000 artisanal miners active in Guinea, with production estimated to be in the 6 t to 8 t per annum range (MDO Data Online Inc., 2025). In the Kouroussa region, which hosts Bankan, approximately 50,000 miners are active. Mining comprises shallow shafts and diggings in weathered ore, with largely manual crushing and gravity separation processes (Doumbouya, Dessertine, Vinches, & Cerceau, 2024).

23.2.2 Kouroussa Gold Project

From 1985 to 1987, the Niandan Mining Association (NMA), a 70% Al Baraka (Saudi Arabia) and 30% Bureau de Recherche Géologiques et Minières, France (BRGM) partnership re-evaluated the mining potential of a 30,000km² concession that included Kouroussa. In 1996, Pacific Comox Resources Ltd (PCR), a Canadian junior mining company, was granted a Prospecting Licence for the northern part of Kouroussa. In 1997, PCR granted Bright Star Ventures Ltd (BSV) the option to acquire a 100% interest in the property, subject to the 15% carried interest in favour of the Guinean Government. BSV completed a soil geochemistry and detailed sampling program over the Koekoe and Sodyanfé artisanal mining areas. Several anomalies were identified by BSV but they could not meet the financial commitment to PCR and the property reverted back to PCR (Macfarlane & Love, 2008).

In 2002, the exploration permit was taken up by Guinea Golden Mines (GGM) and optioned out to Cassidy Gold Corp who acquired a group of tenements surrounding Kouroussa, including the KoeKoe and Kinkine deposits. Hummingbird Resources plc acquired the project from Cassidy Gold Corp in September 2020 (Mining Technology, 2020) and achieved first gold pour in June 2023 (International Mining, 2023).

In late 2018, the Kaninko area was highlighted by PDI during its terrain-scale assessment of the Siguiri Basin, which PDI had previously identified as being both highly prospective for gold mineralisation and underexplored.

23.2.3 Kiniéro Gold Project

The Kiniéro gold deposits, which is located 25 km southeast of the Bankan project area, were discovered in 1943. Various operators subsequently completed mapping, geochemistry, ground geophysics, trenches, shaft sinking and drilling. SEMAFO Inc. commenced open pit mining in 2002, with ore being processed in a 700 ktpa CIL plant. The mine produced 418 Koz Au during its 12 year operational history and was placed on care-and-maintenance in early 2014 with the mining and exploration licences revoked in 2014 (Tucker, et al., 2022).

Sycamore Mining Ltd was awarded the Kiniéro project in 2020 via a competitive tender process. Kiniéro was acquired by the Robex Resources in 2022 as part of the business combination with Sycamore Mining Ltd (Tucker, et al., 2022).

Robex is planning to restart the mine with a new 3.0 Mtpa CIL Processing Plant (potentially being upgraded to 4.1 Mtpa). The mine is targeting to produce 90,000 oz pa over a mine life of 9.5 years based on Mineral Reserves of 27.7 Mt @ 1.09 g/t Au for 968,000 oz. Total Indicated and Inferred Mineral Resources are 88.7 Mt @ 1.10 g/t Au for 3.1 Moz (Szebor, et al., 2025).

23.3 Reliance on Information from Adjacent Properties

The information presented above relating to the historical operations in Guinea and adjacent properties have been drawn from publicly available information and information disclosed by the current or previous owners or operators of the properties and the Qualified Persons have not verified the information regarding these properties. It should also be noted that the information presented is not necessarily indicative of the mineralisation of the property that is the subject of this report.

There has been no reliance on, or use of, information from adjacent properties in the preparation of the Report.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Implementation

A project implementation plan has been developed to define the proposed methodology that will be employed to successfully deliver the Project in line with the budget and schedule reducing the risk of adverse cost or schedule outcomes.

This implementation plan has also been used as a basis for the development of capital cost estimates for the Project.

24.1.1 Project Phases

The Project will be developed across a number of key phases, which will somewhat overlap, including:

- Execution readiness, which will commence soon after the completion of the DFS and continue until project final investment decision (FID) by the PDI board and work is ready to commence on site. This phase will focus on establishment of the owners team, engagement of key engineering consultants and completion of front-end engineering, tendering of early-works construction contracts and implementation of the economic resettlement action plan.
- Project delivery, which will commence following FID and will involve the pre-production mining along with the procurement, contracting and construction of all the facilities.
- Operational readiness, which will include the staged build-up of the operational workforce, implementation of key operational systems, developing operational procedures for operations and maintenance and training of the workforce. This phase will proceed in parallel with the Project delivery.
- Completions and commissioning, which will involve a staged approach to the confirmation of construction, testing, start-up and handover to operations of the facilities, which will commence in parallel with the final stages of construction and culminate in the commencement of production.

24.1.2 Project Management Approach

Execution of the Project will be carried out by a team of appropriately qualified and experienced personnel from the internal resources of PDI, an appointed project management consultant (PMC) group, a group of EPCM contractors and other external engineers as required.

Engagement of an experienced PMC as part of an integrated project management team (IPMT) to support project execution allows PDI to leverage established systems and procedures, as well as seasoned and experienced teams, rather than having to establish these from scratch. PDI will lead on matters relating to operational safety, environmental management, permitting, legal, tendering key operational contracts, mining, geology, government and community relations, funding and payments, operational readiness and exploration and the PMC will focus on the delivery of the capital aspects of the Project (project management), including tendering of engineering contracts, construction contracts, contract administration, project controls (cost and schedule), construction quality and safety, engineering co-ordination, standardisation, design sign-off on behalf of PDI and inter-contractual interface issues.

A project steering committee will be also formed with the purpose of advising the PDI executive and Board of Directors on Project related matters and providing confidence in the Project execution progress and outcomes. In addition, the project steering committee will provide high level guidance and support to IPMT to assist in achieving the Project goals.

The make-up of the project steering committee will include a member of the PDI board, PDI's Chief Operating Officer as well as independent project steering committee members.

The Project Director and Project Manager will present to the project steering committee on safety, progress, expenditure and forecast and risks. The project steering committee will meet prior to board meetings with the aim of reviewing the reporting from the project team and providing feedback to the board and the Project.

The project owners team organisation, including key contractors, is shown in Figure 24.1.

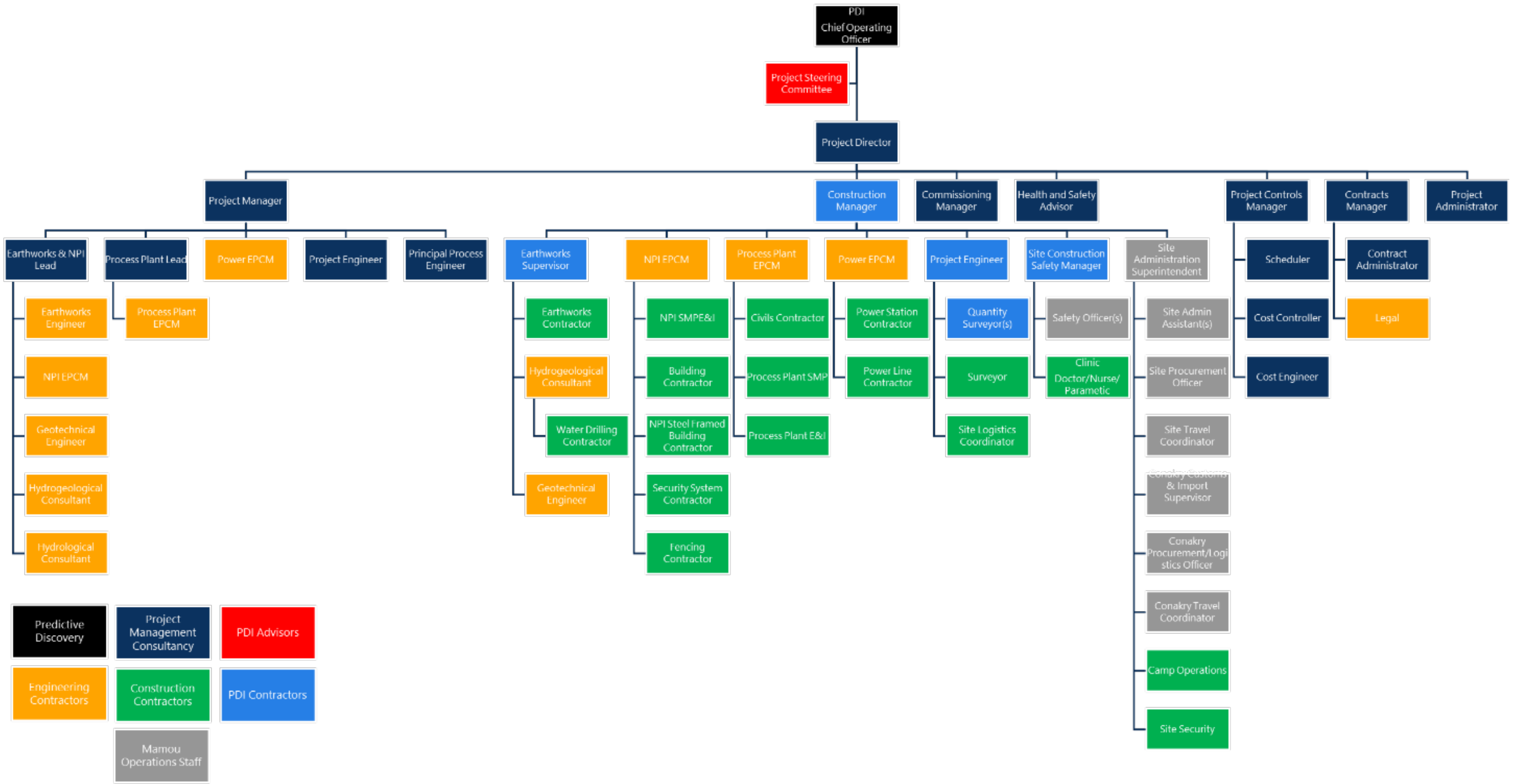

Figure 24.1: Owners Team Organisation Structure

24.1.3 Engineering Approach

The engineering for the Project will be contracted to suitably qualified and experienced groups working in their area of expertise. This will consist of three key EPCM contractors who will carry out engineering design, procurement and construction management on behalf of PDI and a number of specialist engineering consultants who will provide specialist design input with the procurement and construction management carried out by the IPMT. The EPCM contracts will include:

- Process plant EPCM, which includes the processing plant and process plant buildings.
- NPI EPCM, with includes the administration and other support buildings, workshop, warehouse, accommodation village, site water management systems, PDI supplied mining infrastructure and mine power distribution.
- Power supply and distribution EPCM, responsible for the power station, solar PV and power distribution around site.

