predictive discovery

ASX Announcement

25 June 2025

BANKAN DFS CONFIRMS OUTSTANDING PROJECT ECONOMICS

Predictive Discovery Limited (ASX:PDI) ("PDI" or the "Company") is delighted to announce results of the Definitive Feasibility Study ("DFS") for the Bankan Gold Project ("Bankan" or the "Project") in Guinea, West Africa. The DFS builds on the outcomes of the Pre-Feasibility Study ("PFS") announced in April 2024, providing greater confidence in the Project's development and operating plans, and reaffirming its large scale, long mine life and compelling financials.

HIGHLIGHTS

Production and Financial Highlights

- Life-of-mine ("LOM") average production of approximately 250koz per annum over 12.2 years (for total production of 3.03Moz) from mill feed of 54.5Mt @ 1.86g/t containing 3.26Moz of gold.
- DFS delivers outstanding returns with post-tax project $NPV_{5\%}$ of US$1.6bn (A$2.5bn[1]), IRR of 46% and payback period of <2 years at a base case gold price assumption of US$2,400/oz, which reflects the current median long-term real consensus gold price forecast.
- Significant upside at recent average spot gold prices of ~US$3,300/oz. Post-tax $NPV_{5\%}$ increases to US$2.9bn (A$4.5bn[1]) with an IRR of 73% and payback period of just over 1 year.
- Competitive capital and operating costs largely in-line with PFS estimates on an overall basis, but with improved accuracy (±15% vs ±25%) and up-to-date pricing (Q1 2025 vs Q4 2023):
  - Capital cost estimate of US$463m, which includes pre-production operating costs, indirect costs and US$34m contingency.
  - Leading all-in sustaining costs ("AISC")[2] of ~US$1,057/oz, delivering strong free cash flow.

The production targets and forecast financial information in the DFS are based on 90.6% Probable Ore Reserves and 9.4% Inferred Mineral Resources (contained gold basis). Inferred Mineral Resources, which are only included from the lower part of the NEB underground mine, are processed predominantly in years 5 to 9 and represent only 0.5% of contained gold processed in the first 4 years. There is a low level of geological confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of Indicated Mineral Resources or that the production targets will be realised.

- Updated Probable Ore Reserve estimate of 51.6Mt @ 1.78g/t for 2.95Moz of contained gold across the NEB open pit (including Gbengbeden ("GBE")), NEB underground and BC open pit areas (refer to Table 1 below). Ore Reserves estimated at a conversative gold price of US$1,800/oz.

[1] Converted at a USD:AUD exchange rate of 0.65.
[2] Calculated based on the World Gold Council definition.

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Table 1: Ore Reserve estimate

Deposit	Mining Method	Classification	Cut-off (g/t Au)	Tonnes (Mt)	Grade (g/t Au)	Contained (koz Au)
NEB	Open Pit	Probable	0.38-0.48	40.2	1.36	1,751
	Underground	Probable	2.0	7.9	3.95	1,002
	Total			**48.1**	**1.78**	**2,753**
BC Open Pit	Open Pit	Probable	0.38-0.48	3.5	1.78	200
	Total			**3.5**	**1.78**	**200**
Total Open Pit				**43.7**	**1.39**	**1,951**
Total Underground				**7.9**	**3.95**	**1,002**
Total Bankan Project				**51.6**	**1.78**	**2,953**

Development and Operating Highlights

- Additional geotechnical testwork and open pit vs underground transition studies have delivered steeper wall angles across all open pits and a smaller NEB pit, resulting in a significantly reduced LOM waste to ore strip ratio of 1.9:1 (previously 4.6:1 in the PFS) and a larger contribution of ounces from the NEB underground mine at an increased underground mining rate.
- Open pit mining will be a conventional drill, blast, truck and shovel approach. Mining of the NEB underground orebody has been optimised to a hybrid longitudinal/transverse long hole open stoping mining method, with paste fill. Contract mining will be utilised for both open pit and underground operations.
- The processing plant, utilising conventional CIL technology with upfront gravity recovery, has been designed with a 4.5Mtpa capacity to suit the optimised mining schedule. Additional metallurgical testwork has resulted in improved LOM average recoveries of 92.8%.
- Construction is scheduled to occur over a two-year period which is aimed to commence in Q2 2026 following execution readiness activities (including selected early works), enabling the start of commercial production in Q2 2028.
- The execution and production schedule continues to envisage early development of the NEB underground mine to enable delivery of underground ore to the processing plant for the start of production. This allows higher grade material to be brought forward in the mine plan and ensures a minimum of 25% of fresh ore feed to meet plant operational requirements.
- The GBE pit will be mined and stockpiled over the first 9 months of the construction period to enable a portal for underground access to be established in fresh rock at the base of the pit. Underground development will take approximately 15 months to access the first stope ore, with development ore stockpiled during this period.
- The NEB open pit will be mined in three stages to prioritise access to higher grade ore and allow material movements to be balanced over the LOM. Pre-stripping at NEB will be initiated 3 months prior to commencement of production and mining will continue throughout the LOM.
- The BC open pit will be mined in years 9 to 11, deferring the additional capital cost required to establish mining operations at the deposit.

Commenting on the DFS outcomes, Managing Director Andrew Pardey, said:

"The DFS confirms the Bankan Project will be developed into one of the largest gold mines in West Africa in a generation and deliver compelling returns for shareholders and stakeholders alike."

"Bankan will produce approximately 250koz per annum for over 12 years at a highly competitive AISC cost of US$1,057/oz, delivering a post-tax $NPV_{5\%}$ of US$1.6bn, an IRR of 46% and payback of the US$463m of capital in less than two years at a gold price of US$2,400/oz. The Project demonstrates exceptional leverage to the current strong gold price environment, with each additional US$100/oz increasing the NPV by approximately US$140m.

"Through extensive optimisation and strategic reviews since the PFS last year, we have demonstrated that returns will be maximised through significantly reducing the open pit strip ratio and improving the throughput grade, via a strategy which includes a smaller NEB open pit with steeper wall angles together with a larger contribution from the underground mine from the commencement of operations."

"Furthermore, the Project will generate extensive benefits for our host country Guinea, with government revenues totalling approximately US$2bn (undiscounted) across Bankan's mine life plus potential upside from current gold prices and future mine life extensions. Beyond these direct financial benefits, the Project will also create significant job opportunities, with a peak construction workforce of ~1,500 personnel and an operational workforce of >1,100 personnel."

"PDI is in the final stage of the Exploitation Permit review process with the Government of Guinea, and the granting of this permit is a major de-risking event for the Project. We are confident the Exploitation Permit will be secured in the near future."

"In parallel, completion of the DFS will trigger commencement of the financing process, execution readiness activities and selected early works, as we rapidly advance the Project towards construction in Q2 2026."

Permitting Status

- As previously reported, a major de-risking milestone was achieved in January 2025 when the Ministère de l'Environnement et du Développement Durable (Ministry of Environment and Sustainable Development or "MEDD") approved the Environmental and Social Impact Assessment ("ESIA") and issued the Environmental Compliance Certificate ("ECC") for the Project.
- The Company submitted a Permis d'Exploitation Minière Industrielle (Exploitation Permit or Mining Operating Title) application to the Ministère des Mines et de la Géologie (Ministry of Mines and Geology or "MMG") on 31 January 2025 and is collaborating closely with the MMG to complete the review process, which has progressed to the final stage. The grant of the Exploitation Permit will be by presidential decree and will provide PDI with the necessary approval to develop the Project in accordance with the terms of Guinea's Mining Code.

Mine Life Extension Opportunities

There is excellent potential for the Project's mine life to be extended through additional drilling and exploration:

- The NEB deposit is open at depth beneath the existing underground Mineral Resources;
- The BC and GBE deposits have potential to be expanded through additional targeted drilling;
- There is significant exploration potential at near-resource and regional[3] exploration targets.

Next Steps

- Continued engagement with the Government of Guinea to secure the Exploitation Permit.
- Commence the financing process to secure committed funding for the Project.
- Initiate execution readiness activities, which in the near-term will include:
    - Creation of an integrated project management team;
    - Preparation of a detailed project management plan and implementation of project management systems;
    - Advancing the project execution and contracting strategy to support tendering and award of the EPCM and engineering contracts;
    - Engagement of key engineering consultants and commencement of front-end engineering design;
    - Commencing implementation of the economic resettlement action plan and livelihood restoration plan (including land compensation arrangements) following grant of the Exploitation Permit;
    - Further site investigations and early works to support final designs.

CONFERENCE CALL

PDI's executive team is hosting two conference calls, which investors are encouraged to join using the links provided below.

Aus EST	UK BST	US EDT	Link
11:00am, 25-Jun	2:00am, 25-Jun	9:00pm, 24-Jun	https://us06web.zoom.us/webinar/register/WN_gQnqR8Q5QI-BdCdpGh_AYw
11:00pm, 25-Jun	2:00pm, 25-Jun	9:00am, 25-Jun	http://webcasting.buchanan.uk.com/broadcast/6853d45d9f59e500133d4050

[3] Refer to ASX announcement "Argo and Bokoro Exploration Permits Update" dated 28 May 2025.

INTERACTIVE 3D MODEL

An online interactive 3D model of the Bankan Gold Project is available for viewing at the following link: https://inventum3d.com/c/predictivediscovery/bankan.

ASX LISTING RULE 5.9.1 REQUIREMENTS

Material Assumptions and Outcomes of the DFS

Key DFS assumptions and outputs are summarised in Table 2 below. Further details are available in the DFS Summary Report, which is included in this announcement.

Table 2: Key Project and Financial Metrics

	Unit	Number
Mining		
Open Pit Ore Mined	Mt	43.7
Open Pit Strip Ratio	X	1.9
Open Pit Grade	g/t	1.39
Open Pit Contained Gold	koz	1,951
Underground Ore Mined	Mt	10.8
Underground Grade	g/t	3.77
Underground Contained Gold	koz	1,309
Total Ore Mined	Mt	54.5
Average Grade	g/t	1.86
Total Contained Gold	koz	3,260
Contained Gold from Inferred Mineral Resources	%	9.4%
Processing		
Mine Life	Years	12 years and 2 months
Processing Rate	Mtpa	4.5
Total Ore Processed	Mt	54.5
Average Processing Recovery	%	92.8%
Total Gold Production	koz	3,026
Average Gold Production	koz pa	249
Capital Costs		
Pre-Production Mining Costs	US$m	105.6
Direct Construction Costs	US$m	241.5
Indirect Construction Costs	US$m	31.3
Owners Costs	US$m	50.2
Contingency	US$m	34.3
Total Pre-Production Capital Cost	**US$m**	**463.0**
Sustaining Capital Costs	US$m	208.6
Closure Costs (Net of Salvage)	US$m	11.6
Operating Costs		
C1 Cash Costs	US$/oz	841
All-in Sustaining Costs[1]	US$/oz	1,057

		Unit	Number
Key Financial Metrics			
US$2,400/oz Gold Price (Base Case)	Pre-tax $NPV_{5\%}$	US$m	2,279
	Pre-tax IRR	%	58%
	Pre-tax Payback Period	Years	1.5
	Post-tax $NPV_{5\%}$	**US$m**	**1,637**
	Post-tax IRR	**%**	**46%**
	Post-tax Payback Period	**Years**	**1.9**
US$3,300/oz Gold Price (Spot Case)	Pre-tax $NPV_{5\%}$	US$m	4,022
	Pre-tax IRR	%	89%
	Pre-tax Payback Period	Years	1.0
	Post-tax $NPV_{5\%}$	**US$m**	**2,893**
	Post-tax IRR	**%**	**73%**
	Post-tax Payback Period	**Years**	**1.1**

1: AISC based on gold price of US$2,400/oz and increases by ~US$54/oz at a US$3,300/oz gold price due to higher royalties.

Ore Reserve Classification Criteria

The Ore Reserve estimate is based on the Mineral Resource estimate for the NEB and BC deposits announced to the ASX on 7 August 2023, which is shown in Table 3 below together with the Mineral Resource estimates for the Fouwagbe and Sounsoun deposits announced to the ASX on 23 April 2025 (which are not considered in the DFS or Ore Reserve estimate).

Table 3: Mineral Resource estimate

Deposit	Classification	Cut-off (g/t Au)	Tonnes (Mt)	Grade (g/t Au)	Contained (koz Au)
NEB Open Pit	Indicated	0.5	78.4	1.55	3,900
	Inferred	0.5	3.1	0.91	92
	Total		**81.4**	**1.53**	**3,993**
NEB Underground	Inferred	2.0	6.8	4.07	896
NEB Total			**88.3**	**1.72**	**4,888**
BC Open Pit	Indicated	0.4	5.3	1.42	244
	Inferred	0.4	6.9	1.09	243
BC Total			**12.2**	**1.24**	**487**
NEB Area Total			**100.5**	**1.66**	**5,376**
Fouwagbe	Inferred	0.5	2.2	1.68	119
Sounsoun	Inferred	0.5	0.9	1.19	34
Argo Area Total			**3.1**	**1.54**	**153**
Total Bankan Project			**103.6**	**1.66**	**5,528**

Note: Mineral Resources are inclusive of Ore Reserves.

The Mineral Resource estimates were completed by Mr Phil Jankowski as the Competent Person. The Competent Person's statements and JORC Table 1 are included at the end of this announcement. The Company is not aware of any new information or data that materially affects the Mineral Resource estimates contained in this announcement and all material assumptions and technical parameters underpinning the Mineral Resource estimates continue to apply and have not materially changed, noting that PDI intends to appeal the Argo (and Bokoro) revocations announced on 28 May 2025 in accordance with the Mining Code, and that the Argo Inferred Mineral Resources account for just 2.8% of the Company's overall Mineral Resources.

A mining study at a DFS level was carried out on the Indicated portion of the Mineral Resource, including open pit / underground trade-off optimisation, pit optimisation, optimisation of underground stope shapes, mine design (both open pit and underground), production schedules and cost estimation and modelling.

The Ore Reserve was then estimated by taking into consideration the mining, processing, metallurgical, economic, marketing, legal, environmental, social, and governmental factors.

Where applicable, Indicated Mineral Resources are classified as Probable Ore Reserves. There are no Measured Mineral Resources, so all Probable Ore Reserves are based on Indicated Mineral Resources only, after applying appropriate modifying factors as per the guidelines. No Inferred Mineral Resources are included in the Ore Reserve estimate.

The open pit / underground trade-off optimisation conducted for NEB defined a considerably smaller open pit than the optimisation shell used in the Mineral Resource estimate. Consequently, the portion of open pit Indicated Mineral Resources below this smaller pit shell was considered as part of the underground mining study and resulted in the estimation of underground Ore Reserves for NEB.

Some of the open pit Ore Reserve economic cut-off grades for NEB are lower than the 0.5g/t cut-off grade used to report the NEB open pit Mineral Resource. However, the NEB open pit Ore Reserve designs are well within the "reasonable prospects for eventual economic extraction" optimisation shell used to define the NEB open pit Mineral Resource. The NEB open pit Mineral Resource is inclusive of the NEB underground Ore Reserve. The BC open pit Mineral Resource is inclusive of the BC open pit Ore Reserve.

As the cut-off grade for the NEB underground Ore Reserve is higher than the cut-off grade for the NEB open pit Mineral Resource, some of the lower grade parts of the open pit Mineral Resource below the DFS pit design may effectively be sterilised by the proposed underground mining operations.

Mr Ross Cheyne and Mr Julian Broomfield, the Competent Persons for the open pit and underground Ore Reserve estimates respectively, have reviewed the work undertaken to date and consider it sufficiently detailed and relevant to the deposit to allow these Ore Reserves to be classified as Probable.

Mining Method and Assumptions

Open Pit

Open pit mining will be performed as a conventional drill, blast, truck and shovel operation, which is considered appropriate for the style of the deposits. The mining method is based on three pit stages in NEB and a single pit stage each in GBE and BC. Pit slope design parameters were developed in accordance with the recommendations provided by external geotechnical consultants. Dilution and ore loss was modelled in 3D utilising a proprietary edge dilution approach by Orelogy Consultants Pty Ltd. This resulted in a global dilution of 3.6% and ore loss of 4.4% inclusive of operational ore loss of 1%. This is reflective of the broader and more continuous nature of the NEB orebody which makes up 90% of the open pit Ore Reserve. Dilution for the smaller deposits (BC and GBE) is much higher, averaging 5.4% and 8.7% respectively. The minimum mining width applied in the designs is 50m and is appropriate for the selected mining equipment fleet.

Underground

The underground mining method is a combination of transverse and longitudinal long hole stoping with engineered paste fill. Mining methodology is a top-down approach, extracting panels of three mining levels in a bottom-up mining sequence. Mining development and production will be conducted with conventional mechanised equipment incorporating jumbo / long hole development and production drill and blast, as well as load and haul with LHD and trucks. This is considered appropriate for this style of deposit.

The underground orebody will be accessed thought a conventional decline (1 in 7 gradient) from the base of the GBE pit. Ventilation will be provided through a secondary decline and vertical shafts. Geotechnical parameters were provided by external geotechnical consultants. Level intervals of 20m were adopted. Level development designs include a footwall drive located 25m from the orebody, providing ore drive access on multiple fronts. A hangingwall drive is designed to support transverse stoping and installation of geotechnical supports. Longitudinal stopes are situated along the footwall and at the north and south extremities of the orebody, with access provided via dedicated longitudinal ore drives. The design also incorporates various non-production supporting development and infrastructure.

Various mining recovery factors were applied for development (100%), stopes (85%), sill pillar stopes (75%) and the crown pillar (75%). Dilution was based on an overbreak of 0.5m on the hangingwall and footwall of each stope, which equates to between 9% and 16% dilution depending on stope dimensions.

Processing Method and Assumptions

Metallurgical testwork has been completed across three programs between 2021 and 2025 and has confirmed the ore is free milling and suitable for conventional CIL processing. The first two programs were conducted to define the key process design criteria and the third program, which was conducted in parallel with the DFS, used the outcomes of the earlier work and additional samples to complete a variability program and bulk testwork to improve the definition of process performance and design envelope, and to define carbon loading and cyanide destruction performance and design parameters, and critical equipment sizing.

The processing plant design for the DFS incorporates separate crushing circuits for weathered and fresh ore taking into account the different materials handling properties (particularly the saprolite clays). Weathered ore is fed to a single stage mineral sizer which feeds directly to the grinding circuit. Fresh ore is fed to a single stage jaw crusher, with crushed ore placed on a stockpile before being fed to the grinding circuit. The grinding circuit is a SAG/pebble crusher/ball mill circuit ("SABC") which grinds ore to 75µm (P80) ahead of treatment in a conventional gravity/leach/CIL circuit.

The process plant has been designed based on a throughput of 4.5Mtpa. Throughput ramps up from 60% of nameplate capacity in month 1 of operations to 100% of nameplate capacity by month 5.

Power demand for the comminution circuit will vary according to lithology as follows: 29.1kWh/t for fresh ore, 13.1kWh/t for laterite, saprock and shear; and 5.6kWh/t for saprolite.

Cyanide consumption is influenced by the presence of cyanide soluble copper, and review of the variability data set justifies 0.40kg/t of cyanide consumption in the leach. Industry-typical leaching residence time of 24 hours is suitable.

Lime consumption will vary by lithology as follows: 0.33kg/t for fresh ore; and 2.06kg/t for weathered ore.

Gold recovery is influenced by the gold head grade as defined by the formula set out below (and averages 92.8% over the DFS LOM).

Gold recovery (%) = 0.5145 x [head grade, g/t] + 91.533

Cyanide destruction is effective using the following operating conditions: 3g SO_2 / g WAD CN, added as $Na_2S_2O_5$; 30mg/L Cu excess, added as $CuSO_4.5H_2O$; pH of 8.5; 2-hour residence time.

For blends containing less than 50% saprolite, filtration moisture of less than 16% (w/w) is realistically achievable with pressure filtration.

Cut-off Grades

An economic cut-off grade has been applied for the Ore Reserve estimation.

The open pit Ore Reserve is reported using variable cut-off grades as processing costs vary by material type and metallurgical recovery varies by diluted head grade. Dilution has been applied in the block model and cut-off grades are reported for a diluted grade. Based on average diluted grades, the average cut-off grades by material type are: 0.41g/t for laterite; 0.38g/t for saprolite / mottled; 0.41g/t for saprock; and 0.48g/t for fresh ore. The open pit cut-off grades are based on a gold price assumption of US$1,800/oz.

The underground Ore Reserve cut-off grade is based on a calculation according to the following formula:

Cut-off Grade (g/t) = Total Cost ($/t) / (Process Recovery (%) * (Metal Price ($/g) – Other Costs ($/g))

Cut-off grade parameters for the underground Ore Reserve include government royalties of 6%, selling cost of US$4/oz, average processing recovery of 92.6%, processing cost of US$22.34/t (based on underground material being almost exclusively fresh), G&A cost of US$2.28/t and underground mining cost of US$74.47/t. The underground cut-off grade was estimated to be 2.0g/t.

The cut-off grades are based on a gold price assumption of US$1,800/oz.

Estimation Methodology

Estimation for the open pit Ore Reserve was based on optimisations conducted in GEOVIA Whittle™ software. Input parameters for the pit optimisations were based on data from PDI and external parties, such as geotechnical reports and metallurgical results, and a conservative gold price of US$1,800/oz. Designs based on optimised pit shells and subsequent schedules were completed using the Hexagon MinePlan Schedule Optimizer ("MPSO") scheduling tool.

The estimation for the underground was based on underground stope optimisations conducted in SO™ (Deswik proprietary software) based on input parameters, using the part of the NEB block model situated beneath the NEB open pit design. An underground mine design and schedule was created using Deswik CAD based on the SO™ stope shapes and incorporating geotechnical recommendations and all required capital development to access the orebody and operate the mine.

The Indicated Mineral Resources within the open pit and underground designs and above the relevant cut-off grades are defined as Ore Reserves.

Material Modifying Factors

General mining infrastructure required for the Project will include a ROM pad, tailings storage facility ("TSF"), topsoil and waste rock dumps, stockpiles, haul roads, workshops, processing plant, power station and offices. 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. The establishment of this infrastructure is included in the capital cost estimate or the operating cost estimate (where assumed to be provided by a contractor as part of a rate for services or a build-own-operate arrangement) for the Project.

Power for the Project will be generated through a heavy fuel oil ("HFO") power plant in combination with a solar farm. 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.

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 site will be predominately in Kouroussa although an accommodation village is included for the expatriate and senior staff. The camp will be of a sufficient size for the workforce requirements and will be expanded with temporary facilities for the construction period.

The Project is located within the Peripheral Zone of the Upper Niger National Park. PDI completed an ESIA for the Project in April 2024. The MEDD approved the ESIA and issued the ECC for the Project on 17 January 2025. An Exploitation Permit application was submitted on 31 January 2025 and PDI is collaborating closely with the MMG to complete the review process, which has progressed to an advanced stage.

Design and control measures for biodiversity have been factored into the mine plan to avoid sensitive biodiversity features in the Project area. Mitigation will be managed through the Biodiversity Action Plan to identify and protect set-asides, biological corridors, restoration of habitats, and biodiversity offsets. Geochemical testwork has indicated that the majority of samples are non-acid forming or acid consuming, and as such, only high-level controls are required to manage the risk of acid generation in the mine waste rock. Proposed waste rock dumps have been designed to minimise impact and incorporate boundary sediment traps and sumps to collect surface water runoff. A filtered TSF has been adopted due to the Project's location within the Peripheral Zone of the Upper Niger National Park and proximity to the regional centre of Kouroussa. The filtered TSF will be fully lined with HDPE, and as such, has a low risk of any leachate from the TSF entering groundwater.

The Project's social area of influence is mainly over a low-density rural area with three small villages (Bankan, Kignédouba, and Sokoro), extending to Kouroussa to the east, the closest large town, with a population of around 50,000. Artisanal gold mining and agriculture are the main occupations. The Project is expected to generate positive social impacts at a local, regional and national level. Social impacts have been identified from land acquisition and access restrictions that will affect agricultural land, land used for grazing, access to ecosystem services and artisanal mining sites. Impacts on community cohesion and health have been identified from the in-migration of people. PDI is developing and will implement plans and procedures to manage and mitigate social risks including but not limited to livelihood restoration, stakeholder engagement and community health. Stakeholder consultation will also continue through the Project construction and operations phases.

Capital and operating cost estimates have been prepared to sufficient accuracy for a DFS confidence level (±15%). The capital cost estimate is a bottom-up estimate generated from designs of sufficient detail to support a Class 3 estimate and market information with only a small percentage of costs priced on industry norms and typical estimating factors. The operating cost estimate is a bottom-up estimate, incorporating mining contractor budget quotations for both open pit and underground mining. All significant and measurable items are itemised with smaller items factored in as per industry practice. A 5% royalty plus a 1% local development contribution has been included. Treatment and refining charges are based on pricing from a global refining company.

ASX LISTING RULE 5.16 AND 5.17 REQUIREMENTS

The material assumptions on which the production target for the Project and the forecast financial information derived therefrom are based are detailed in the DFS Summary Report, which is included in this announcement.

The production target is based on Probable Ore Reserves (90.6% contained gold basis) and Inferred Mineral Resources (9.4% contained gold basis) that have been prepared by Competent Persons in accordance with the requirements of the JORC Code (2012). Refer also to Cautionary Statement below regarding Inferred Mineral Resources.

COMPETENT PERSONS STATEMENTS

The information in this announcement that relates to Mineral Resources is based on and fairly represents information compiled by Mr Phil Jankowski (MSc) a Competent Person who is a Fellow of the Australasian Institute of Mining and Metallurgy. Mr Jankowski is a full-time employee of ERM Australia Consultants Pty Ltd. Mr Jankowski has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the Joint Ore Reserves Committee (JORC) "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves". Mr Jankowski consents to the inclusion in this announcement of the matters based on his information in the form and context in which it appears.

The information in this announcement that relates to open pit Ore Reserves is based on and fairly represents information compiled by Mr Ross Cheyne, BEng (Mining), a Competent Person who is a Fellow of the Australasian Institute of Mining and Metallurgy. Mr Cheyne is a full-time employee of Orelogy Consulting Pty Ltd. Mr Cheyne has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the JORC "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves". Mr Cheyne consents to the inclusion in this announcement of the matters based on his information in the form and context in which it appears.

The information in this announcement that relates to the underground Ore Reserves is based on and fairly represents information compiled by Mr Julian Broomfield, BEng (Mining), a Competent Person who is a Member of the Australasian Institute of Mining and Metallurgy. Mr Broomfield is a full-time employee of Orelogy Consulting Pty Ltd. Mr Broomfield has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the JORC "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves". Mr Broomfield consents to the inclusion in this announcement of the matters based on his information in the form and context in which it appears.

The information in this announcement that relates to metallurgy, processing, non-mining operating costs, non-mining capital costs, project delivery and environmental aspects is based on and fairly represents information reviewed and compiled by Mr Stewart Watkins, BEng (Hons), a Competent Person who is a Fellow of the Australasian Institute of Mining and Metallurgy. Mr Watkins is a full-time Director of Dhamana Consulting Pty Ltd. Mr Watkins has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the JORC "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves". Mr Watkins consents to the inclusion in this announcement of the matters based on his information in the form and context in which it appears.

FORWARD LOOKING STATEMENTS AND IMPORTANT NOTICE

This announcement contains certain forecasts, projections and forward-looking statements. Forward-looking statements may generally be identified by the use of forward-looking terminology, including, without limitation, the terms "believes", "estimates", "anticipates", "expects, "predicts", "intends", "plans", "goals", "targets", "aims", "outlook", "guidance", "forecasts", "may", "will", "would", "could" or "should" or, in each case, their negative or other variations or comparable terminology. Such forward-looking statements involve known and unknown risks, uncertainties and other factors which because of their nature may cause the actual results or performance of PDI to be materially different from the results or performance expressed or implied by such forward-looking statements. Such forward-looking statements are based on numerous assumptions regarding PDI's present and future operations and the political and economic environment in which PDI will operate in the future, and are not guarantees or predictions of future performance. Although PDI believes that its expectations, estimates and forecast outcomes are based on reasonable assumptions it cannot give any assurances that these will be achieved. Unless stated otherwise, forward-looking statements speak only as at the date of this announcement or the DFS (as applicable).

To the maximum extent permitted by applicable laws, PDI and its related bodies corporate and their respective officers, directors, employees, agents and advisers ("Related Parties") make no representation and can give no assurance, guarantee or warranty, expressed or implied as to, and take no responsibility and assume no liability for the authenticity, validity, accuracy, suitability or completeness of, or any errors in or omissions from, any information, statement or opinion contained in this announcement and without prejudice, to the generality of the foregoing, the achievement or accuracy of any forecasts, projections or other forward-looking statements contained or referred to in this announcement. Except as required by applicable laws, PDI and its related bodies corporate and their respective Related Parties disclaim any obligation update or revise any forward-looking statement in this announcement.

Investors should make and rely upon their own enquiries before deciding to deal in PDI's securities.

CAUTIONARY STATEMENT

The DFS documented in this announcement is considered to have a ±15% level of accuracy.

The DFS is based on a previously announced Mineral Resource estimate (refer to ASX release "Bankan Mineral Resource Increases to 5.38Moz" dated 7 August 2023) and an Ore Reserve estimate has been prepared as part of the DFS. The Ore Reserve and Mineral Resource estimates have been prepared by Competent Persons in accordance with the 2012 JORC Code.

The production targets and forecast financial information in this announcement and the DFS Summary Report are based on 90.6% Probable Ore Reserves and 9.4% Inferred Mineral Resources (on a contained gold basis). Inferred Mineral Resources, which are only included from the lower part of the NEB underground mine, are processed predominantly in years 5 to 9 and represent only 0.5% of contained gold processed in the first 4 years. There is a low level of geological confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of Indicated Mineral Resources or that the production targets will be realised.

The DFS is based on the material assumptions outlined in the DFS Summary Report enclosed with this announcement. This includes assumptions about the availability of funding. While PDI considers the material assumptions to be based on reasonable grounds, there is no certainty that they will prove correct or that the range of outcomes indicated by the DFS will be achieved. Investors should note that there is no certainty that PDI will be able to raise that amount of funding when needed. It is possible that such funding may only be available on terms that may be dilutive to or otherwise affect the value of PDI's existing shares. Given the uncertainties involved, investors should not make any investment decisions based solely on the results of the DFS.