The engineering contracts will include:

- Geotechnical engineer, responsible for the site geotechnical investigations and basis of design as well as the TSF design and construction QAQC.
- Earthworks engineer, responsible for the earthworks design.
- Hydrogeological consultant, responsible for the design of the dewatering and monitoring bores, including supervision of drilling and pump testing.
- Hydrological consultant, responsible for the specification and design of the surface water management infrastructure.
- Mining consultant, responsible for the early stage detailed pit and underground mine designs, tendering of the mining contracts and supporting of the pre-production mining.
- Mine geotechnical consultant, responsible for providing mining geotechnical advice to the mining engineers and operations.

24.1.4 Construction Contracting Strategy

Construction contracts will generally be let where possible to provide similar services across the entirety of the Project. In some instances, smaller contract packages have been selected to be broken out of this strategy in an effort to identify works that can be allocated to Guinean contractors. The construction contract packages will include:

- Earthworks contractor, covering all site earthworks including the access roads, accommodation village and process plant earthworks, surface water management infrastructure and the TSF. Tendered and managed directly by the IPMT.
- Civils contractor, covering all concrete works on the Project including the supply of the batch plant and wet concrete. Tendered and managed by the process plant EPCM.
- Process plant structural, mechanical and piping contractor, covering the fabrication of all structural steel, platework and piping materials and installation of these as well as free-issued

mechanical equipment and specialty items. Tendered and managed by the process plant EPCM.

- Process plant electrical and instrumentation contractor, covering the procurement of all electrical bulks and the installation of all free issued electrical equipment, cabling and instrumentation. Tendered and managed by the process plant EPCM.
- NPI structural, mechanical, piping, electrical and instrumentation contractor, being a smaller, preferably, Guinean contractor to carry out the minor construction works around interfaces between construction contracts and installation of minor NPI supply equipment. Tendered and managed by the NPI EPCM.
- Building contractor, covering the design, fabrication and installation of the transportable buildings on site including the administration buildings and also the accommodation village. Tendered and managed by the NPI EPCM.
- Steel framed building contractor, covering the design, fabrication and installation of the steel framed buildings on site such as the warehouse and workshop. Tendered and managed by the NPI EPCM.
- Power station contractor, designing, supplying and installing the power station on a build-own-operate (BOO) basis. Tendered and managed by the power station and distribution EPCM.
- Power line contractor, supplying and installing the overhead powerlines on site. Tendered and managed by the power station and distribution EPCM.
- Logistics contractor, who will carry out all project logistics other than specialised transport (such as mining fleet) across all aspects of the Project. Tendered and managed by the process plant EPCM.

In addition, a number of smaller contractors will be tendered and managed by the IPMT, including:

- Fencing contractor.
- Security system contractor, for the specification, supply and installation supervision of the electronic security system.
- Water bore drilling contractor.
- Surveying contractor.
- Site security contractor, who will carry over from construction into operations.
- Medical services contractor, who will carry over from construction into operations.
- Camp operations contractor, who will carry over from construction into operations.

24.1.5 Execution Readiness Works

Key to successful delivery of a project is a suitable level of preparedness for execution. In addition, being able to carry out key engineering, tendering and other workstream in parallel with project funding activities also de-risks the project execution schedule through creation of additional float and limiting the number of critical or near critical path activities.

For the Project an execution readiness program of approximately nine months is planned. The key workstreams in this period include:

- Engagement of the PMC and formation of the IPMT.
- Preparation of a detailed project management plan and supporting plans such as safety management plan, construction management plan, procurement management plan, cost management plan etc.
- Implementation of project management systems.
- Following grant of the exploitation permits for the project, implementation of the economic resettlement action plan to gain full land access.
- Tendering and award of the EPCM and engineering contracts.
- Front end engineering for the key aspects of the Project, including:
  - Process plant, to enable finalisation of the plant layout and tendering of long lead equipment ready for award following funding completion.
  - NPI, focussed on the design and procurement of the dewatering bore systems, accommodation village (including award of the contract for the design and supply of the accommodation village to allow commencement of fabrication), underground mine power supply infrastructure (to allow fabrication and installation to coincide with commencement of underground mining) and fuel supply and distribution systems.
  - Bulk earthworks, such that the bulk earthworks can commence upon funding completion.
  - Mining, such that mining of the GBE pit can commence and the underground mining contract can be awarded on funding completion.
  - Power station, such that the power station BOO contract can be awarded as soon as possible to allow ordering of the HFO generating.
- Site investigations and early works to support the final design, including:
  - Grade control drilling of the GBE pit, to allow mining to proceed without being slowed by grade control.
  - Further process plant and TSF geotechnical site investigations.
  - Additional underground mine geotechnical drilling, particularly in the raise bore area, portal and early stopes.
  - Dewatering and monitoring bore drilling, to allow commencement of dewatering early in the mining sequence.
- Tendering, award and mobilisation of the bulk earthworks contract, which includes the mining of the GBE pit, such that the contractor can commence work on FID.

- Construction readiness activities such as finalising the early works construction insurance, tendering of the site security contract and tendering of the medical services contract.

24.1.6 Operational Readiness

Operational readiness and ramp-up of pre-production labour is key to successful start-up of the Project. On that basis and taking into account the early commencement of mining operations, a significant portion of the proposed operational workforce is engaged well prior to commencement of operations. Figure 24.2 shows the build up in workforce, excluding contractors, prior to the commencement of production.

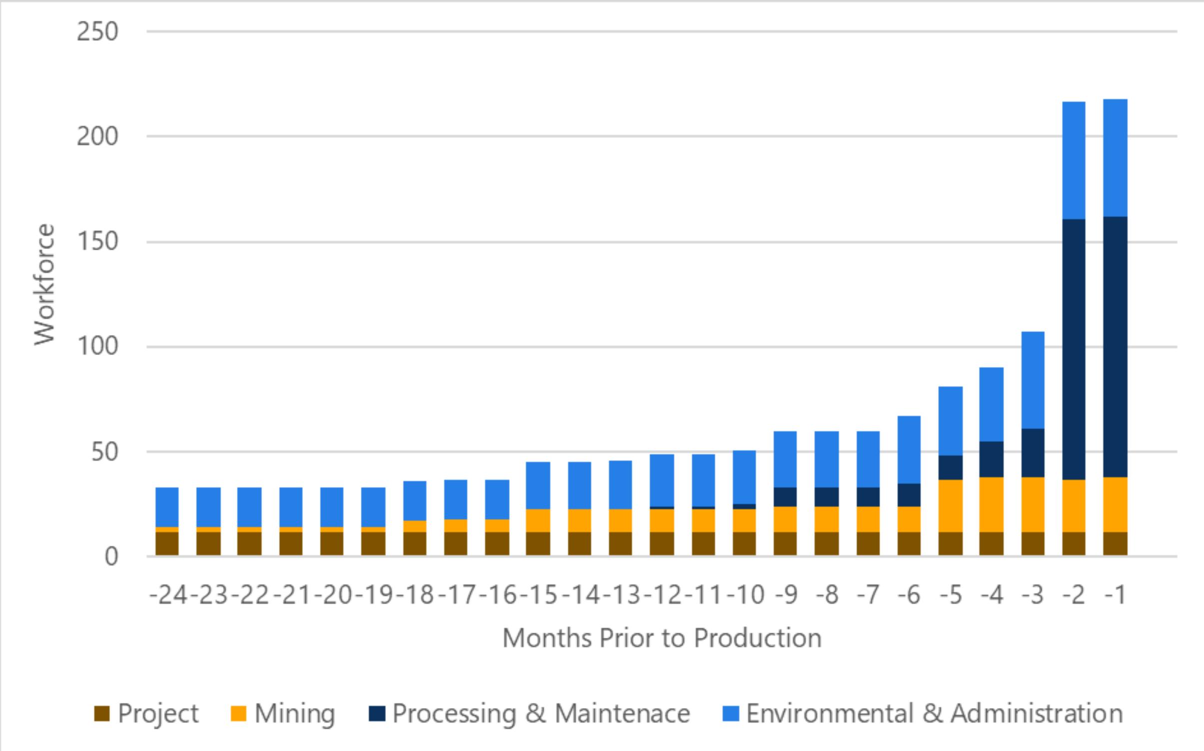

Figure 24.2: Pre-Production Workforce Ramp-up

In addition to the workforce build-up and the operational readiness tasks they will undertake, additional operational readiness activities that will be undertaken will include:

- Significant mining engineering and mine geotechnical support for the early phases of mining, particularly with regard to underground mining operations.
- Engagement of an experienced Operational Readiness Manager to carry out the planning and coordination of operational readiness activities in support of the General Manager.
- Implementation of the enterprise resources planning system will be undertaken by specialist consultants.
- Assistance with setting up the asset register, stockholding and warehouse management systems and bringing all the required spares and consumables into the warehouse ready for use.