- END -

This announcement is authorised for release by PDI Managing Director, Andrew Pardey.

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ABOUT PREDICTIVE DISCOVERY

PDI's strategy is to identify and develop gold deposits within the Siguiri Basin, Guinea. The Company's key asset is the Tier-1 Bankan Gold Project. A Mineral Resource Estimate of 5.53Moz has been defined to date at the NEB (4.89Moz), BC (487koz), Fouwagbe (119koz) and Sounsoun (34koz) deposits, making Bankan the largest gold discovery in West Africa in a decade. The Project is also highly prospective for additional discoveries.

PDI completed a Definitive Feasibility Study for the Bankan Project in June 2025, outlining a ~250koz per annum operation over 12.2 years, an Ore Reserve estimate of 2.95Moz and strong financials. The Government of Guinea approved the Project's Environmental & Social Impact Assessment and issued the Environmental Compliance Certificate in January 2025 and the Exploitation Permit application is at an advanced stage of the Government's review process.

Bankan Gold Project
Definitive Feasibility Study
Summary Report

24 June 2025

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ACRONYMS AND ABBREVIATIONS

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
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
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 rise
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
PFS	Prefeasibility study
PMC	Project management consultant
PPA	Power purchase agreement
PV	Photovoltaic
QAQC	Quality assurance / quality control
RAR	Return air rise
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
WHO	World Health Organisation
WRD	Waste rock dump
WWTP	Wastewater treatment plant
XRD	X-ray diffraction

UNITS AND SYMBOLS

%	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
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

1 INTRODUCTION

Predictive Discovery Limited's (PDI or the Company) Bankan Gold Project (the Project) is located in the Siguiri Basin in the northeast of the Republic of Guinea (Guinea), West Africa. PDI commenced assembling the Bankan Gold Project permits and other permits within the Siguiri Basin in mid-2019. Exploration commenced shortly after, and PDI made significant gold discoveries at NEB and BC in April 2020. Exploration and resource definition drilling programmes completed since have defined an Indicated and Inferred Mineral Resource in accordance with the JORC Code (2012) of 103.6 Mt at 1.66 g/t for 5.53 Moz of contained gold, of which 4.14 Moz or 75% lies in the Indicated category.

PDI published a pre-feasibility study (PFS) on the development of the Project in April 2024. A major de-risking milestone was achieved in January 2025 when the *Ministère de l'Environnement et du Développement Durable* (Ministry of Environment and Sustainable Development or MEDD) approved the Environmental and Social Impact Assessment (ESIA) and issued the Certificate of Environmental Compliance (ECC) for the Project. On 31 January 2025, PDI and its 100% owned subsidiary, Mamou Resources SARLU (Mamou) submitted exploitation permit applications for 50% of the Kaninko and Saman permit areas to the Ministry of Mines and Geology (MMG) and the Centre de Promotion et de Développement Minier (CPDM) in accordance with Guinean mining law. The applications are at an advanced stage and are still being processed and the Company is not aware of any immediate obstacles to the granting of the exploitation permits..

Based on positive outcomes of the PFS, and the progress on the environmental approval, PDI commenced a definitive feasibility study (DFS) for the Project in July 2024.

The DFS commenced with a strategic review, resulting in the following outcomes:

- Impact of updated geotechnical information, which led to being able to adopt more aggressive pit wall angles compared to the PFS.
- Trade-off between open pit and underground mining which determined the optimal transition depth at approximately 250 m, which is results in a smaller open pit and a significantly reduced strip ratio compared to the PFS.
- Review of timing for commencement of underground mining production which showed that accessing the high-grade underground ore at the commencement of production improved economic outcomes.
- Assessment of the underground mining approach, considering transverse, longitudinal and a hybrid stoping strategies, with a hybrid strategy adopted reducing ore loss and dilution from the underground mine.
- Scale of operations review, which selected 4.5 Mtpa processing rate as the most appropriate scale.

As a result of this strategic review, the DFS considers the development of open pit mines at NEB, Gbengbeden (GBE) and BC as well as an underground mine at NEB, a 4.5 Mtpa conventional carbon-in-leach (CIL) processing plant, a filtered tailings storage facility and various supporting facilities and infrastructure.

The DFS has defined an updated Probable Ore Reserve in accordance with the JORC Code (2012) of 51.6 Mt at 1.78 g/t containing 2.95 Moz of gold.

A mining inventory, which includes only 5.3% of material and 9.4% of contained gold from Inferred Mineral Resources in the later stages of the proposed underground mine, has been developed for the Project which totals 54.5 Mt at 1.86 g/t containing 3.26 Moz of gold. It should be noted that there is a low level of geological confidence associated with the included Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of Indicated Mineral Resources or that the production target associated with the mining inventory itself will be realised.

Scheduling of the mining inventory produces an average of approximately 249,000 oz per annum over a >12-year mine life at an all-in sustaining cost (AISC) of US$1,057 per oz.

A capital cost estimate for the Project has been prepared as an Association for the Advancement of Cost Engineering (AACE) Class 3 estimate with a level of accuracy of ±15%. Capital costs over a 24-month development period have been estimated at US$463m. This cost estimate includes pre-production operating costs, construction owners team and project management costs, operational readiness costs and a contingency which has been estimated at 10.5%. This estimate excludes corporate costs, Project funding costs and escalation.

At a consensus gold price assumption of US$2,400 per oz, the Project delivers a post-tax net present value at 5% discount (NPV5%) of US$1,637 million and an internal rate of return (IRR) of 46%, achieving payback in 1.9 years. Financial outcomes improve significantly at the spot gold price of US$3,300. Key project metrics are included in Table 1.1 and key financial metrics in Table 1.2.

Table 1.1: 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	10.8
Underground Grade	g/t	3.77
Underground Contained Gold	koz	1,309
Total Ore Mined	Mt	54.5
Average Grade	g/t	1.86
Total Contained Gold	koz	3,260
Contained Gold from Inferred Mineral Resources	%	9.4

	Units	
Processing		
Mine Life	Years	12 years and 2 months
Processing Rate	Mtpa	4.5
Total Ore Processed	Mt	54.5
Average Processing Recovery	%	92.8
Total Gold Production	koz	3,026
Average Gold Production	koz pa	249
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	208.6
Closure Costs (excluding salvage)	US$m	39.6
Operating Cost		
C1 Cash Cost	US$/oz	841
All-in Sustaining Cost	US$/oz	1,057

Table 1.2: Key Financial Metrics

	Units	Base Case US$2,400/oz	Spot Price US$3,300/oz
Pre-Tax $NPV_{5\%}$	US$m	2,279	4,022
Pre-Tax IRR	%	58	89
Pre-Tax Payback Period	Years	1.6	1.0
Post-Tax $NPV_{5\%}$	US$m	1,637	2,893
Post-Tax IRR	%	46	73
Post-Tax Payback Period	Years	1.9	1.1

Development of the Project will be preceded by an eight-month execution readiness phase during which time project execution plans will be finalised, engineering and consulting groups will be engaged, front-end engineering and design will be progressed, final site investigations will be initiated, and long-lead equipment and critical path contracts will be tendered.

In parallel with these execution readiness activities, economic resettlement will be undertaken, following grant of the exploitation permits, to provide full access to the Project site.

Project funding arrangements, which are already in motion, will also be progressed and sufficiently finalised prior to commencement of construction and significant site works.

2 DEFINITIVE FEASIBILITY STUDY CONTRIBUTORS

A number of independent consultants have contributed to the DFS with key consultants and their responsibilities outlined in Table 2.1. These consultants were managed by the PDI executive who was assisted by Dhamana Consulting Pty Ltd (Dhamana Consulting), a specialist project management and environmental consultancy based in Perth, Western Australia.

Table 2.1: Study Contributors

Area	Contributor
Geology & Mineral Resource Estimation	ERM
Mining Reserve Estimation, Mine Planning & Design	Orelogy
Mine Geotechnical	PETER O'BRYAN & Associates – Consultants in Mining Geomechanics
Metallurgical Testwork	ALS; Dhamana consulting
Process Plant & Infrastructure Design	DRA; Dhamana consulting
Tailings Management & Site Geotechnical Engineering	Knight Piésold Consulting
Hydrogeology & Hydrology	Australasian Groundwater & Environmental Consultants; Hydro Logic
Power Supply & Distribution	ECG Engineering
Capital Cost Estimation	DRA; Orelogy; Australasian Groundwater & Environmental Consultants; ECG Engineering; Dhamana consulting
Operating Cost Estimation	DRA; Orelogy; ECG Engineering; FUTURE/C (Clarity | Sustainability | Ethics); Dhamana consulting; predictive discovery
Financial Analysis	Infinity Corporate Finance – Partners for Growth
Environmental	ERM; predictive discovery
Mine Closure Estimation	Orelogy; Australian Mining Advisors; Dhamana consulting
Overall Study Management	Dhamana consulting

3 PROJECT DESCRIPTION

3.1 Project Location

The Project is located in the northeast part of Guinea, as depicted in Figure 3.1. It is approximately 450 km east-northeast of Guinea's capital city, Conakry, in Kouroussa Prefecture.

Figure 3.1: Project Location

At a regional level, as shown in Figure 3.2, 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.

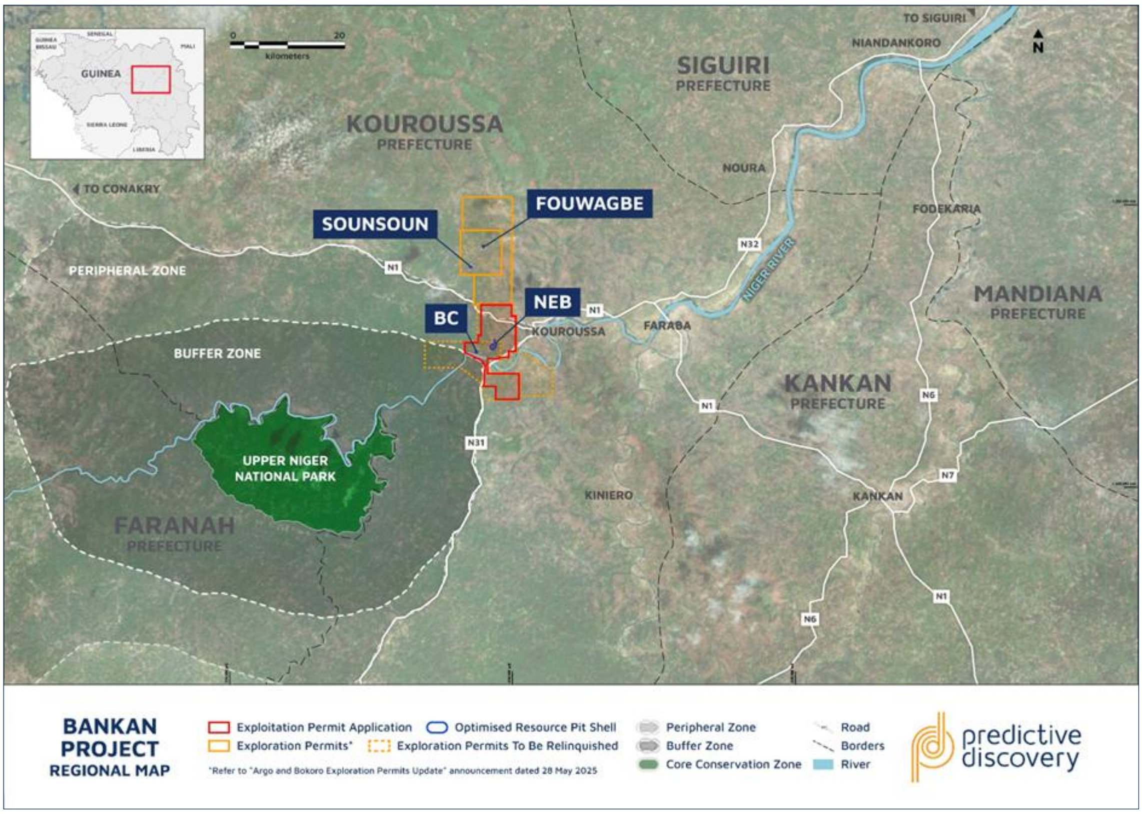

Figure 3.2: Project Region

3.2 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 3.3.

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. The PDI Executive Director – Legal and ESG indicates 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 3.3 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.

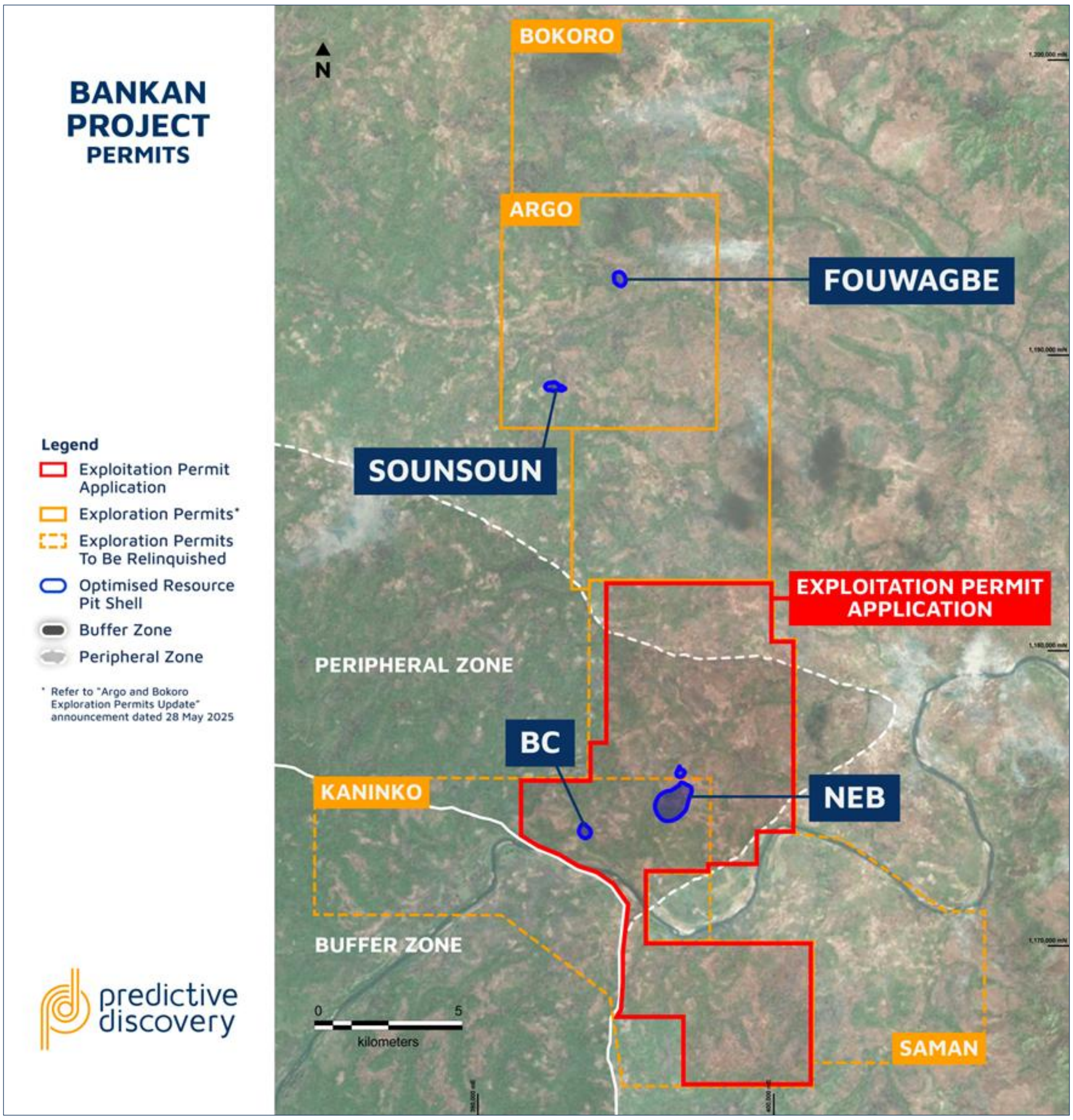

Figure 3.3: Project Permits

3.3 Project Layout and Access

3.3.1 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.