- Specialist contractors to work with the maintenance planners to set up maintenance safety procedures, maintenance monitoring plans and procedures, preventative maintenance procedures, and preliminary shutdown plans for regular shutdown maintenance (e.g. mill relining).
- Assistance to the site process plant operations team to set up Project specific work practices and training modules.
- General training courses for mine site operations for all personnel as required.

24.1.7 Completions and Commissioning

Completions and commissioning will be carried out across a number of phases and will have key deliverables and responsibilities across each phase. The phases will include:

- C0 – Construction equipment installed. Aimed at verifying that the construction of the facility is substantially complete and installed. Responsibility of the construction contractor.
- C1 – Construction verification. Aimed at completing all as-built drawings (as required) and verifying against the design that all installation is fully complete. Responsibility of the construction contractor.
- C2 – Construction completion. Full sign off of all inputs/output and electrical connections, all circuits locked out and all piping isolated leading. All regulatory notifications completed and issuance of the construction completion certificate. Responsibility of the EPCM engineer.
- C3 – Dry commissioning. Facility verified and tested and ready for introduction of fluids. Responsibility of the EPCM engineer.
- C4 – Water commissioning. All systems tested with water or other suitable fluid and ready for introduction of ore and chemicals. Testing of sub-systems and systems and calibration and tuning of controls as possible. Responsibility of the EPCM engineer.
- C5 – Process commissioning. Commencement of processing ore to recovery gold and achieving relatively continuous operation. Production of first product. Responsibility of the owners team.
- C6 – Ramp-up. Bringing the facility up to design processing rate and production specified in performance tests. Responsibility of the operations team.

All facilities delivered as part of the Project will follow the same workflow, however some aspects may not require all phases before hand-over.

24.1.8 Implementation Schedule

An execution schedule has been developed for the Project's delivery using Oracle Primavera P6. From the release of the DFS, it is anticipated to take approximately nine months for PDI to reach FID and commencement of full development of the Project, which is anticipated in the second quarter of 2026. In parallel with this PDI will secure the Project’s exploitation permits and once granted deliver the economic resettlement action plan and compensation.

The execution readiness activities detailed in Section 24.1.5 will commence in the third quarter of 2025 and be completed in parallel with the completion of funding.

The construction phase of the Project will commence following FID in the second quarter of 2026, with the breaking of first ground to commence site earthworks and mining of the GBE pit, which provides access for establishment of the underground once it reaches fresh rock approximately nine months after FID, around the end of 2026. Following establishment of the portal the underground development, which includes minor production of development ore, it will take approximately 15 months to commence stoping in the underground mine, delivering a sustainable supply of fresh ore for the processing plant and allow commencement of production in the second quarter of 2028.

Construction of infrastructure and services will proceed over the 2-year construction period in parallel with the mine development and pre-production mining. The enabling infrastructure, such as the accommodation village, bulk earthworks and access roads, will commence immediately, followed by the commencement of the process plant and power station in the fourth quarter 2026.

The TSF and associated infrastructure will be constructed during the dry season across the fourth quarter 2026 to the second quarter 2027 to enable construction to follow on from other earthworks activities and complete construction in the dry season.

Establishment of open pit operations at NEB will commence in the fourth quarter of 2027 to supply ore from the weathered zone for processing in parallel with fresh ore from underground and stockpiled ore from the GBE pit.

Commissioning will occur approximately three months prior to first production at the start of Year 1.

The critical path for the development of the Project from FID to commencement of production is the mining of GBE pit and underground mine development. Other areas of the Project have sufficient float, as shown in Table 24.1, to be considered to not be on the critical path.

Table 24.1: Schedule Float

Area	Days Float
Mining	0
Process Plant	23
Power Station	86

Figure 24.3 provides a summary implementation schedule.

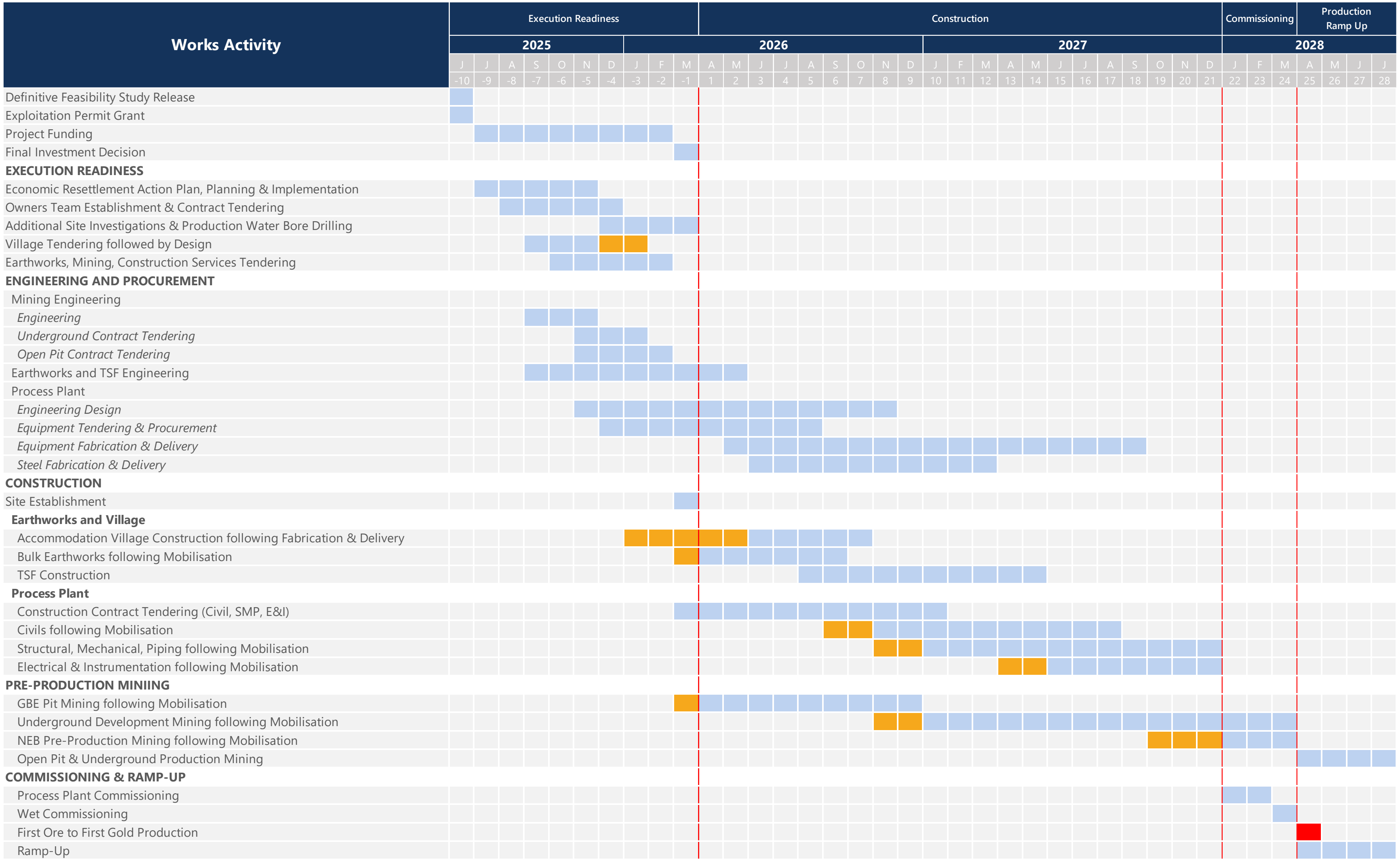

Figure 24.3: Summary Project Implementation Schedule

24.2 Operations

24.2.1 Operations Strategy

Operations across the Project will generally occur 24 hours per day with personnel working eight-hour shifts in line with Guinean labour laws which mandate a maximum of 10 hours per day. Personnel not working in continuous shift rosters will generally work Monday to Friday on a 40-hour week.

In order to bring in specialist skills and access to equipment, the following aspects of the Project operation will be contracted to suitably experienced contractors:

- Open pit mining, including drill and blast activities, underground ore and waste rehandle, ROM pad operations and mine road maintenance.
- Underground mining, including drill and blast activities, haulage of ore and waste to the surface and feeding of the paste plant feed.
- Filtered tails handling to the TSF, including tailings placement and maintenance of the filtered TSF, and the paste plant.
- Power station operations and maintenance, and site fuel supply.
- Laboratory services, including provision of equipment, consumables, systems and staff.
- Camp management, including messing for camp residents and mid-shift meals for all staff, camp cleaning, site office cleaning and laundry operations.
- Security services for site.
- Medical services for site, including doctor, nurses and paramedic/emergency response coordinator.

All other operations on site will be carried out by Company employees.

24.2.2 Logistics

The majority of reagents (other than lime and binder), consumables and spares for the Project will be sourced internationally and will therefore be imported via the port of Conakry. A single logistics contractor for general freight to site will be engaged to provide port handling management, customs clearance and inspection management, consolidation in Conakry and transport to site. The exceptions to this main logistics contractor will be:

- Fuel deliveries to site, which will be handled by the power station and fuel supply contractor.
- Explosives delivery, which will be handled by the explosives supplier, or their contracted transport contractor.
- Lime and binder delivery from ex-works in Guinea potentially by the supplier or a separate contractor (although this could be incorporated into the main logistics contract).

24.2.3 Ramp-up

The ramp-up of gold processing plants, given their simplicity and well understood nature, is generally quite rapid. Typically, based on data from a number of process plant engineering contractors, within

10 to 60 days of the introduction of ore, with an average of 30 days. However, since the Project also incorporates tailings filtration and stacking of the filtered tailings, there is additional complexity which may lead to a longer ramp up. Based on experience and input from the DFS consultants and others, the ramp-up schedule presented in Table 24.2 has been adopted for the DFS.

Table 24.2: Bankan Project Production Ramp-up

Month of Operation	% of Design Ore Treatment
1	60%
2	80%
3	90%
4	95%
5 onwards	100%

This ramp-up schedule includes impacts reductions in of instantaneous throughput, availability, utilisation and metal recovery.

24.2.4 Human Resources

The high-level organisation chart for the Project is presented in Figure 24.4.

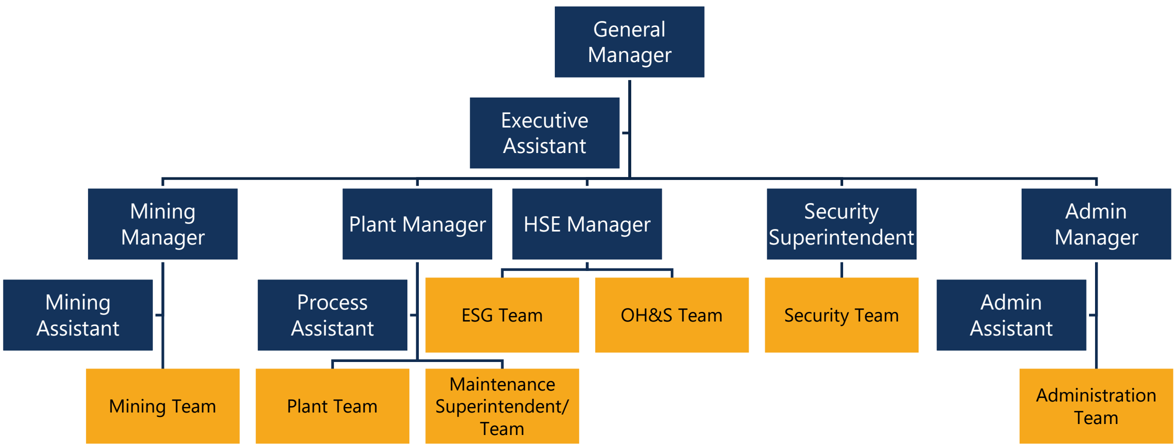

Figure 24.4: High Level Organisation Chart

This shows the operation split into five key departments, being:

- Mining, which includes:
  - Technical services comprising geologists, mining engineers and survey.
  - Open pit mining, managing the open pit mining contractor and operations.
  - Underground mining, managing the underground mining contractor and operations.
- Process plant, which includes:

- Metallurgical and engineering resources.
- Process plant operations team including shift operations team, day operations teams and a TSF operations team, which includes the operation of the paste plant.
- Process plant maintenance team which includes mechanical and electrical maintenance crews as well as maintenance planning and workshop support.
- Power station operations.
- Laboratory operations.

- Health, safety and environment, which includes:
  - Environmental management and community relations teams.
  - Occupational health and safety teams.
  - Medical doctor, nurses and paramedic.
- Security, managing all aspects of site security.
- Administration, which includes:
  - Human resources team.
  - Finance team.
  - Warehouse and procurement teams.
  - Information technology team.
  - Camp operations.

The site team will also be supported by a small team based in Conakry to provide government relations as well as support with customs, logistics and procurement.

Table 24.3 provides the breakdown of the total personnel, broken down by department, staff and contractors, and expatriate and local for the Project operations.

Table 24.3: Bankan Operations Staff

Department	Expatriate (Regional & International)	Guinean	Total
Staff			
Management/Administration	6	53	59
Mining	13	13	26
Processing	3	121	124
Conakry	-	9	9
Total Staff	**22**	**196**	**218**

Department	Expatriate (Regional & International)	Guinean	Total
Contractors			
Administration	-	144	144
Mining	18	722	740
Processing	1	24	25
Total Contractors	**19**	**890**	**909**
Total	**41**	**1,086**	**1,127**

Staff and contractors will predominately relocate to and live in Kouroussa, which is a key requirement of the Guinean government and, given the size of Kouroussa with a population of approximately 50,000, should be possible.

Expatriate and key Guinean senior personnel who do not relocate to Kouroussa, will reside in the accommodation village and will work a fly in/fly out roster to site. The total number of personnel in the accommodation village and working on a fly in/fly out roster is 73, comprising 37 staff and 36 contractors.

24.2.5 Security

A dedicated and specialist security contractor will be engaged to provide security services for site during operations (and construction). The security force will comprise approximately 126 personnel, with 6 on day shift plus approximately 30 on each 8-hour shift, who will be split across the following areas:

- Management of security operations.
- Access control staff at key entry points including the main site gatehouse, process plant and administration area gatehouse, mining infrastructure area, process plant entry and gold room entry.
- Closed circuit television (CCTV) supervisor and operators.
- Gold room supervisor and security personnel.
- Patrol officers to patrol the process plant perimeter, process plant/administration/mining infrastructure perimeter and the site perimeter.

25 INTERPRETATION AND CONCLUSIONS

25.1 Mineral Resources

All the deposits are open at depth and along strike. The PDI tenement package has widespread gold mineralisation and numerous targets and anomalies have been identified. PDI's recent resource development work has been to identify near-surface resources within a trucking distance of the proposed processing plant. Recent success from this program includes the definition of mineral resources at Fouwagbe and Sounsoun.

At NEB, the Inferred underground resource is open at depth. Resource development drilling of this target from surface requires a large number of very deep drillholes (>900m). A more attractive option would be drilling from underground positions established in the hangingwall once underground development commences.

GBE is open at depth and to the north along strike. As an interpreted along strike extension of NEB, it is expected to have similar mineralisation controls, however the high-grade core at NEB is not known to be present at GBE.

At BC the mineralisation is open at depth, however the optimisations suggest that the future potential is more likely to be an underground target. Resource development drilling is required to further define the structural controls and design the most effective drillhole patterns.

Close to NEB, the 800W area has highly anomalous drillhole intersections at an interpreted intersection of two shears. Given its proximity to NEB it is a high priority for further resource development.

25.2 Mineral Reserves

The Mineral Reserve is based on an economic analysis to feasibility study level of accuracy using the metal prices, production rates, metal recoveries, and capital and operating costs detailed in this report. The resulting positive economic outcome for the project supports the associated Mineral Reserve estimate.

It is the opinion of the Qualified Persons for both the open pit and underground that Mineral Reserves were estimated using industry-accepted practices and conform to the CIM Definition Standards for Mineral Resources and Mineral Reserves and the JORC Code 2012. Mineral Reserves stated in this report are based on a combination of open pit and underground mining methods and the Mineral Reserves are stated separately for each methodology.

The reported Mineral Reserve may be affected by future updates of modifying factors, including but not limited to mining, processing, environmental, permitting, taxation, socio-economic and other considerations.

Key technical factors which may affect the Mineral Reserve estimate include:

- Metal price assumptions as an increase or decrease can significantly impact the cut-off grade and forecast cashflow outcomes.
- Metallurgical recovery estimates can affect financial outcomes as changes impact revenue cut-off grade and forecast cashflow outcomes.

- Changes to geotechnical design parameters or mining assumptions as geotechnical variations can impact the open pit strip ratio and the underground mining methodology and costs.

25.3 Mining Methods

The mining of the Project is considered by the Qualified Persons as reasonable and technically viable based on the parameters and assumptions contained within this report. The combined open pit and underground mining methodology has been assessed as part of the feasibility study and has been determined as the approach that provides the best financial outcome for the project.

The open pits will be mined using a conventional truck and shovel methodology, typical for West Africa. The underground will utilise a combination of transverse and longitudinal long hole open stoping with paste backfill with these methodologies well understood and utilised globally in mining operations. Both the open pit and underground operations will utilise experienced mining contractors, and costs have been estimated accordingly, to ensure safe, timely and successful execution of the project in line with the feasibility study outcomes.

25.4 Mineral Processing and Metallurgical Testing

The results of comminution and metallurgical testwork conducted on a range of representative samples across the NEB (both open pit and underground, and also including GBE) and BC deposits are considered by the Qualified Person to be of sufficient scope and quality to support the proposed process plant design and production planning. Comminution testing determined that weathered lithologies are between very low competency to competent and very soft to medium hardness. The fresh lithology is very competent and very hard. Abrasiveness across all lithologies is slightly abrasive to abrasive. The single stage crushing and SABC comminution circuit selected can accommodate the spectrum of material hardness and competency observed from testwork, and can achieve the target plant throughput and grind size (P80 of 75 µm). Specific comminution power requirements varied from 5.6 kWh/t for saprolite material, 13.1 kWh/t for laterite, saprock and shear zone material and 29.2 kWh/t for fresh material.

Materials handling teswork indicated that the saprolite lithology was difficult to handle and “sticky” in nature and as such a separate mineral sizer circuit, bypassing the stockpile, was included. The design should carefully control moisture, minimise drop energies, incorporate chute angles of greater than 74 degrees from horizontal and include low friction resistance liner materials.

It has been demonstrated that the deposits are ‘free-milling’ and amenable to gold recovery by a standard gravity and CIL flowsheet. Gravity/cyanidation variability test work at optimised flowsheet conditions determined that average gold extractions by gravity is in the order of 32% with overall combined extraction ranging between 81.2% and 97.5% with a simple linear relationship between head grade and extraction best describing the leach performance. Sodium cyanide was found to be independent of lithology with a median consumption of 0.4 kg/t deemed appropriate. Lime consumption was low for fresh material at 0.33 kg/t, however increased markedly for weathered ore to 2.06 kg/t.