3.3.2 Project 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 3.4

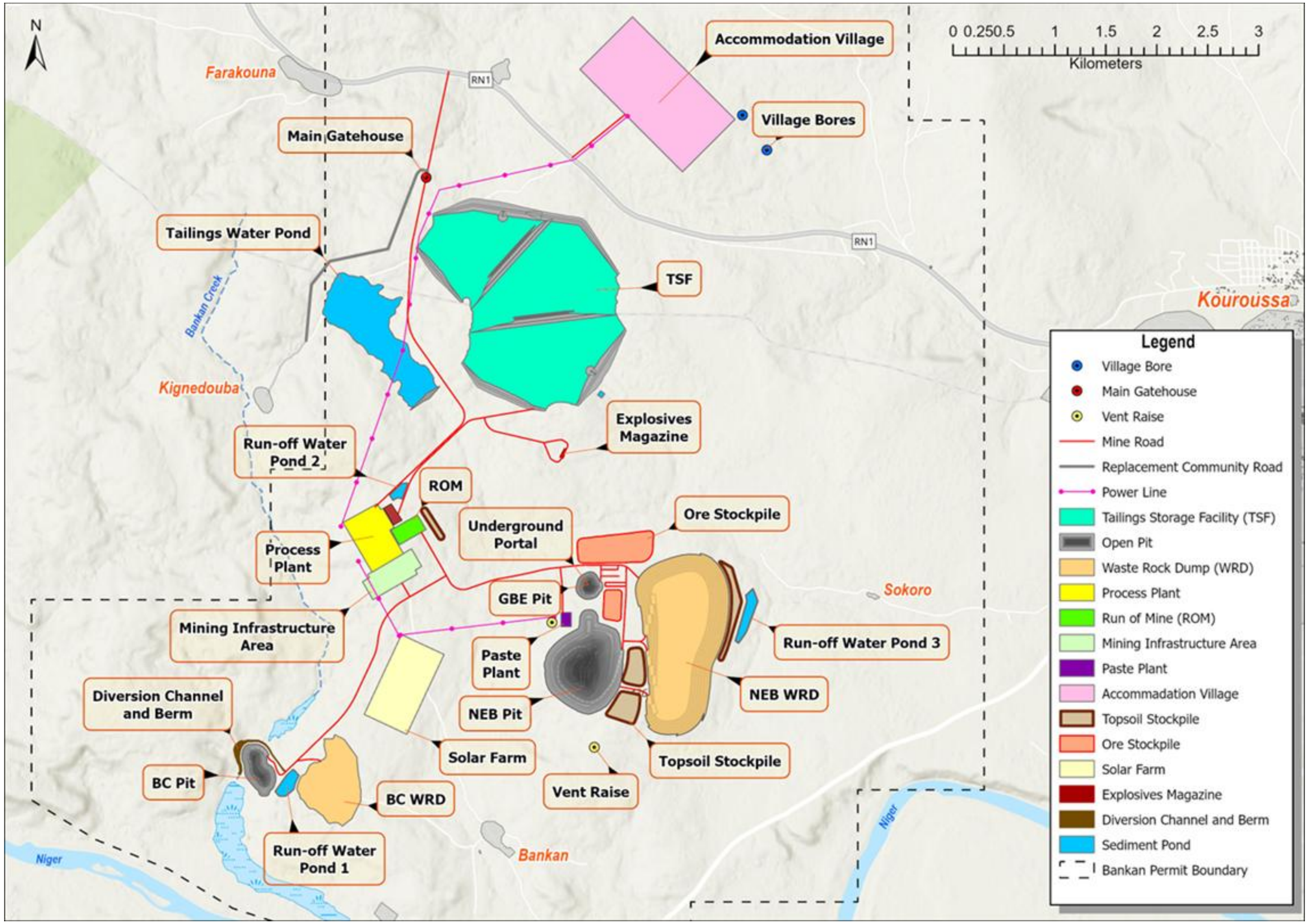

Figure 3.4: Overall Site Layout

3.4 Climate

The Project site has a tropical savannah climate. There is a distinct wet and dry season, with the wet season spanning from May to October and the dry season from November to April. Mean annual precipitation is approximately 1,375 mm with significant variability across the year. Evapotranspiration 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. Prevailing winds are generally light and come from the southwest. The strongest breezes are in the wet season.

3.5 Local Resources and 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. Various villages are also located near the Project which, in addition to Kouroussa, could provide labour for the Project. There is a long history of small-scale artisanal gold mining in the region.

Grid power in the region is currently limited, although Guinea has long-term plans to increase the availability of grid power within the country, including developing the high voltage Linsan-Fomi transmission line, which has a planned alignment close to the Project site. 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. Hydrogeological investigations conducted for the DFS suggest that the Project will be net water positive when mine dewatering and water capture on the TSF is considered.

PDI has existing facilities for its exploration activities, including an accommodation camp, offices and core sheds. The existing accommodation camp has approximately 60 beds (with the potential for additional beds) plus supporting facilities, including two mess buildings, two laundry buildings, a site medical clinic, security, water wells and two generators to power the facility. It is expected to be used for initial construction and ongoing exploration activities.

3.6 Regulatory Environment

On 5 September 2021, the political and legal order of Guinea changed with the arrival of new authorities in power. The 2020 constitution was replaced by the Transitional Charter, published on 27 September 2021, which serves as a constitution pending the drafting of a new constitution. However, national laws and international treaties in force before the arrival of the new administration are maintained and continue to apply in all their terms.

The Transitional Charter contains provisions reaffirming and respecting fundamental rights and freedoms and managing the transition to achieve a state governed by the rule of law with strong institutions. The charter guarantees the right to property against any form of expropriation, except for expropriation in the public interest. The legal framework for the Project consists of national laws in force before 5 September 2021, and these laws continue to apply.

The administrative framework applicable to the Project includes, among others, national laws and regulations administered by the MMG, the MEDD and local authorisations relating to prefectures, regions and urban and rural communes.

The key national laws that apply to the Project include the following:

- Mining Code (law L/2011/006/CNT of September 2011, amended by law L/2013/053/CNT of 8 April 2013).
- Environmental Code (law L/2019/0034/AN of 4 July 2019).
- Wildlife Code (law 2018/0049/AN OF June 2018).
- Labour Code (law L/2014/072/CNT of 10 January 2014).

In addition, due to the Projects location within the Peripheral Zone of the Upper Niger National Park, the Project is subject to additional regulations on natural protected areas which have been taken into account in the preparation and subsequent approval of the Project's ESIA and the granting of the ECC.

4 GEOLOGY AND MINERAL RESOURCES

The Project is in an area of greenstones near the southwest margin of the Siguiri Basin, which is situated in upper Guinea and southwest Mali. 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.

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.

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

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 Competent Person considers the entire dataset acceptable for resource estimation subject to the preceding comments regarding the analytical accuracy and precision.

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 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$1800/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 overall Mineral Resource estimate for the Project is tabulated in Table 4.1. The Mineral Resource estimate has been classified in accordance with the JORC Code (2012). The Mineral Resource Statement is a global estimate of in-situ tonnes and grade. It is suitable for reporting as a global resource, however, the relatively wide sampling grid has produced a model 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.

Table 4.1: Mineral Resource Estimate

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
	Total			88.3	1.72	4,888
BC	Open Pit	Indicated	0.4	5.3	1.42	244
		Inferred	0.4	6.9	1.09	243
	Total			12.2	1.24	487
Fouwagbe	Open Pit	Inferred	0.5	2.2	1.68	119
Sounsoun	Open Pit	Inferred	0.5	0.9	1.19	34
Total Bankan Project				**103.6**	**1.66**	**5,528**

Notes:

1. The Mineral Resources is estimated with all drilling data available at 29th July 2023 (NEB and BC) and 25th of February 2025 (Fouwagbe and Sounsoun).
2. The Mineral Resource is reported in accordance with the JORC Code 2012 Edition.
3. The Competent 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$1,800 per ounce Au (NEB and BC) or US$2,300 per ounce Au (Fouwagbe and Sounsoun) and process recovery of 94%. The NEB underground Mineral Resource is constrained by the high-grade domain below the NEB optimised pit shell.
5. 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.
6. Rounding may lead to minor apparent discrepancies.
7. While included in the Bankan Project Mineral Resource estimate, the Mineral Resources reported at Fouwagbe and Sounsoun are not considered further in the DFS.
8. 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.

4.1 Further Exploration Potential

All the deposits are open at depth and along strike. The Project's permit 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 Mineral Resource is open at depth. Resource development drilling of this target from surface requires a large number of very deep drillholes (greater than 900 m). A more attractive option would be drilling from underground positions established in the hanging wall 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. In addition, there are other prospective exploration targets situated in the NEB area.

Figure 4.1 shows the regional interpreted geology across the Project permits, current Mineral Resources and areas prospective for exploration.

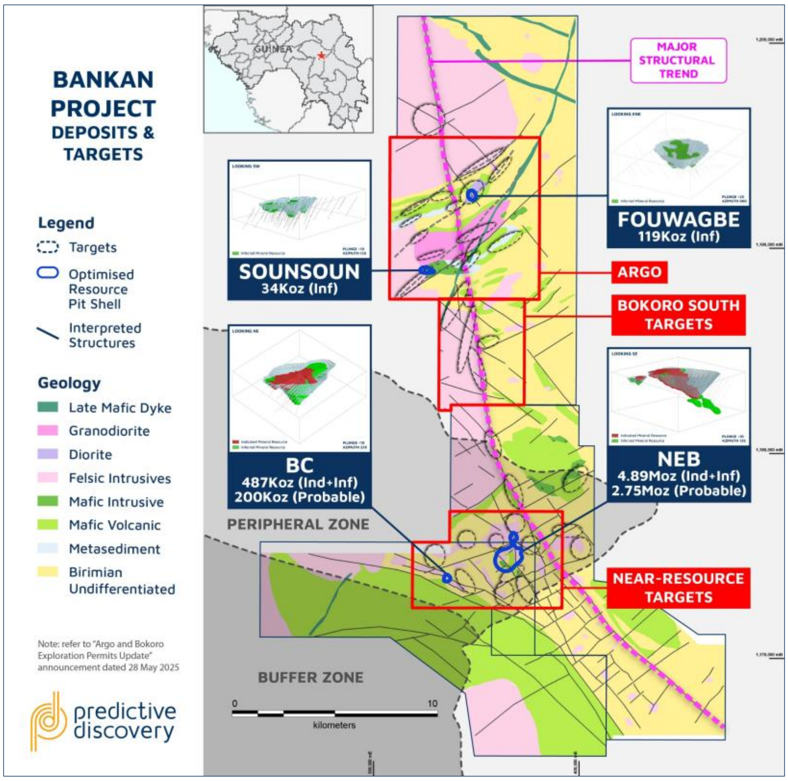

Figure 4.1: Regional Interpreted Geology, Resources and Targets

5 GEOTECHNICAL

5.1 Mining

5.1.1 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.

5.1.2 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.

5.1.2.1 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 5.1, for the NEB pit comprise:

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

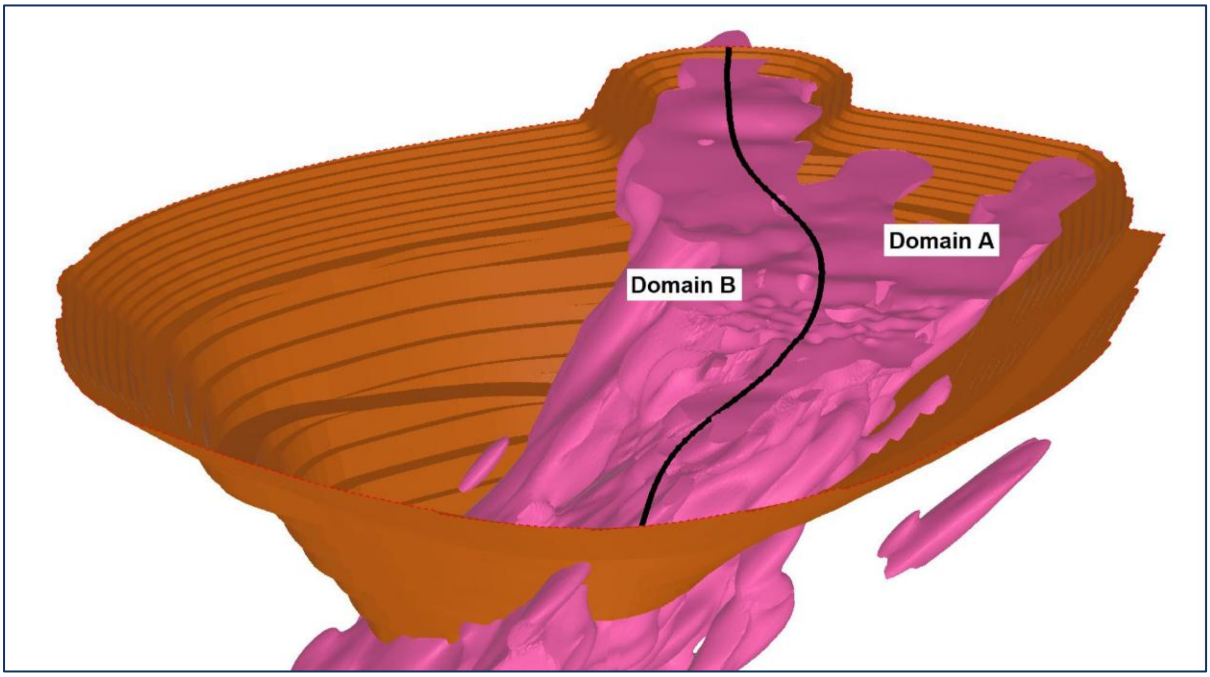

Figure 5.1: NEB Pit Domains

Base case geotechnical design parameters have been developed for the NEB pit as outlined in Table 5.1 and shown visually in Figure 5.2 and Figure 5.3.

Table 5.1: 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°.