It was noted in some samples that higher levels of cyanide soluble copper was present in the ore increasing the sodium cyanide consumption and reducing the gold loading on carbon in the CIL circuit. This level of cyanide soluble copper was not predictable based on the geological information available and the as such the decision was made to incorporate facilities to increase cyanide levels in the leach to reduce copper loading on carbon and also for a copper wash of the loaded carbon in the elution circuit providing the operational flexibility to accommodate periods of higher cyanide soluble copper in the mill feed.

Cyanide destruction testwork demonstrated that typical operating conditions will reduce WAD cyanide levels in tails to below 50 ppm reliably.

Paste testwork on deslimed tails demonstrated the required range of paste strengths, with varying levels of binder addition.

25.5 Recovery Methods

The process plant has been designed and costed to DFS level, and is based on a robust metallurgical flowsheet designed for optimum recovery with minimal operating costs. The flowsheet is based on unit operations that are well proven in industry. Processing parameters, costs, and revenues are well understood and can be considered reliable predictors of the mineral processing performance for the Project.

25.6 Project Infrastructure

Infrastructure for the process plant, and the Project more widely, has been considered and defined to DFS level. Key infrastructure for the Project includes:

- Access road to the process plant site and internal site roads.
- Offices, workshops and other buildings.
- Accommodation village for senior staff.
- Site hybrid HFO/solar PV power station, delivered on a BOO basis, and site power distribution infrastructure.
- Tailings storage facility and tailings water storage dam.
- Site water supply and management systems.
- Surface water management infrastructure.
- Mining infrastructure, including workshops, stores, fuel distribution and washdown faiclities.
- Underground mine surface infrastructure including power supply, vent fans and cooling plant.
- Paste plant.

25.7 Environment, Social Impact and Permitting

The Project is located in the Peripheral Zone of the Upper Niger National Park and is therefore sensitive from and environmental perspective. In addition, the Project is located close to villages and the large regional centre of Kouroussa giving a level of social sensitivity.

An ESIA for the Project was submitted in 2024 and the environmental compliance certificate (the *Certificat de Conformité Environnementale*) was approved on the 17 January 2025 (CCE/00070). The ESIA clearly defines the baseline conditions at the site and range of key environmental and social risks as outlined in Section 20.3. While there are a number of risks identified it is expected that these will be successfully managed through the implementation of the mitigations and management measure outlined in the management plans and included in the DFS designs. The ESIA also included a closure plan and the costs for this closure, and requisite post-closure monitoring, have been included into the costs of the Project and the financial model.

The Company is in the process of developing the Environmental and Social Management System for the Project based off the environmental and social management plans already developed.

On 31 January 2025, the Company submitted exploitation permit applications for 50% of the Kaninko and Saman permit areas to the MMG and CPDM in accordance with Guinean mining law. The PDI Executive Director – Legal and ESG has indicated that the applications are at an advanced stage and are still being processed. PDI is not aware of any immediate obstacles to the granting of the exploitation permits.

Following the grant of the exploitation permits, PDI intends to negotiate a *Convention minière*, or mining agreement, in relation to the exploitation permits which will ultimately be ratified by the National Assembly of Guinea (currently the National Transition Council). The *Convention minière* is defined in the Mining Code as the agreement establishing the rights and obligations of the holder of an exploitation title with regard to the legal, technical, financial, fiscal, administrative, environmental and social conditions applicable to the title.

25.8 Economic Analysis

Economic analysis of the Project has shown the that at a gold price of US$2,400/oz the project is economically viable and robust offering significant financial returns.

25.9 Risks and Opportunities

A project risk and opportunity register was maintained and reviewed in a series of project risk workshops. The focus of the workshops was on identification of key risks that would delay or prevent project execution, materially impact on project costs or economic outcomes, require significant design changes or threaten project approvals.

A risk ranking matrix considering likelihood and consequence as well as industry experience from the workshop attendees were used to assess each risk.

25.9.1 Risks

A summary, which is not exhaustive, of the key risks identified during the DFS, including mitigating strategies and actions, is presented in Table 25.1.

Table 25.1: Key Risks and Mitigating Strategies

Risk	Mitigation
Geology and mineral resource – the Project's mine designs, production schedules and financial analysis are based on the estimation of tonnages and grades contained within the deposits which are inherently uncertain in nature and there is a risk that the actual tonnes and grades will differ from the estimates.	The estimates have been prepared and classified in accordance with the JORC Code (2012) which clearly outlines the requirements for estimation and classification of Mineral Resources.
Regulatory risk with any adverse changes in government policies or legislation having the potential to affect ownership of mineral interests, taxation, royalties, land access, labour relations, and Project activities.	Applications have been made for exploitation permits and are currently under consideration. In addition, once the exploitation permits are granted then the Company will seek to negotiate a mining agreement which will be binding and cement the fiscal and social arrangement agreed.
Lack of geotechnical information relating to the underground mine may result in elevated in-situ or mining induce rock stresses which may cause partial or widespread failure of the underground mine, impacting on the stated production target or costs.	Acoustic Emission in-situ rock stress measurement program and numerical modelling of mine design and sequence currently in progress. In additional, additional geotechnical drilling, logging and testing is scheduled post completion of the DFS in parallel with project funding and has been included in DFS costing.
Tailings thickening and filtration testing has been carried out on a small number of samples and therefore there is a risk that the stated production target may not be achieved, or additional sustaining capital or operating costs may be incurred.	Thickening and filtration tests have been carried out across a range of blends with performance based on 75% saprolite used to size equipment, however a maximum of 50% saprolite has been used in mine and production scheduling. Process equipment in these critical areas have been selected with significant design margin. Plant design has considered future installation of additional filtration capacity without undue cost or complexity.
Filtered stack TSF operation may be difficult during the wet season or when significant proportion of the plant feed is saprolite. This may lead to reduction or cessation of production.	The DFS design of the filtered stack TSF includes for tailings to be placed in thin layers of 1m to 2m to allow effective traffic compaction of the stack. In addition, the design allows for using low ground bearing pressure equipment (ADT60 or similar) and the lining of causeways with crushed rock to provide roadways on the tailings stack. Surface water will also be collected and removed from the TSF during the wet season to maintain a relatively dry environment.
Delays in the execution of a PPA for the power station may impact on the schedule for delivery of the power station generating sets and HV switchgear which in turn may delay production commencement.	DFS execution strategy has included the early tendering of the power station in parallel with project funding finalisation. In addition, the strategy includes execution of an agreement for generating set procurement and HV design in parallel with PPA negotiation.

25.9.2 Opportunities

In addition to the risks identified several key opportunities were identified during the DFS, including:

- Upgrades to, or extensions of, the existing Mineral Resources, particularly in the underground portion of the NEB resource, may be possible with additional drilling which would extend the life of mine and potentially improve the production rate.
- Identified additional near mine targets (such as the 800W area) or existing Inferred Mineral Resources at Fouwagbe and Sounsoun may be expanded and potentially converted into Mineral Reserves with additional drilling which would extend the life of mine.
- Increase of gold production is possible as the CIL and gold recovery circuits are capable of up to 375 koz per annum with an increase in the comminution capacity possible through addition of secondary crushing for a low cost increase of approximately 15% throughout.

26 RECOMMENDATIONS

The recommendations that have been identified by the authors are presented in this section and it is recommended that they be progressively implemented as the Project progresses or in parallel with the operations phase of the Project.

26.1 Mine Geotechnical

Mine geotechnical drilling and logging has been extensive for the NEB open pit, however additional drilling and logging is recommended for the NEB underground. This drilling should include the following:

- 4 DDHs in the proposed upper levels of the underground mine and through the crown pillar.
- 2 DDHs in the portal and upper sections of the proposed decline.
- 1 DDH in the proposed vent raise location.

The estimated costs of the recommended drilling, logging, sample testing and interpretation is US$388,000.

In addition, prior to commencement of final design for the BC pit, additional drilling, logging and testing should be undertaken to firm up on the design parameters for this pit. The cost of this work has not been estimated as the work would not be undertaken for several years under the current mine plan.

26.2 Mining

Due to the mining of the GBE pit being on the critical path it is recommended that this area be drilled for grade control prior to the commencement of mining. This program includes approximately 18,950 m of RC drilling and has been estimated to cost US$965,925.

26.3 Metallurgical Test Work

Due to the sensitivity of the process plant throughput to the performance of the tailings filters it is recommended that variability testing across a range of samples and using a range of lithological blends be undertaken on the filtration of ore samples. This, coupled with the provision in the design for easy addition of supplemental filtration capacity, will ensure that the risk of reduced throughput due to poor filtration is minimised.

In addition, the samples produced from the filtration should be subject to additional geotechnical testing to better define the design and operational processes for the TSF.

It is anticipated that this work would cost approximately US$40,000, excluding sample collection and transport.

26.4 Site Geotechnical Investigation

In order to finalise the foundation designs for the TSF and also the key parts of the process plant further geotechnical investigations are required. The will involve the following:

- 7 boreholes to a depth of up to 60m (or until encountering competent rock) across the TSF, TSF water storage dam and process plant site with down hole SPT tests.

- 55 additional test pits with DCP tests across all areas of infrastructure.
- Ripping trials on the bonded ferrocrete across the plant and infrastructure site.
- Geotechnical testing of the range of materials collected in the testing.

It has been estimated that this program of work, including supervision by a suitably experienced geotechnical engineer, would cost approximately US$450,000.

26.5 Hydrogeology

Critical to the commencement of mining activities, particularly underground mining, is the dewatering of the saprolite aquifer that sits above the fractured saprock and fresh rock. In order to reduce the groundwater levels and pore pressures as far as possible ahead of mining requirements it is recommended to commence drilling and fit out of the dewatering bores in the GBE pit and NEB pit areas as early as possible in advance of FID. These drilling, procurement and fit out costs, along with hydrogeological design and supervision, are estimated at US$1,066,234.

26.6 Hydrology

As part of the preparation for the design of the Bankan Creek diversion channel it is recommended that a stream flow gauge be installed in Bankan Creek upstream of the proposed diversion channel and pit. This will allow the flow to be measured in the waterway and the effective run-off coefficient to be accurately estimated based on the rainfall and catchment area. Installing this early in the Project would ensure multi-year data could be gathered providing more confidence in the results.

It is estimated this stream gauge would cost approximately US$60,000 and then a further US$50,000 consulting to update the various surface water models prior to the design of the diversion channel.

26.7 Environmental and Social

Following the completion of the DFS it is recommended to commence the finalisation of the economic resettlement action plan to secure unrestricted access to the land required for the Project. Then on grant of the exploitation permit it is recommended to implement this action plan as soon as possible to free the land of current users. This will allow easier implementation of many of the site investigation programs and, once the company makes FID, commencement of site works.

The completion and implementation of the economic resettlement action plan is anticipated to cost approximately US$5,852,143.

26.8 Project Implementation

As the first part of Project implementation, it is recommended to complete an execution readiness phase prior to FID. The execution readiness activities are outlined in detail in Section 24.1.5. In general, the recommended activities include:

- Engagement of the PMC and formation of the IPMT.
- Preparation of detailed project management and other plans.
- Implementation of project management systems.

- Following grant of the exploitation permits for the project, implementation of the economic resettlement action plan to gain full land access.
- Tendering and award of the EPCM and engineering contracts.
- Front end engineering for the key aspects of the Project
- Site investigations and early works as outlined above.
- Tendering, award and mobilisation of the bulk earthworks contract, which includes the mining of the GBE pit, such that the contractor can commence work on FID.
- Construction readiness activities such as finalising the early works construction insurance, tendering of the site security contract and tendering of the medical services contract.

The estimated cost for these programs, including costs outlined in the preceding sections, is outlined in Table 26.1.

Table 26.1: Recommended Execution Readiness and FEED Program Costs

Area	US$m
Owners Team and Tendering Costs	2.9
Land Acquisition and Economic Resettlement Action Plan Implementation	5.9
Site Investigations to Support Detailed Design	1.9
Dewatering Bore Installation	1.1
Total Execution Readiness	**11.8**
Process Plant FEED & Procurement	1.6
NPI FEED & Procurement	3.3
Bulk Earthworks Design	0.1
General FEED	0.3
Total FEED	**5.3**

27 REFERENCES

ALS Metallurgy. (2025). *Metallurgy Testwork conducted upon samples from the Bankan Gold Project. Report No. A26173.* Perth: ALS Metallurgy.

AngloGold Ashanti. (2025). *Siguiri, Guinea*. Retrieved from AngloGold Ashanti Corporate Website: https://www.anglogoldashanti.com/portfolio/africa/siguiri/

Barradas, S. (2016, 09 02). *Siguiri mine, Guinea.* Retrieved from Engineering News: https://www.engineeringnews.co.za/print-version/siguiri-mine-guinea-2016-08-26

Barradas, S. (2023, 02 03). *Kiniero gold project, Guinea – update*. Retrieved from Mining Weekly: https://www.miningweekly.com/article/kiniero-gold-project-guinea-update-2023-02-03

Barton, N. (1978). Suggested Methods for the Quantitative Description of Discontinuities in Rock Masses. *International Journal of Rock Mechanics and Mining Sciences & Geomechanics Abstracts 15*, 319-368.

Bolay, M. (2016). Artisanal Gold Miners Encountering Large-Scale Mining in Guinea: Expulsion, Tolerance and Interference. In T. Niederberger, T. Haller, H. Gambon, M. Kobi, & I. (. Wenk, *The Open Cut: Mining, Transnational Corporations and Local Populations. Action Anthropology / Aktionsethnologie* (pp. Vol 2: 187-204). Zurich/Berlin: LIT Verlag.

Chardon, D., Grimauld, J.-L., Beauvaise, A., & Bamba, O. (2018). West African Lateritic Pediments: Landform-regolith Evolution Processes and Mineral Exploration Pitfalls. *Earth-Science Reviews 179*, 124-140.

Chen, J. (2025, May 6). Email Message to Brad Milne.

CIM Standing Committee on Reserve Definitions. (2014). *CIM Definition Standards for Mineral Resources & Mineral Reserves.* Westmount: Canadian Institute of Mining, Metallurgy and Petroleum.

Climate Hazrds Centre, University of California, Santa Barbara. (2025). *CHIRPS: Rainfall Estimates from Rain Gauge and Satellite Observations*. Retrieved from https://www.chc.ucsb.edu/data/chirps

Climatic Research Unit, University of East Anglia. (2025, March). *CRU TS Version 4.09*. Retrieved from https://crudata.uea.ac.uk/cru/data/hrg/cru_ts_4.09/

Como Engineers. (2023). *Bankan Gold Project Metallurgical Testwork Review. Job No. 3903.01.* Perth: Como Engineers.

Como Engineers. (2024). *Banakan Gold Project Prefeasibilty Study. Job No. 3903.01. Rev E.* Perth: Como Engineers.

Doumbouya, I. D., Dessertine, A., Vinches, M., & Cerceau, J. (2024). Mechanization of artisanal and small-scale gold mining in Guinea: Socio-technical trajectory of a rural mining site in Upper Guinea. *Journal of Rural Studies 112*. Retrieved from https://imt-mines-ales.hal.science/hal-04748611/document

ERM. (2024). *Bankan Gold Project - Environmental and Social Impact Assessment.* London: Environmental Resource Management.

Food and Agriculture Organization of the United Nations. (2025, January 29). *Global Weather for Agriculture - AgERA5*. Retrieved from https://data.apps.fao.org/catalog/dataset/global-weather-for-agriculture-agera5

Global Run off Data Centre. (n.d.). Retrieved from https://grdc.bafg.de/

Government of Western Australia. (2022). Work Health and Safety (Mines) Regulations 2022 (SL 2022/32). Retrieved from https://www.legislation.wa.gov.au/legislation/statutes.nsf/law_s53266.html

IMO. (2024). *Bankan Gold Project Ore Characteriation Testwork. Project 6637.* Perth: IMO.

International Council on Mining and Metals. (2022). *Mining Principles: Performance Expectations.* ICMM. Retrieved from https://www.icmm.com/en-gb/our-principles/mining-principles/mining-principles

International Cyanide Management Institute. (2005). *International Cyanide Management Code for the Manufacture, Transport and Use of Cyanide in the Production of Gold.* International Cyanide Management Institute. Retrieved from https://cyanidecode.org

International Cyanide Management Institute. (2021). *The International Cyanide Management Code - Guidance for the use of the Mining Operations Verification Protocol.* Washington: International Cyanide Management Institute.

International Finance Corporation. (2007). *Environmental, Health, and Safety General Guidelines.* Washington: Internation Finance Corporation.

International Finance Corporation. (2012). *IFC Performance Standards on Environmental and Social Sustainability.* Washington: International Finance Corporataion.

International Mining. (2023, 06 09). *International Mining*. Retrieved from Hummingbird achieves first gold pour at Kouroussa mine in Guinea: https://im-mining.com/2023/06/09/hummingbird-achieves-first-gold-pour-at-kouroussa-mine-in-guinea/

International Organization for Standardization. (2010). *ISO 26000:2010 Guidance on social responsibility.* ISO. Retrieved from https://www.iso.org/iso-26000-social-responsibility.html

International Organization for Standardization. (2015). *ISO 14001:2015 Environmental management systems — Requirements with guidance for use (3rd ed.).* ISO. Retrieved from https://www.iso.org/standard/60857.html

International Organization for Standardization. (2018). *ISO 45001:2018 Occupational health and safety management systems — Requirements with guidance for use.* ISO. Retrieved from https://www.iso.org/standard/63787.html

Jenike & Johanson. (2025). *Gold Ore Flow Properties Report, 73065-1 Rev 0.* Perth: Jenike & Johanson.

Joint Ore Reserves Committee. (2012). Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (The JORC Code). The Joint Ore Reserves Committee of The Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists, and Minerals Council of Australia. Retrieved from https://www.jorc.org/docs/JORC_code_2012.pdf[1](https://www.jorc.org/docs/JORC_code_2012.pdf

Lahondère, D., Egal, E., Lacomme, A., Le Berre, P., Costea, C. A., Devnoux, M., . . . Feybesse, J.-L. (1999). *Projet de cartographie géologique du Nord-Est de la République de Guinée et Notice explicative de la carte géologique à 1/200 000. Feuille n° 12 87. Siguiri. 1ère édition.* Conakry: Edité sous la responsabilité du Ministère des Mines, de la Géologie et de l'Environnement.

Lebrun, E., Thébaud, N., Miller, J., Roberts, M., & Evans, N. (2017). Mineralisation Footprints and Regional Timing of the World-Class Siguiri Orogenic Gold District (Guinea, West Africa). *Mineralium Deposita 52(4)*, 539-564.