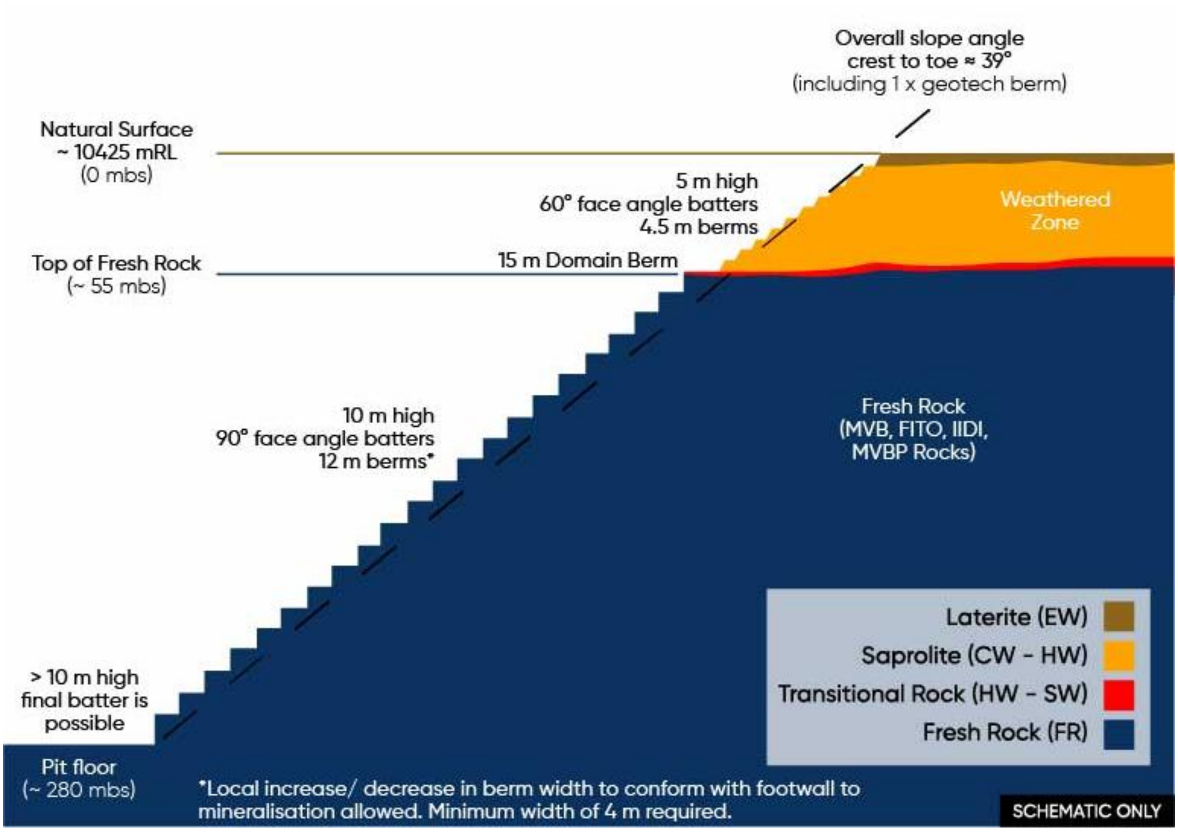

Figure 5.2: NEB Pit Domain A Wall Design Parameters

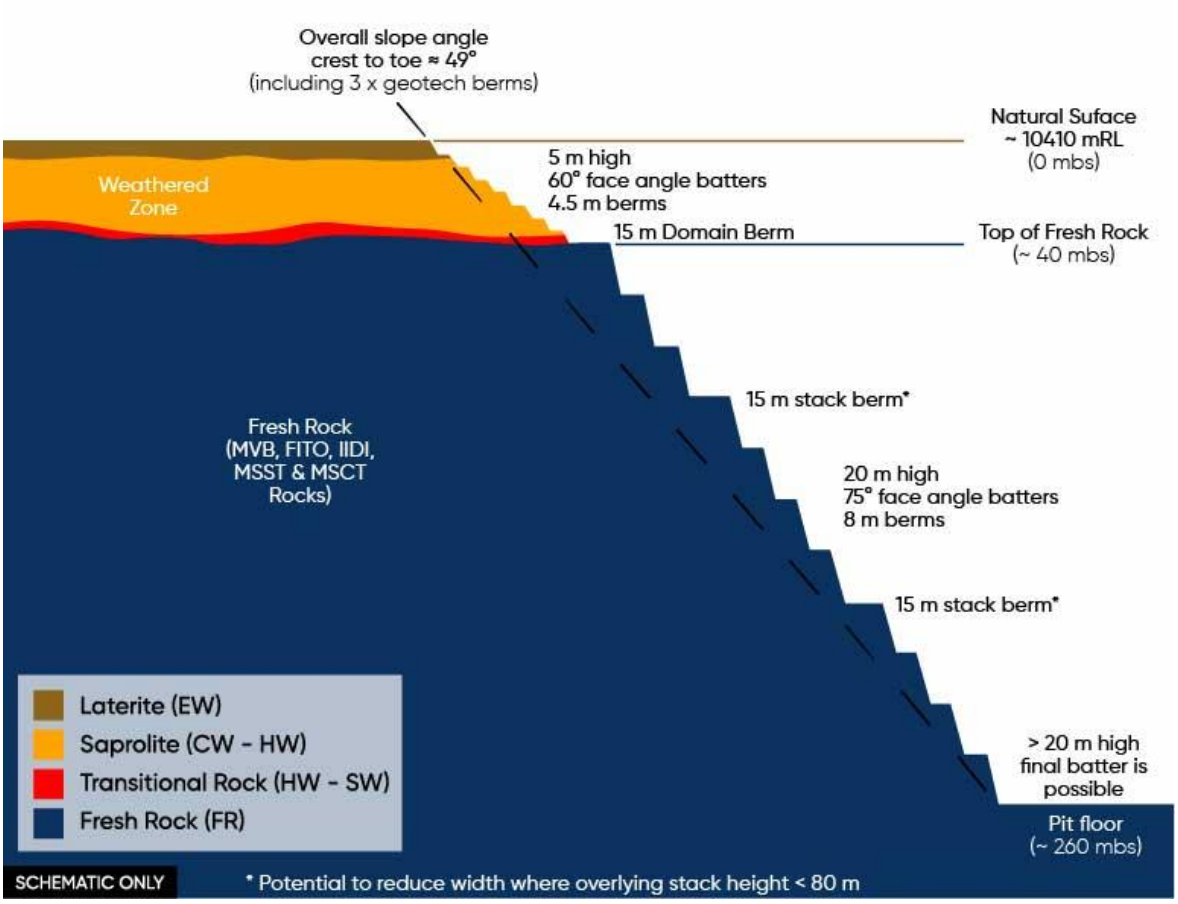

Figure 5.3: NEB Pit Domain B Wall Design Parameters

5.1.2.2 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 5.4, for the BC pit comprise:

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

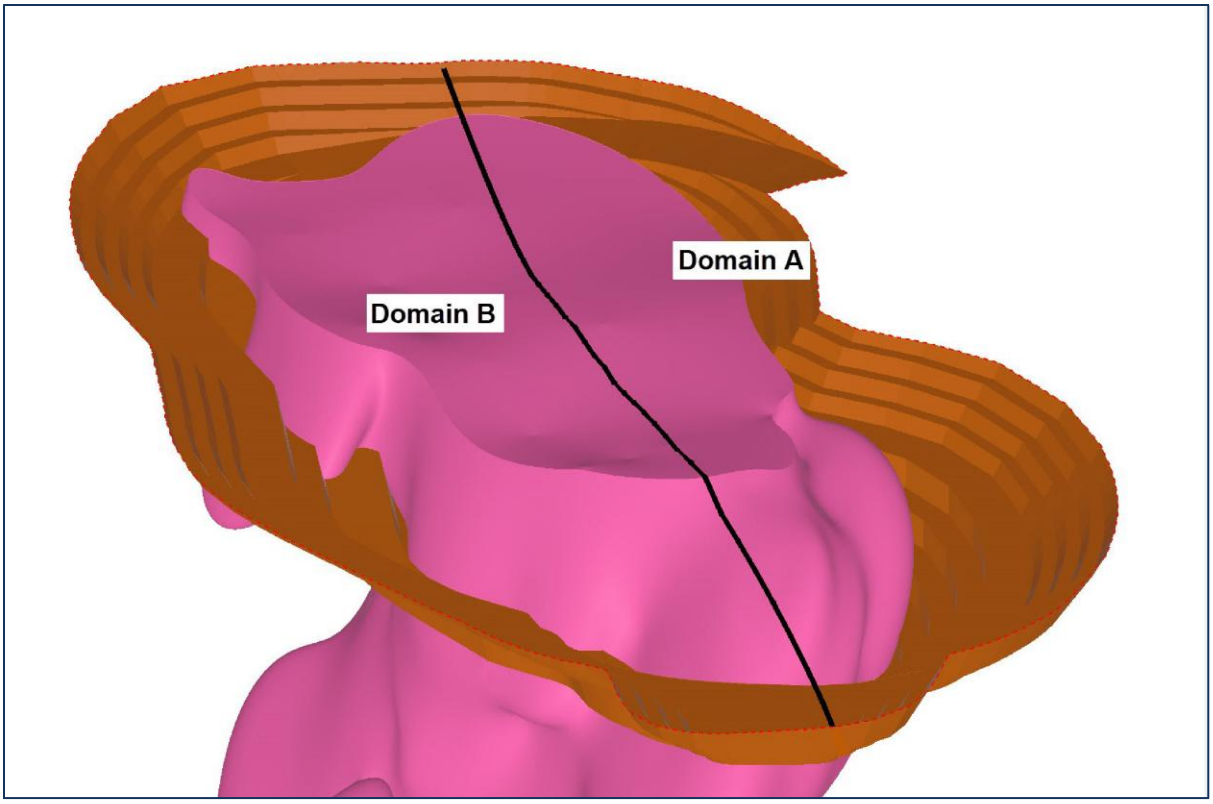

Figure 5.4: BC Geotechnical Pit Domains

Base case geotechnical design parameters have been developed for the BC pit as outlined in Table 5.2 and shown visually in Figure 5.5 and Figure 5.6.

Table 5.2: 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°.

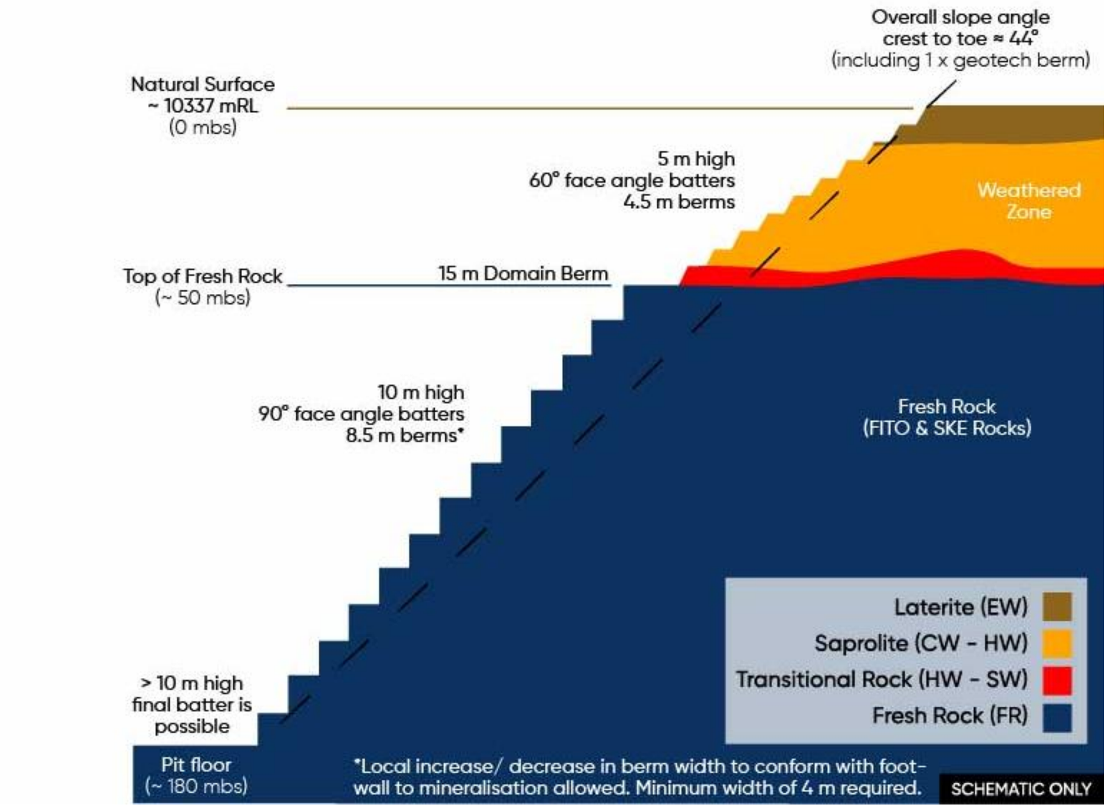

Figure 5.5: BC Pit Domain A Wall Design Parameters

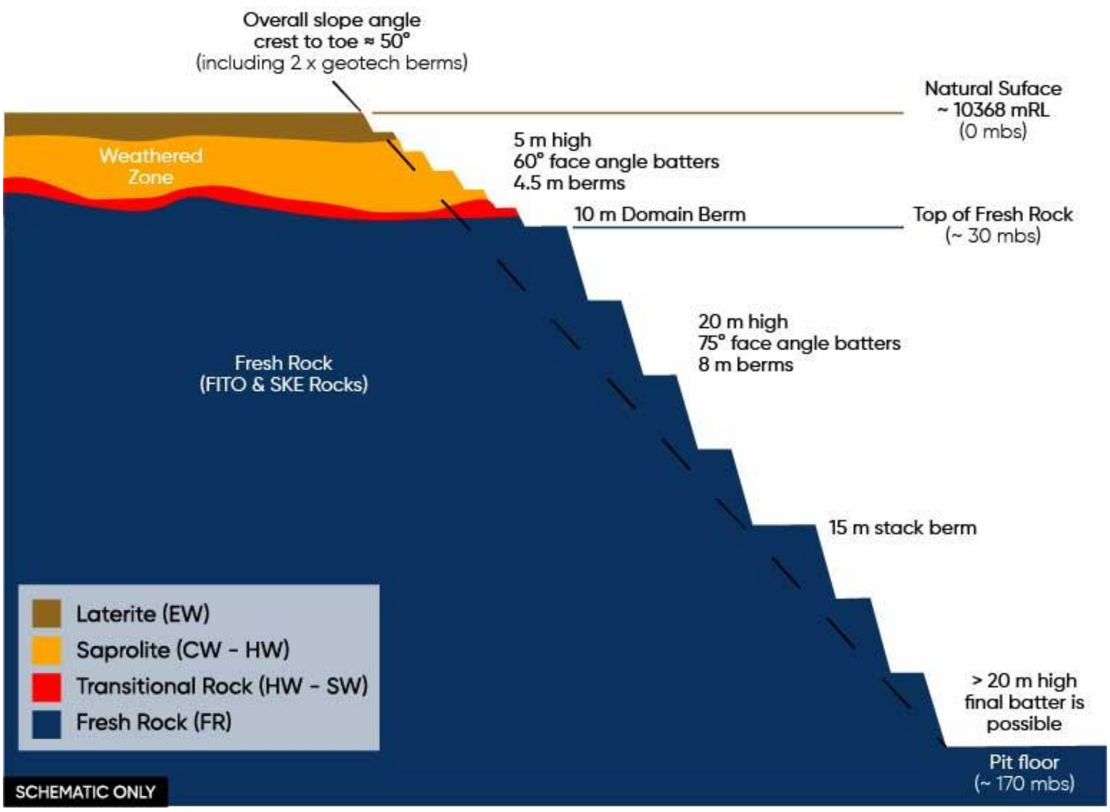

Figure 5.6: BC Pit Domain B Wall Design Parameters

5.1.2.3 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 5.3 and shown visually in Figure 5.7.