Macfarlane, I., & Love, S. (2008). *Technical Report: Independent Mineral Resource Estimate on the Kouroussa Gold Project Guinea, West Africa.* Perth: Coffey Mining Pty Ltd. Retrieved from https://www.sedarplus.ca/csa-party/records/document.html?id=02e290ae362a5b4ce4fcbc6e05c71d4d864e360a5f7a2f92c9c6a23a941c3d2f

MDO Data Online Inc. (2025). *Kouroussa Mine*. Retrieved from Mining Intelligence and News: https://miningdataonline.com/property/1919/profile.aspx?pid=1919

Metso. (2024). *Bankan Gold Project Thickening Report. Test Number 3516297.* Perth: Metso.

Metso. (2024). *Filtration Test Report for Bankan Gold Project. Test Number 3516299.* Perth: Metso.

Micklethwaite, S., Sheldon, H., & Baker, T. (2010). Active Fault and Shear Processes and their Implications for Mineral Deposit Formation and Discovery. *Journal of Structural Geology 32*, 151-165.

Miller-Tait, L., Pakalnis, R., & Poulin, R. (1995). *UBC Mining Method Selector.* Calgary: International Symposion on Mine Planning.

Minefill Services. (2025). *Bankan Paste Prefeasibility Study Testwork. Report No. 24054-RPT-0002. Rev A.* Tighes Hill: Minefill Services.

Minefill Services. (2025). *Bankan Paste Pre-Feasibility Study. Report Number. 24054-RPT-0001. Rev B.* Tighes Hill: Minefill Services.

Mining Technology. (2020, 09 01). *Mining Technology*. Retrieved from Hummingbird completes Kouroussa gold project acquisition: https://www.mining-technology.com/news/hummingbird-acquires-kouroussa-gold-project/?cf-view

Mintrex. (2021). *Bankan Project Scoping Testwork Report. Project Number: 21017-PDI.* Perth: Mintrex.

Murphy, B. (2022). Structural Features and Paragenesis of the NE Bankan Gold Deposit: A Preliminary Study. Predictive Discovery Limited (Internal Report).

NASA. (n.d.). *Global Precipitation Measurement*. Retrieved from https://gpm.nasa.gov/data/imerg

Ngo, D. (2022). *Memo: 22033 PDI - Additional Leach Results.* Perth: Mintrex.

Orway Mineral Consultants. (2024). Bankan Gold Project Comminution Option Assessment. Report No. 7826 Rev 0. Perth: OMC.

Orway Mineral Consultants. (2025). Bankan Gold Project Mine Plan Blend Comminution Design. Perth: OMC.

Orway Mineral Consultants. (2025). Bankan Gold Project Mine Plan Blend Comminution Design. Perth: OMC.

Peter O'Bryan & Associates. (2025). Technical Note - Definitive Feasibility Study Geotechnical Assessment Preliminary Base Case Design Parameters.

Peter O'Bryan and Associates. (2025). *Bankan Gold Project, Geotechnical Assessment, Open Pit and Underground Mining Bankan North-East Deposit, Open Pit Mining Bankan Creek and Gbengbeden Deposits*. Perth.

Predictive Discovery Limited. (2024, April 15). *PFS Delivers Attractive Financials & 3.05 Moz Ore Reserve*. Retrieved from Predictive Discovery Corporate Web Site: https://pdi.live.irmau.com/pdf/3fbd3f02-b385-44a8-8a29-c943ca6dca18/PFS-Delivers-Attractive-Financials-305MOZ-Ore-Reserve.pdf

Predictive Discovery Limited. (2025, June 25). *Bankan DFS Confirms Outstanding Project Economics*. Retrieved from Predictive Discovery Corporate Website: https://pdi.live.irmau.com/pdf/6c354ad4-645c-4c38-bf49-121dd834f39b/Bankan-DFS-Confirms-Outstanding-Project-Economics.pdf

Szebor, N., Williamson, G., Kent, M., Kirchner, I., Cunningham, R., Thompson, J., . . . Coetzee, F. (2025). *Technical Report, Kiniero Gold Project, Guinea*. Perth: AMC Consultants. Retrieved from https://www.sedarplus.ca/csa-party/records/document.html?id=abc50645d06a13066c8cfb07403ac3673a08549e73c465d114889407862e2d24

The Perth Mint. (2025). *Live Australian Prices Gold, Silver, Platinum*. Retrieved from The Perth Mint Australia: https://www.perthmint.com/invest/information-for-investors/metal-prices/

Tucker, D., Carneiro, A., Wiid, G., Berton, A., Thompson, J., & Coetzee, F. (2022). *Kiniero Gold Project, Guinea - Pre-Feasibility Study (NI43-101 Technical Report)*. Bristol: Mining Plus. Retrieved from https://www.sedarplus.ca/csa-party/records/document.html?id=50db3bc82d6419f6b937bbd388e2ccf794d3c859770e1ed8127a160fde1333f0

World Gold Council. (2019). *Responsible Gold Mining Principles*. London: World Gold Council.

28 QUALIFIED PERSON CERTIFICATES

Level 3
1 Havelock St
West Perth WA 6872

T +61 8 6467 1600

erm.com

DATE
31 July 2025

SUBJECT
PDI QP Certificate

REFERENCE

CERTIFICATE OF QUALIFIED PERSON – PHILIP JANKOWSKI

I, Philip Edward Jankowski, FAusIMM, as an author of the report entitled "Technical Report, Bankan Gold Project, Siguiri Basin Guinea" (the "Report") prepared for Predictive Discovery Limited ("PDI") and with an effective date of July 31st 2025, do hereby certify that:

1. I am employed as Technical Consulting Director of ERM Consulting Pty Ltd of Level 3,1 Havelock Street, West Perth, Western Australia.
2. I graduated with a Bachelor of Science in 1987 and a Graduate Diploma in Science in 1988 from the Australian National University, and a Masters of Science from the University of Western Australia in 2003.
3. I am a registered Fellow of the Australasian Institute of Mining and Metallurgy (#111081). I have been practicing my profession continuously since 1988 and my experience covers 37 years in the mining industry involving mining geology, resource development, resource estimation and due diligence reviews in both corporate and consulting roles, in Australia, Africa and Asia.
4. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (Standards for Disclosure of Mineral Projects) (the "Instrument") and certify that by reason of my education, affiliation with a professional association (as defined in the Instrument) and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of the Instrument.
5. I have visited the Bankan Gold Project on four occasions, from the 10th to the 15th June 2022, from the 10th to the 21st November 2022, from the 11th to the 27th January 2023 and from 28th August 2024 to the 5th September 2024 .
6. I am responsible for Sections 1.2, 1.3, 1.4, 1.5, 1.6, 1.7, 1.9, 1.18.1, 6, 7, 8, 9, 10, 11, 12, 14, and 25.1 of the Report.
7. I am independent of PDI as described in Section 1.5 of the Instrument.
8. I have completed four previous consulting assignments on the properties that are the subject of the Report, comprising Mineral Resource Estimates and technical advice on geology, sampling and modelling.
9. I have read the Instrument and the parts of the Report that I am responsible have been prepared in compliance with the Instrument and I consent to the filing of the Report.

10. As of the effective date of the Report, to the best of my knowledge, information and belief, the parts of the Report that I am responsible for, contain all scientific and technical information that is required to be disclosed and to make the Technical Report not misleading.

Dated this 31st day of July, 2025.

P Jankowski

Philip Jankowski MSc, FAusIMM
TECHNICAL CONSULTING DIRECTOR
ERM Consulting Pty Ltd

orelogy™

CERTIFICATE OF QUALIFIED PERSON – ROSS CHEYNE

I, Ross Cheyne, FAusIMM, as an author of the report entitled "Technical Report, Bankan Gold Project, Siguiri Basin Guinea" (the "Report") prepared for Predictive Discovery Limited ("PDI") and with an effective date of July 31, 2025, do hereby certify that:

1. I am employed as Principal Mining Consultant of Orelogy Consulting Pty Ltd of Level 2, 101 St Georges Terrace, Perth, Western Australia.
2. I graduated with a Bachelor of Engineering in Mining Engineering from the University of Auckland, New Zealand in 1989.
3. I am a registered Fellow of the Australasian Institute of Mining and Metallurgy (#109345). I have been practicing my profession continuously since 1990 and my experience covers 35 years in the mining industry involving the design, mine planning, scheduling, operations, and project management, of mining projects and technical studies, in Australia, Africa and Asia.
4. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (Standards for Disclosure of Mineral Projects) (the "Instrument") and certify that by reason of my education, affiliation with a professional association (as defined in the Instrument) and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of the Instrument.
5. I visited the Bankan Gold Project in January 2025 for a period of approximately 4 days.
6. I am responsible for Sections 1.10 (excluding underground Mineral Reserves), 1.11 (excluding 1,11.2), 1.18.2 , 15.1, 15.2, 15.3, 15.6, 16.1, 16.2.1, 16.3, 16.4, 16.6, 16.7, 21.2.3.1, 25.2, 25.3 and 26.2 of the Report.
7. I am independent of PDI as described in Section 1.5 of the Instrument.
8. I have had no prior involvement with the properties that are the subject of the Report.
9. I have read the Instrument and the parts of the Report that I am responsible have been prepared in compliance with the Instrument and I consent to the filing of the Report.
10. As of the effective date of the Report, to the best of my knowledge, information and belief, the parts of the Report that I am responsible for, contain all scientific and technical information that is required to be disclosed and to make the Technical Report not misleading.

Dated this 31st day of July, 2025.