Table 5.3: 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	

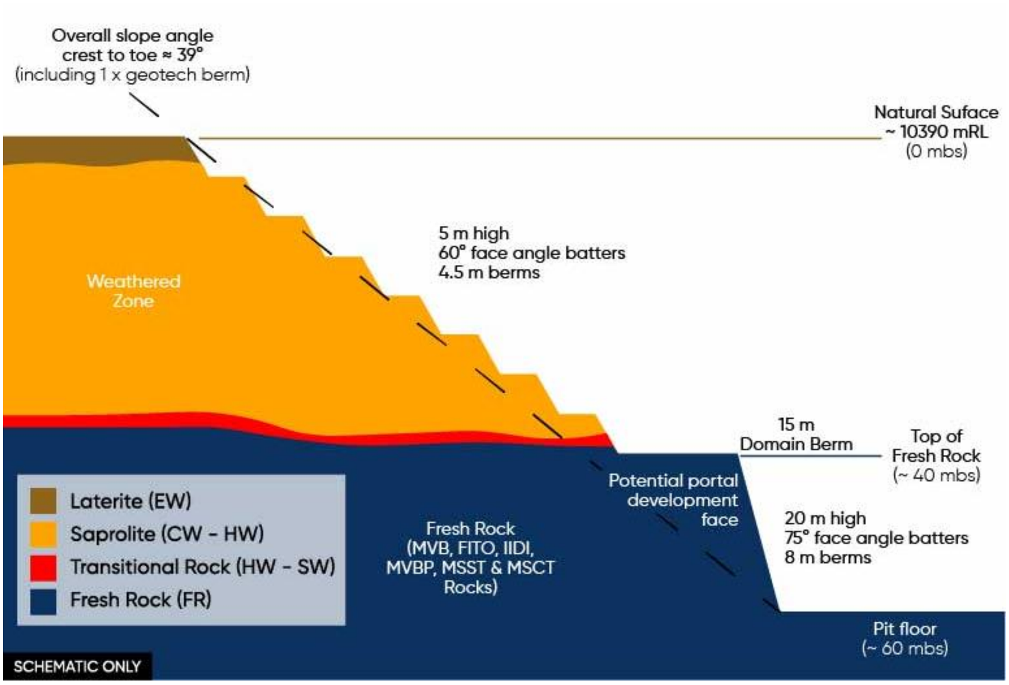

Figure 5.7: GBE Pit Wall Design Parameters

5.1.3 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 5.4 have been developed for underground stope design and configurations.

Table 5.4: NEB Underground Geotechnical Stope Design Parameters

Depth (mRL)	Maximum HW Height (m)	Maximum Strike Length (m)	Maximum Backs Width (m)
10160 – 10020	20	25	10
10020 – 9880	20	22	10
9880 – 9740	20	17	10
9740 – 9620	20	12	10

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 5.5.

Table 5.5: 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 5.0 m High x 5.0 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 (floor to 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	• ≥ 75 mm FRS across backs and sidewall (floor to 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.

5.2 Infrastructure

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.

6 HYDROGEOLOGY AND HYDROLOGY

The infrastructure of interest to the hydrogeological and hydrological assessments, to be developed alongside the mining includes, (but not limited to):

- WRD areas.
- Processing plant and associated facilities.
- Filtered TSF.
- Haul roads between the pits and plant area.
- Accommodation village.
- Supporting infrastructure.

6.1 Hydrogeology

6.1.1 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.

6.1.2 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.

6.1.3 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.

6.1.3.1 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.

6.1.4 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 the majority of 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, a number of 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.

6.1.5 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 also to provide water for construction activities.

6.2 Hydrology

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.

6.2.1 Climate

6.2.1.1 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.

6.2.1.2 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.

6.2.1.3 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.

6.2.1.4 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.

6.2.1.5 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.

6.2.2 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.

6.2.3 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 6.1.

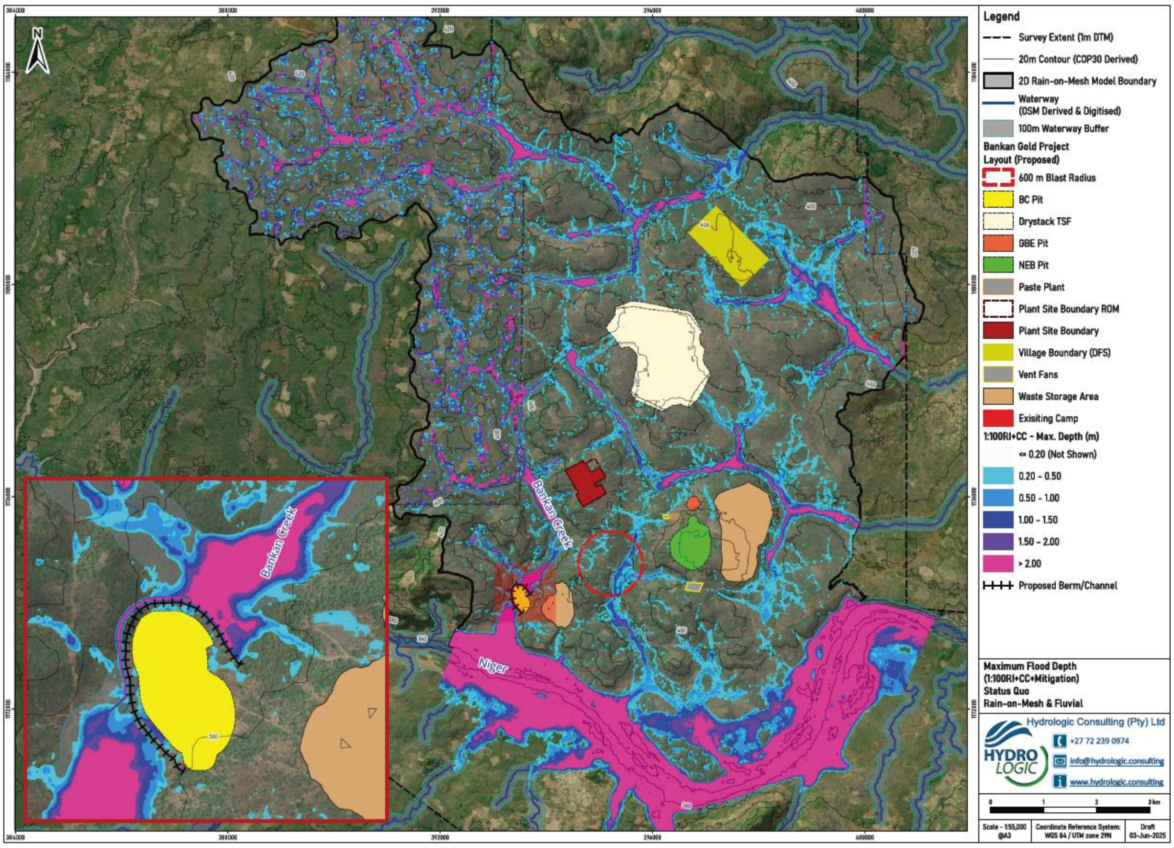

Figure 6.1: Flood Modelling Outcomes 1:100 RI plus Climate Change Event

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.

6.2.4 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.

6.3 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.

7 METALLURGICAL TESTWORK

7.1 Introduction

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.

7.2 Sample Selection

Across the three programs, 82 individual mineralised samples were selected to represent the range of lithologies, grades and deposits as presented in Table 7.1. With the NEB and GBE pits being part of the same geological area, samples from GBE were not distinguished from NEB and NEB should be construed to include samples from GBE.

Table 7.1: Summary of Samples Tested

Deposit	Lithology	Program 1	Program 2	Program 3	Total
NEB	Mafic	2	3	5	**8**
NEB	Tonalite	13	3	8	**24**
NEB	Shear	-	3	9	**12**
NEB	Weathered	3	2	5	**10**
BC	Tonalite	-	17	3	**20**
BC	Shear	-	2	2	**4**
BC	Weathered	2	2	-	**4**
Total		**20**	**32**	**32**	**84**

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.

7.3 Comminution Testwork

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.

Program 1 tested eight samples with the full standard suite of comminution tests that included:

- SMC.
- Bond rod mill work index (BRMWi).
- Bond ball mill work index (BBMWi).
- Bond abrasion index (Ai).
- Specific gravity (SG).

Program 3 extended the comminution data set by testing a further five samples with the full standard suite of comminution tests and 19 samples using the Geopyora test. The Geopyora test was selected since the mass of sample required for the full comminution suite (~35kg) limited the ability to test significantly more samples, however Geopyora test requires significantly less sample mass (<10kg). Due to this this reduction in sample requirements, the Geopyora test was therefore employed to supplement the comminution dataset where sample mass was limited. The program included five samples which were subject to the full comminution suite and the Geopyora test such that calibration between the techniques was possible.

In total, 13 samples were tested with the full standard comminution suite and 19 samples were tested with the Geopyora test. As some samples were paired (both Geopyora and full suite) a total of 27 samples contribute to the comminution dataset. The comminution testwork results are summarised in Table 7.2.

Table 7.2: 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

7.4 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 (Section 7.1) and applied in the variability testing in Program 3. The variability program combined gravity and leaching on 23 samples (Figure 7.1) to define the geometallurgical relationships and design parameters for the processing facility.

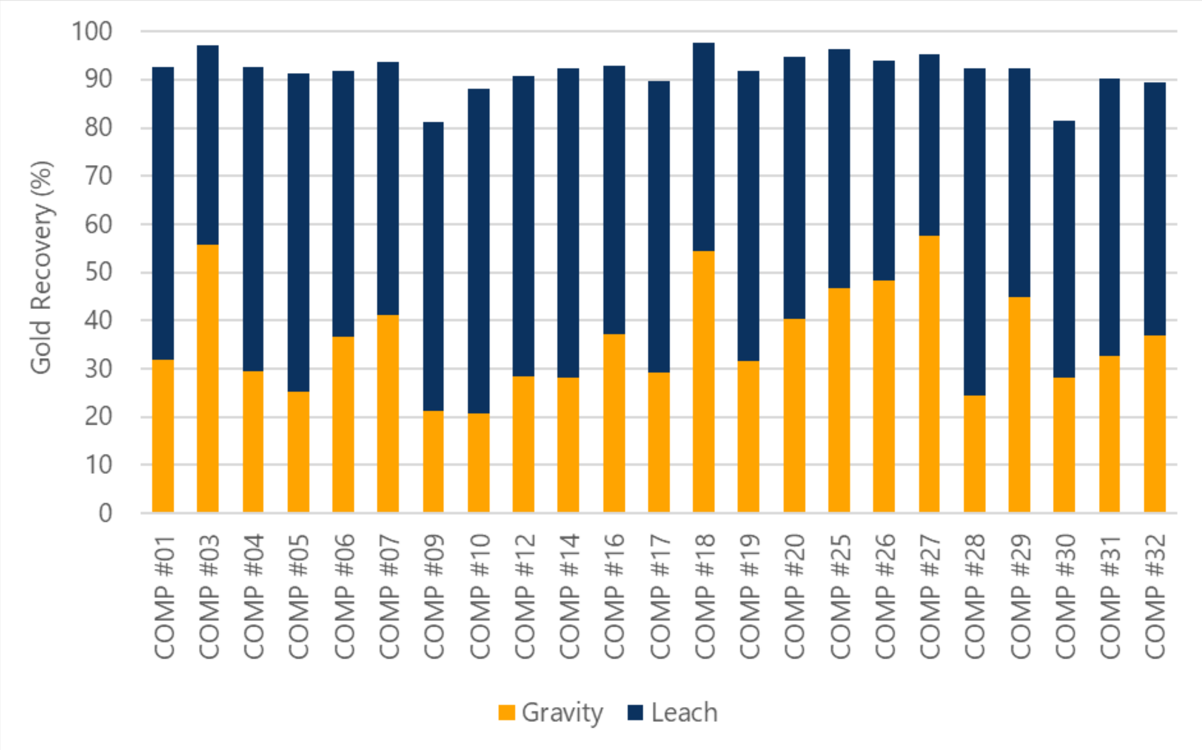

Figure 7.1: Variability Gravity/Leach Results

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

7.5 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. Initial carbon loading testwork carried out in Program 2 indicated that gold loading onto carbon loading can be inhibited if soluble copper is present and the cyanide concentrations in solution decreases below 150ppm.

The bulk leaching (approximately 13kg of sample) was conducted at intentionally high cyanide concentrations (>500ppm) to ensure that copper loading did not interfere with gold loading on the subsequent carbon loading tests, as would be conventional operating practice in an operation that experiences elevated soluble copper levels.

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 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.

7.6 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^{2}h and acceptable performance (greater than 55% solids) was achieved with less than 75% saprolite in the blend at 1.0 t/m^{2}h.

The filtration program tested vacuum and pressure filtration. Vacuum filtration was not successful in achieving an acceptable product on any of the samples tested.

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 of 200 kg DS/m^{2}h compared to 175 kg DS/m^{2}h with the 50% saprolite sample and the 40mm chamber.

For the 25% saprolite sample, product moisture of less than 16% could be achieved with a 50mm chamber and significantly improved filtration rates (266 kg DS/m^{2}h).

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.

7.7 Other Testwork

Since saprolite was identified as a physical constraint because of its "sticky clay like" appearance, materials handling and rheology testwork was specifically completed to assess the risk it 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.

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.

An opportunity remains to source a slag based low heat (LH) which measured acceptable strength for all applications with 6% blend.

7.8 Geometallurgical and Design Parameters

A summary of the key geometallurgical and design parameters determined from the three testwork programs are:

- Materials handling testwork 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 has been implemented for this ore type with the 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.
- 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 32%.
- Cyanide concentration to target leach residual levels of 150 to 200 mg/L of NaCN. 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.
- Air addition to the leach, with no justification for oxygen addition.
- For blends containing less than 75% saprolite, thickener underflows can realistically achieve greater than 52% solids (w/w), and rheology 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.
- Cyanide consumption is influenced by the presence of cyanide soluble copper, 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 with average lime consumptions of:
  - Fresh ore, 0.33 kg/t.
  - Weathered ore, 2.06 kg/t.
- Gold recovery is influenced by gold head grade as defined by the following formula:

$$\text{Au Recovery (\%)} = 0.5145 \times [\text{Au head, g/t}] + 91.533$$

- Cyanide destruction is effective 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.
- The estimation of WAD cyanide is majority driven by free cyanide, and cyanide associated with soluble copper, 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. The median copper in solution is 27 mg/L.

8 MINING AND ORE RESERVES

8.1 Project Mining Strategy

An assessment of a number of project development strategies was carried out as part of a review of the PFS, prior to commencement of the DFS.

The results indicated that early commencement of the underground operation generated considerable benefit to the Project as it provided:

- High-grade plant feed from the start of production.
- Blended feed of weathered and fresh ore which provides sufficient hard rock for efficient operation of the SAG mill in the comminution circuit as well as providing suitable rheology tailings filtration characteristics, which would not be possible from only weathered feed generated from initial open pit mining.

Consequently, 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 was as follows:

- Open pit mining comprises three pits NEB, BC and GBE.
- GBE pit will 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. This has the key benefits of:
  – Mitigating the risks of establishing the underground decline access through the geotechnically challenging weathered 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 will then progress following completion of GBE mining, targeting the sustainable delivery of a minimum of 25% fresh ore feed to the plant from the start of production.
- Pre-stripping of the NEB pit begins in the three months prior to production commencement 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 the BC pit is deferred until the end of the mine life due to the added cost and complexity of establishing surface water management measures around this pit.

8.2 Operations Strategy

The mining operating strategy has been based on contract mining for both open pit and underground mining, with PDI responsible for the overall mining operation and assuming any statutory requirements.

In addition, the approach for engaging the mining contractor for the open pit is dictated by the mining strategy detailed in Section 8.1.

Since the GBE pit mining will occur in the first nine months of the development period and the mining of NEB pit will only commence six months prior to the commencement of production, there will be a nine-month period where only underground mine development is being undertaken with no open pit mining activity. This would result in significant costs as a result of either equipment stand-by charges or de-mobilisation/re-mobilisation of the contractor fleet. Therefore, the operational strategy adopted is that excavation of the GBE pit will be carried out by the bulk earthworks contractor engaged for the Project construction, with a contractor selected who also has some mining experience.

The underground mining contractor will need to be mobilised in parallel with the completion of mining in the GBE pit to ensure a timely transition to underground.

8.3 Mining Methods

8.3.1 Open Pit

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 competent 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 50% and 25% free dig respectively.

The primary production fleet proposed by the selected mining contractor is provided in Table 8.1. The numbers are shown at steady state mining.

Table 8.1: 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

Category	Type	Model	Specification		Number
			Unit	Value	
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**

The contractor will be responsible for de-mobilising all of the mobile mining fleet and decommissioning all mining infrastructure.

8.3.2 Underground

The underground mining philosophy is designed to maximise resource recovery, optimise operational efficiency, and align with processing constraints, while effectively managing geotechnical risks. Selection of the underground mining method was undertaken by evaluating geological, operational, and economic factors to optimise ore recovery and ensure sustainable operations. This included consideration of cost-effective bulk mining techniques, operational flexibility and enhanced ore recovery with fill support.

Following the UBC mining method selection process (Miller-Tait, 1995), sub-level stoping was chosen as the best suited mining method for the Project. 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. In addition, a top-down mining approach has been adopted through extracting panels of three mining levels in a bottom-up mining sequence.

The high-grade nature of the NEB orebody identified the use of paste backfill as a key enabler to maximizing resource recovery.

The underground will be mined with 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 ground support throughout most drives. In areas with elevated geotechnical requirements shotcrete application and cable bolting will provide additional reinforcement. Vertical development will be executed using a combination of raise boring equipment for larger diameter infrastructure and long-hole drilling techniques for smaller/shorter openings. Grade control drilling will be undertaken by RC drills and diamond drills used for resource extension and definition.

Production stopes will be planned to maximise recovery while controlling dilution and will be drilled using long-hole production rigs. Blasting will be carried out with emulsion explosives and electronic detonators to ensure accurate breakage. Broken ore will be mucked by load-haul-dump units and stored in level stockpiles before being transferred to underground trucks and hauled to the surface stockpile. From there, open pit mining trucks will deliver the material to the ROM pad for processing.

The underground mining fleet proposed by the selected mining contractor is provided in Table 8.2. The fleet numbers are indicative of the underground mining requirements during steady state.

Table 8.2: Proposed Underground Mining Fleet

Category	Type	Model (or equivalent)	Number
Mine Development	Loader	Sandvik LH621i (21t)	1
	Twin Boom Jumbo	Axera DD422i	2
	Charge Wagon	Normet 1610B	1
Ground Support	Spraymec	Normet	1
	Agitator Truck	Normet	1
	Cabolter	DS422i	1
Production & Haulage	Loader	Sandvik LH517 (17t)	4
	Truck	Sandvik TH663i	6
	Charge Wagon	Normet 1610B	2
	Production Drill	Sandvik DL432i	2
	ITH Drill	DU411	1
Mine Support	Grader	Cat 12H	1
	Integrated Tool Carrier	Volvo 120	4
	RC Drill Rig		1
	Diamond Drill Rig		1
Total			**29**

8.3.3 ROM and Stockpile Operations

The ROM operating strategy is based on stockpiling ore at a central location adjacent to the primary crusher. The ROM stockpile will comprise a minimum three fingers for fresh ore, saprolite ore and other materials (laterite, saprock and mottled) to facilitate the feed blend requirements of:

- Saprolite clay, being a maximum of 50% of crusher feed and fed via the mineral sizer crushing circuit bypassing the stockpile.
- Fresh rock, being a minimum of 25% of crusher feed and fed via the jaw crusher and stockpiled on the coarse ore stockpile.

Ore will be fed to the primary crusher from these fingers utilising a front-end loader. It is assumed 30% of ore directed to the ROM pad can be direct tipped into the jaw crushing circuit.

Underground ore will be stockpiled at surface adjacent to the GBE pit from where the open pit fleet will reclaim it to the ROM pad. It is assumed this material will be 100% direct tipped due to its consistently high-grade.

Over the life of the Project, some low-grade ore will be stockpiled, to allow for preferential high-grade feed to the plant, and reclaimed when required.

8.4 Open Pit Optimisation

Open pit optimisation applies physical, technical and economic parameters to the Mineral Resource block model to generate the "ideal" open pit excavation geometry. As the Project combines both open pit and underground operations, evaluation of the optimal transition between the open pit and underground designs was required. To accommodate this an average mining cost for extraction via underground methods was applied in Whittle®, and the software limits the shell at the point where the underground method returns improved value over open pit mining.

An initial open pit optimisation was carried utilising costs derived from the PFS inflated to 2025 costs. This optimisation was used as the basis for the pit designs, and a subsequent validation optimisation was carried at the end of the DFS using the most up-to-date parameters available at the time, including:

- Measured and Indicated Mineral Resources only from the most recently published Mineral Resource estimate.
- Geotechnical parameters outlined in Section 5.1.2.
- Mining costs based on the preferred submission from open pit mining contractors as part of the budget pricing process.
- Updated estimates developed from the DFS design for processing, general and administration costs.
- Updated average underground mining cost estimate based on initial information from underground mining contractors.

The pits were optimised in a single multi- resource model optimisation (covering the combined resource models from NEB/GBE and BC) as this provides so that the final shell generated is inclusive of all pits.

The parameters used in the final optimisation are detailed Table 8.3 and the outcomes of the optimisation are presented in Table 8.4. It should be noted that no capital allowance was included in the optimisation.

Table 8.3: Optimisation Parameters

Item	Units	Value	Basis
Mining			
Dilution	%	3.5	Modelled using block-based approach
Ore Loss	%	4.5	Modelled at 3.5% using block based approach plus 1% operational allowance
Open Pit Mining Costs	US$/t mined	4.50	Average based on contractor submissions
Underground Mining Cost	US$/t ore	70.00	Based on initial underground mining contractor rates
Ore Mining Costs			
Grade Control	US$/t ore	0.58	From contractor submissions
Assaying	US$/t ore	0.31	From contractor submissions
ROM & Rehandle	US$/t ore	1.21	From contractor submissions
Processing Ore Costs			
Saprolite/Mottled	US$/t ore	11.51	Process operating cost estimate based on metallurgical testwork and DFS design
Laterite/Saprock	US$/t ore	13.05	
Fresh	US$/t ore	16.49	
Tailings Management	US$/t ore	5.52	Estimated from first principles based on TSF design and operating principles
General & Administration	US$/t ore	4.06	Detailed cost estimate
Total Ore Cost & Recovery			
Saprolite/Mottled	US$/t ore	23.19	Total of Mining Ore Cost and Processing Ore Cost
Laterite/Saprock	US$/t ore	24.73	
Fresh	US$/t ore	28.17	
Processing Recovery	%	0.5145 x Au(g/t) + 91.533	DFS metallurgical variability testwork
Revenue			
Gold Price	US$/oz	1,800	Provided by PDI
Government Royalty	% of Revenue	5.0	
Local Development Contribution	% of Revenue	1.0	
Refining Charge	US$/oz	4.00	
Net Price	US$/oz	1,688	Calculated
	US$/g	54.27	
Discount Rate	%	8	Provided by PDI

Table 8.4: Optimisation Results

Item	Units	Value
Shell	-	36
Revenue Factor	-	1.0
Ore		
Weathered	Mt	10.8
	g/t Au	1.05
Fresh	Mt	34.3
	g/t Au	1.52
Total	Mt	45.1
	g/t Au	1.41
	koz Au	2,043
Waste	Mt	80.4
Total Mined	Mt	125.5
Strip Ratio	-	1.78
Costs & Cashflow		
Mining	US$m	565
Ore	US$m	1,221
Selling	US$m	212
Revenue	US$m	3,407
Cashflow	US$m	1,409
Discounted Cashflow		
Best	US$m	1,103
Worst	US$m	960
Average	US$m	1,032
Mine Life	years	10

Table 8.7 provides the inventory in the optimised pit shell broken down by deposit and a comparison against the pit designs, confirming that the validation optimisation supports the pit designs and Ore Reserve.

An optimisation based on open pit mining only was also carried out. The selected shell detailed above was a close match to the unconstrained shell with a revenue factor of 0.87, indicating the shell is a robust selection.

As dilution is modelled locationally within the block model, the dilution and ore loss varies dependent on the geometry of the shell. The global dilution and ore loss for the selected Shell 36 is 3.3% and 3.1% respectively.

8.5 Open Pit Mine Design

8.5.1 Basis of Pit Design

The pit wall design criteria used in the design of the pits for the Project are outlined in Table 8.5.

Table 8.5: Pit Design Basis

Wall	Weathering	Face Height	Face Angle	Berm Width	Geotech Berm	Stack Berm
		(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

The ramp width used for dual lane and single lane ramps was 29 m and 19 m respectively. Ex-pit roads are designed at 31.5 m width.

For drill and blast, the assumed blast patterns are shown in Table 8.6.

Table 8.6: Assumed Blasting Patterns

Description	Bench Height (m)	Hole Dia. (mm)	Burden (m)	Spacing (m)	Powder Factor (kg/m^3)	% Free Dig	% Blasted
Saprolite Clay	5	152	3.8	4.6	0.64	75%	25%
Mottled Clay						50%	50%
Laterite / Saprock			5.3	8.0	0.26	0%	100%
Fresh Rock			4.6	6.0	0.41	0%	100%
Saprolite Clay	10	178	4.5	5.4	0.86	75%	25%
Mottled Clays						50%	50%
Laterite / Saprock			6.2	9.3	0.36	0%	100%
Fresh Rock			5.3	6.9	0.57	0%	100%

8.5.2 GBE Pit

The GBE pit, approximately 250 m by 250 m as shown in Figure 8.1, was 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.

The GBE pit is serviced by a single lane ramp on the assumption that the bulk earthworks contractor will be using smaller trucks than the main mining fleet.

The mining inventory from the GBE pit is outlined in Table 8.7.

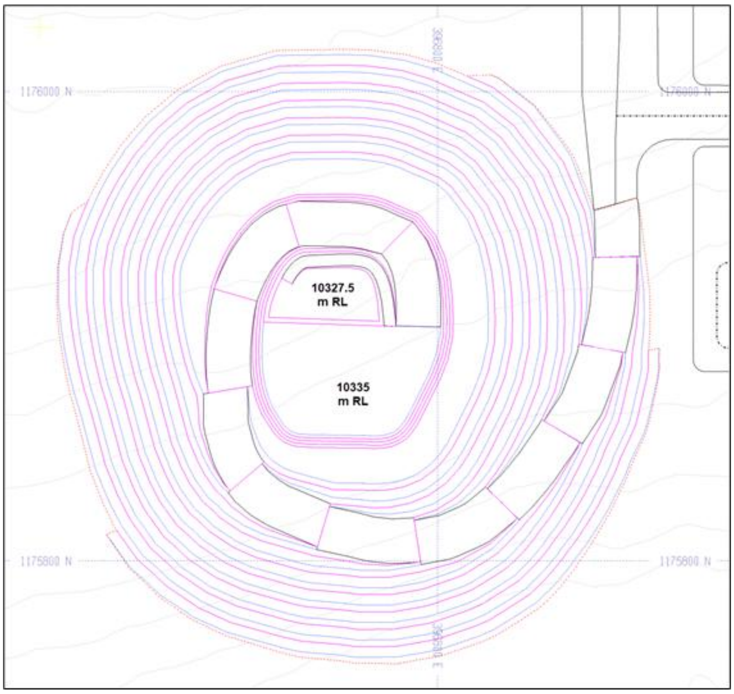

Figure 8.1: GBE Pit Layout

8.5.3 NEB Pit

The larger NEB pit was split into two internal stages and then a final pushback to allow material movement to be balanced over the LOM production schedule. The layouts of the stages of the pit development are shown in Figure 8.2, Figure 8.3 and Figure 8.4.

Stage 1 is an annular stage, approximately 460 m by 630 m that sits in the middle of the final NEB pit targeting high-grade fresh ore to a depth of 75 m.

Stage 2 is a pushback to Stage 1 of approximately 700 m by 930 m, which extends to the final pit limit around the north end of the pit. This stage is targeting low strip ratio low-grade material to the north and along the footwall to an average depth of 145 m below surface as shown in Figure 8.5.

Stage 3 is a final push-back on the western hanging wall of the deposit that targets high-grade fresh ore at depth, as shown in Figure 8.5. 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.

The mining inventory from the NEB pit is outlined in Table 8.7.