"signed and sealed"

Ross Cheyne, BEng (Mining), FAusIMM
Principal Mining Consultant
Orelogy Consulting Pty Ltd

CERTIFICATE OF QUALIFIED PERSON – JULIAN BROOMFIELD

I, Julian Martin Broomfield, FAusIMM, as an author of the report entitled "Technical Report, Bankan Gold Project, Siguiri Basin Guinea" (the "Report") prepared for Predictive Discovery Limited ("PDI") and with an effective date of July 31, 2025, do hereby certify that:

1. I am employed as Principal Mining Consultant of Orelogy Consulting Pty Ltd of Level 2, 101 St Georges Terrace, Perth, Western Australia.
2. I graduated with a Bachelor of Engineering in Mining Engineering from Curtin University of Technology, Perth, Australia in 1995.
3. I am a registered Fellow of the Australasian Institute of Mining and Metallurgy (#222417). I have been practicing my profession continuously since 1995 and my experience covers 28 years in the mining industry involving the design, mine planning, scheduling, operations, and project management, of mining projects and technical studies, in Australia, Africa, America and Asia.
4. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (Standards for Disclosure of Mineral Projects) (the "Instrument") and certify that by reason of my education, affiliation with a professional association (as defined in the Instrument) and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of the Instrument.
5. I have not visited the Bankan Gold Project, but relied on information from Ross Cheyne (Orelogy Consulting) who visited in January 2025 for a period of approximately 4 days.
6. I am responsible for Sections 1.10 (underground Mineral Reserves only), 1.11.2, 15.7, 16.2.2, 16.5, 16.8, and 21.2.3.2 of the Report.
7. I am independent of PDI as described in Section 1.5 of the Instrument.
8. I have had no prior involvement with the properties that are the subject of the Report.
9. I have read the Instrument and the parts of the Report that I am responsible have been prepared in compliance with the Instrument and I consent to the filing of the Report.
10. As of the effective date of the Report, to the best of my knowledge, information and belief, the parts of the Report that I am responsible for, contain all scientific and technical information that is required to be disclosed and to make the Technical Report not misleading.

Dated this 31st day of July, 2025.

"signed and sealed"

Julian Broomfield, BEng (Mining), FAusIMM
Principal Mining Consultant
Orelogy Consulting Pty Ltd

PETER O'BRYAN & Associates

consultants in mining geomechanics

CERTIFICATE OF QUALIFIED PERSON – PETER O'BRYAN

I, Peter Robin O'Bryan, MAusIMM (CP), as an author of the report entitled "Technical Report, Bankan Gold Project, Siguiri Basin Guinea" (the "Report") prepared for Predictive Discovery Limited ("PDI") and with an effective date of 31 July 2025, do hereby certify that:

1. I am employed as Principal Geotechnical Engineer of Peter O'Bryan & Associates of Suite 4, 114 Churchill Avenue, Subiaco, Western Australia.
2. I graduated with a Bachelor of Engineering in Mining Engineering from the University of New South Wales in 1981 and as a Master of Engineering Science (Rock Engineering) from James Cook University in 1990.
3. I am a registered Member (Chartered Professional) of the Australasian Institute of Mining and Metallurgy (#203335). I have been practicing my profession continuously since 1982 and my experience in geomechanics spans 43 years in the mining industry. I have been involved in research, underground and open pit operations and in consulting. My involvement has included investigation and assessment, selection of mining methods and derivation of mining parameters, operational support, guidance and troubleshooting, geotechnical monitoring and mine closure assessment in Australasia, Africa, Asia, Europe and North and South America.
4. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (Standards for Disclosure of Mineral Projects) (the "Instrument") and certify that by reason of my education, affiliation with a professional association (as defined in the Instrument) and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of the Instrument.
5. I have not visited the Bankan Gold Project. Scott Campbell (BEng (Geological Engineering), Royal Melbourne Institute of Technology, 1997), an Associate of Peter O'Bryan & Associates, visited the site for a period of nine (9) days in October 2024 and acted under my supervision.
6. I am responsible for Sections 15.4 and 26.1 of the Report.
7. I am independent of PDI as described in Section 1.5 of the Instrument.
8. I have had no prior involvement with the properties that are the subject of the Report.
9. I have read the Instrument and the parts of the Report that I am responsible have been prepared in compliance with the Instrument and I consent to the filing of the Report.
10. As of the effective date of the Report, to the best of my knowledge, information and belief, the parts of the Report that I am responsible for, contain all scientific and technical information that is required to be disclosed and to make the Technical Report not misleading.

Dated this 31st day of July 2025.

Peter O'Bryan, BEng (Mining), MAusIMM (CP)

Director | Principal Geotechnical Engineer

PETER O'BRYAN & *Associates*

Australasian
Groundwater
& Environmental
Consultants

CERTIFICATE OF QUALIFIED PERSON – Pieter Labuschagne

I, Pieter Ferdinandus Labuschagne (Pr.Sci.Nat.400386/11), as an author of the report entitled “Technical Report, Bankan Gold Project, Siguiri Basin Guinea” (the “Report”) prepared for Predictive Discovery Limited (“PDI”) and with an effective date of 31 July 2025, do hereby certify that:

1. I am employed as Principal Hydrogeologist, with the firm of AGE (Pty) Ltd and situated in 15 Mallon Street, Bowen Hills, Queensland 4006 Australia.
2. I graduated with a Master’s of Science degree in Hydrogeology (2004) from the University of the Free State, Bloemfontein, South Africa.
3. I am a practising Hydrogeologist and registered Member of the South African Council for Natural Scientific Professions – SACNASP (Pr.Sci.Nat.400386/11) as well as a member of the International Association of Hydrogeologists (IAH) and associate of the Australasian Institute of Mining and Metallurgy (AusIMM membership number: 3047226). I have practised my profession continuously since 1998 and completed more than 50 mining-related hydrogeological studies across Africa and Australia.
4. I have read the definition of “Qualified Person” set out in National Instrument 43-101 (Standards for Disclosure of Mineral Projects) (the “Instrument”) and certify that by reason of my education, affiliation with a professional association (as defined in the Instrument) and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of the Instrument.
5. I have not visited the Project site and have relied on the observations of Neville Paxton (Master of Science in Hydrogeology and a Bachelor of Science Honours in Environmental and Engineering Geology), Senior Project Hydrogeologist from Australasian Groundwater and Environmental Consultants, who visited the site for six (6) days in November 2024 and acted under my supervision.
6. I am responsible for Sections 15.5, 26.5, 26.6 of the Report.
7. I am independent of PDI as described in Section 1.5 of the Instrument.
8. I have had no prior involvement with the properties that are the subject of the Report.
9. I have read the Instrument and the parts of the Report that I am responsible have been prepared in compliance with the Instrument and I consent to the filing of the Report.
10. As of the effective date of the Report, to the best of my knowledge, information and belief, the parts of the Report that I am responsible for, contain all scientific and technical information that is required to be disclosed and to make the Technical Report not misleading.

Dated this 31st day of July 2025.

Pieter Ferdinandus Labuschagne (Pr.Sci.Nat.400386/11)
Principal Hydrogeologist
Australasian Groundwater and Environmental Consultants Pty Ltd

Brisbane Head Office
Level 2, 15 Mallon Street
Bowen Hills QLD 4006
t: (07) 3257 2055

Newcastle
4 Hudson Street
Hamilton NSW 2303
t: (02) 4962 2091

Perth
46B Angove Street
North Perth WA 6006
t: (08) 6383 9970

Townsville
Unit 1, 60 Ingham Road
West End QLD 4810
t: (07) 4413 2020

ageconsultants.com.au | linkedin.com/company/ageconsultants

CERTIFICATE OF QUALIFIED PERSON – STEWART WATKINS

I, Stewart Thomas Watkins, FAusIMM, as an author of the report entitled "Technical Report, Bankan Gold Project, Siguiri Basin Guinea" (the "Report") prepared for Predictive Discovery Limited ("PDI") and with an effective date of July 31, 2025, do hereby certify that:

1. I am Director and Principal Project Consultant of Dhamana Consulting Pty Ltd of Suite 10, 50 Oxford Close, West Leederville, Western Australia.
2. I graduated with a Bachelor of Engineering in Chemical Engineering from Curtin University of Technology in 1991.
3. I am a registered Fellow of the Australasian Institute of Mining and Metallurgy (#112142). I have been practicing my profession continuously since 1991 and my experience covers 34 years in the mining industry involving the operation, design, development and management of mining projects and corporate roles, both technical and managerial, on Projects in Australia, Africa and Asia.
4. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (Standards for Disclosure of Mineral Projects) (the "Instrument") and certify that by reason of my education, affiliation with a professional association (as defined in the Instrument) and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of the Instrument.
5. I visited the Bankan Gold Project in January 2025 for a period of approximately 4 days.
6. I am responsible for Sections 1.1, 1.8, 1.12, 1.13, 1.14, 1.15, 1.16, 1.17, 1.18 (excluding 1.18.1 and 1.18.2), 1.19, 2, 3, 4, 5, 13, 17, 18, 19, 20, 21.1, 21.2 (excluding 21.2.3), 22, 23, 24, 25 (excluding 25.1, 25.2 and 25.3), 26 (excluding 26.1, 26.2, 26.5 and 26.6) and 27 of the Report.
7. I am independent of PDI as described in Section 1.5 of the Instrument.
8. I have had no prior involvement with the properties that are the subject of the Report.
9. I have read the Instrument and the parts of the Report that I am responsible have been prepared in compliance with the Instrument and I consent to the filing of the Report.
10. As of the effective date of the Report, to the best of my knowledge, information and belief, the parts of the Report that I am responsible for, contain all scientific and technical information that is required to be disclosed and to make the Technical Report not misleading.

Dated this 31 day of July, 2025.

"signed and sealed"
Stewart Watkins, BEng (Chem), FAusIMM
DIRECTOR | PRINCIPAL PROJECT CONSULTANT
Dhamana Consulting Pty Ltd