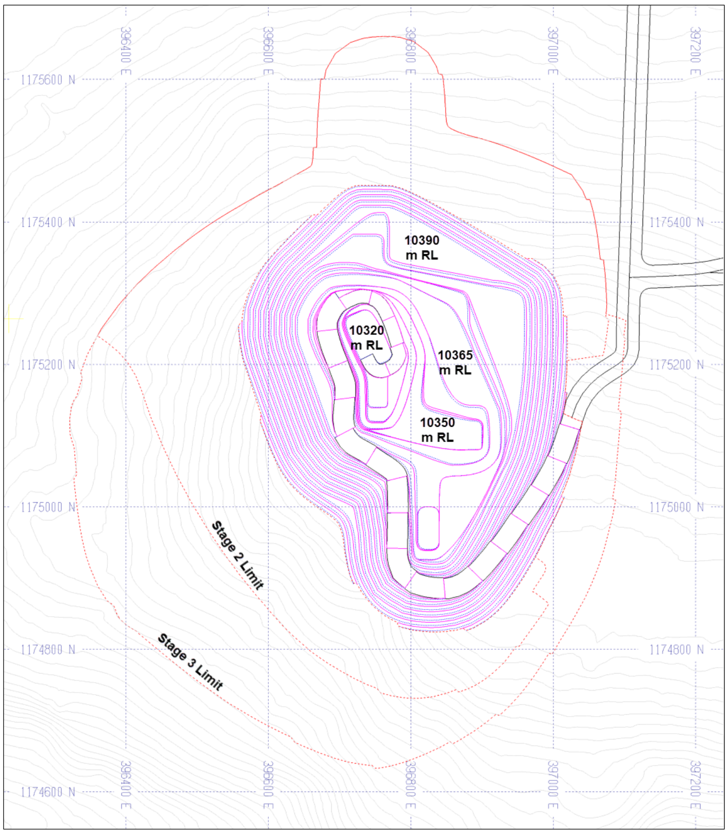

Figure 8.2: NEB Pit Stage 1 Layout

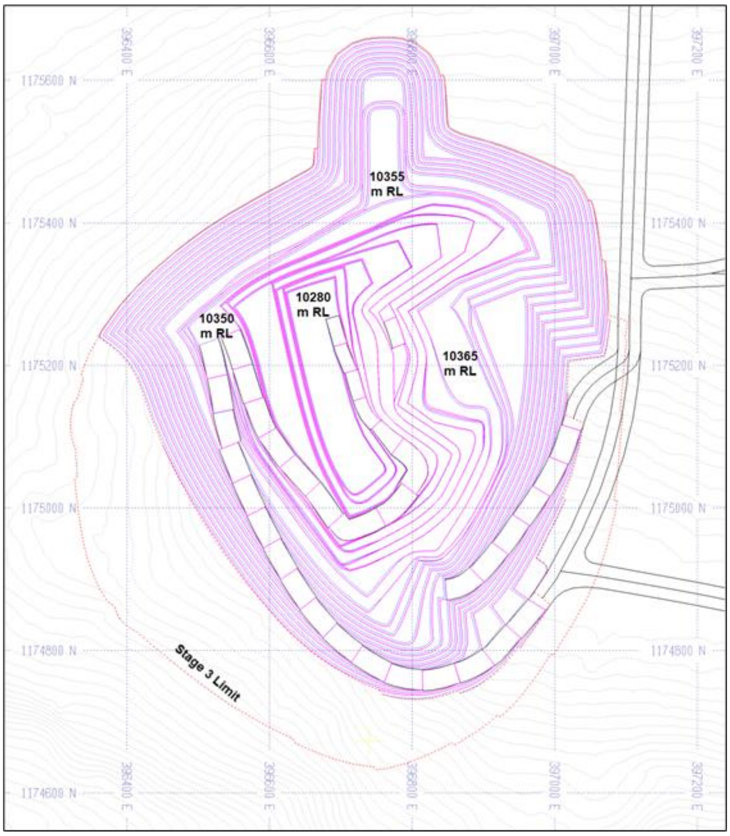

Figure 8.3: NEB Pit Stage 2 Layout

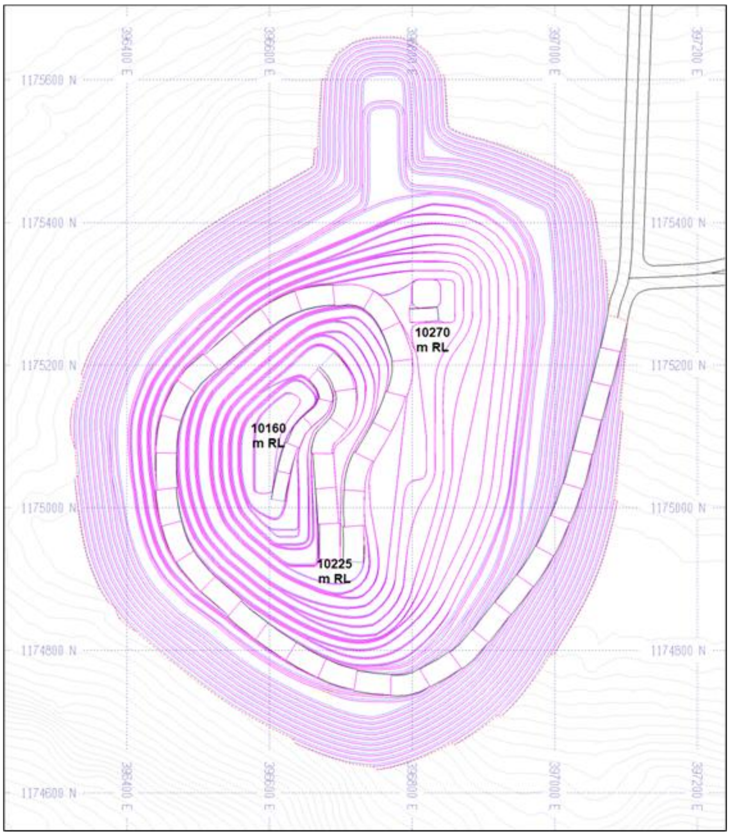

Figure 8.4: Final NEB Pit Layout

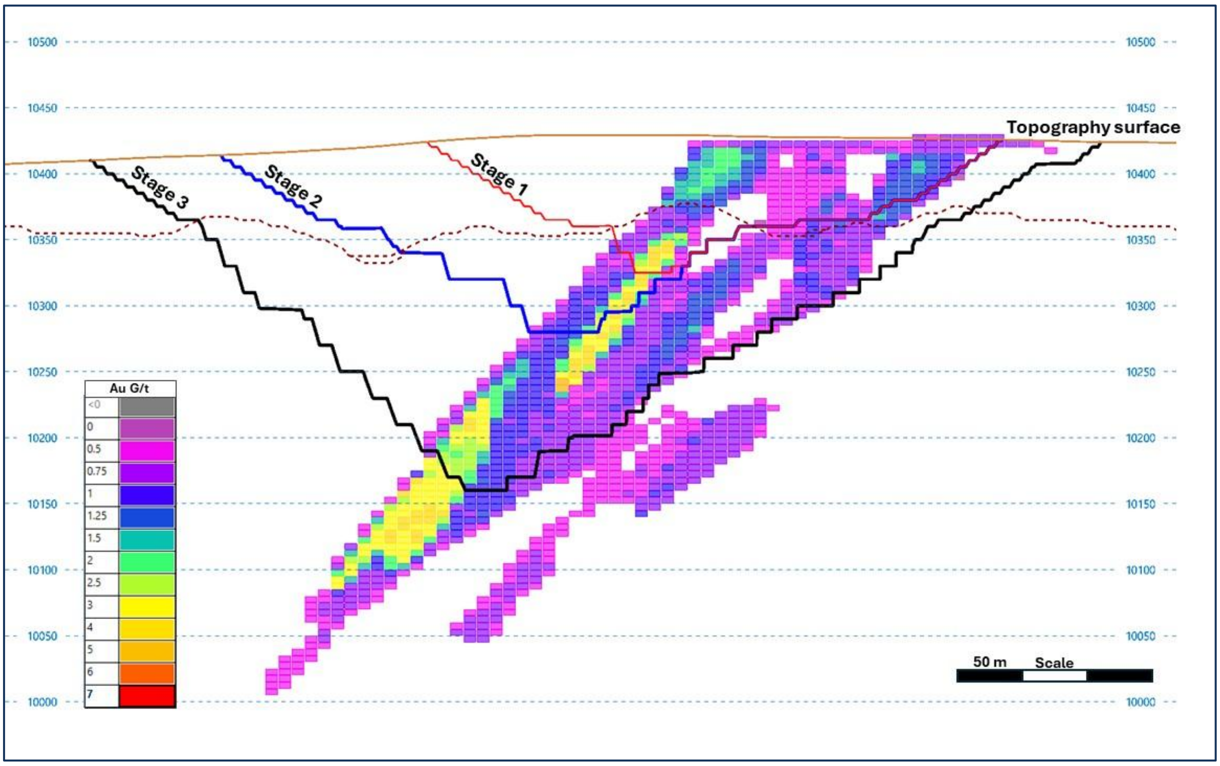

Figure 8.5: NEB Pit Cross Section

8.5.4 BC Pit

The BC pit is mined in a single stage towards the end of the LOM extending down to depth of approximately 70 m. The BC pit targets Indicated mineralisation down to a depth of 70m below surface, below which is only Inferred 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 the Bankan Creek and proximity to the Niger River, incorporates significant surface water management structures including an upstream diversion berm and channel and a downstream flood protection bund.

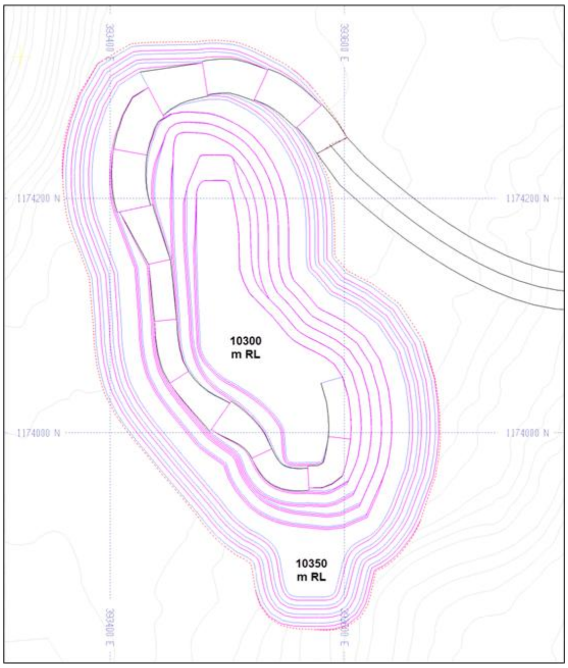

Figure 8.6: BC Pit Layout

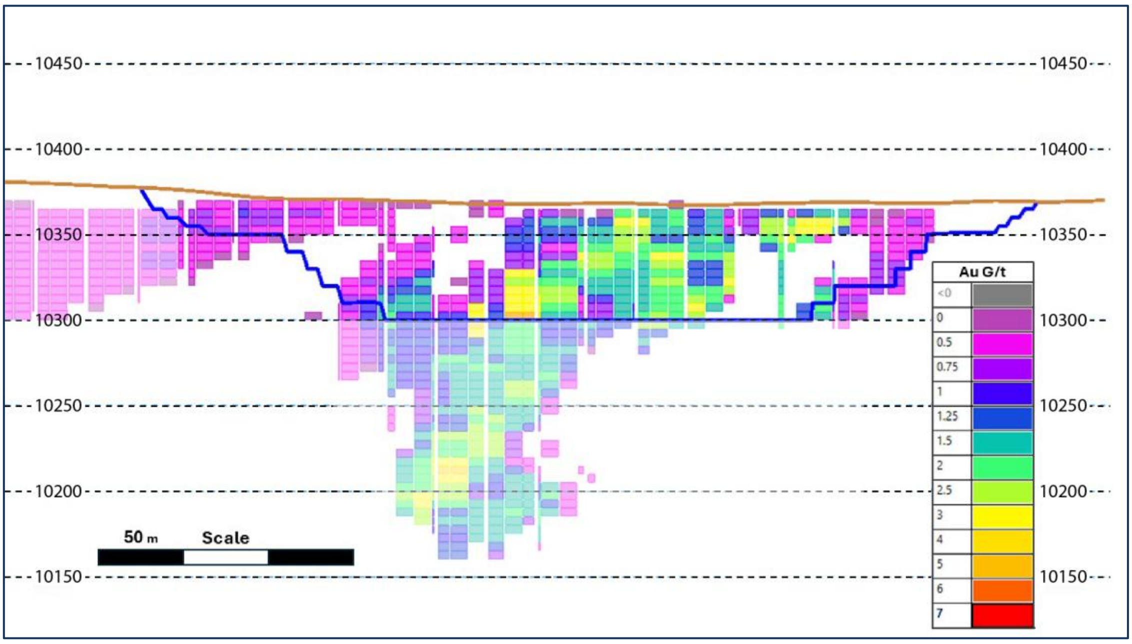

Figure 8.7: BC Pit Cross Section (with Inferred Mineral Resources Shown)

8.5.5 Open Pit Mining Inventory

Table 8.7 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 of the optimisation and design outcomes. In general, the reconciliation is very close, with only GBE generating discrepancy due to its targeting of deeper material to facilitate the underground portal development in fresh rock.

Table 8.7: 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 8.8 details the average dilution and ore loss for each pit and a global average.

Table 8.8: Dilution and Ore Loss Within Pit Designs

Deposit	Ore Loss			Dilution
	Modelled	Operational	Total	Modelled
GBE	11.3%	1.0%	12.3%	8.7%
NEB	3.2%	1.0%	4.2%	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 is a reflection of 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.

8.5.6 Waste Rock Dumps

The waste rock dumps for the open pit mining are designed provide sufficient capacity for all the as-mined pit waste inclusive of allowances for appropriate swell. The main NEB waste dump will be located on the eastern footwall side of the pit. It will be approximately 750 m by 1,750 m and have an average height above surface of 45 m on the west to 65 m on the east as shown in Figure 8.8. The road shown to the south services NEB Stage 2. The dumps will be developed from the south to the north over the life of mine.

A temporary WRD will be located to the north of the GBE pit as shown in Figure 8.9. 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 location is then used for long term stockpiling of low-grade ore.

The BC waste rock dump is located northeast of the pit out of the water course as shown Figure 8.10. It will be approximately 470 m by 570 m and will have an average height above surface of 25 m.

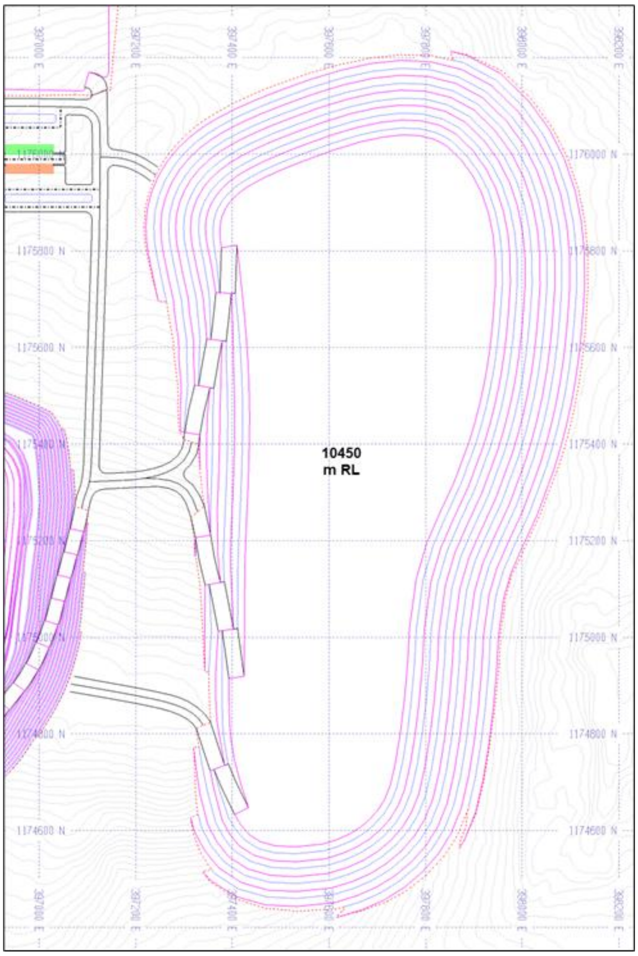

Figure 8.8: NEB Waste Rock Dump Layout

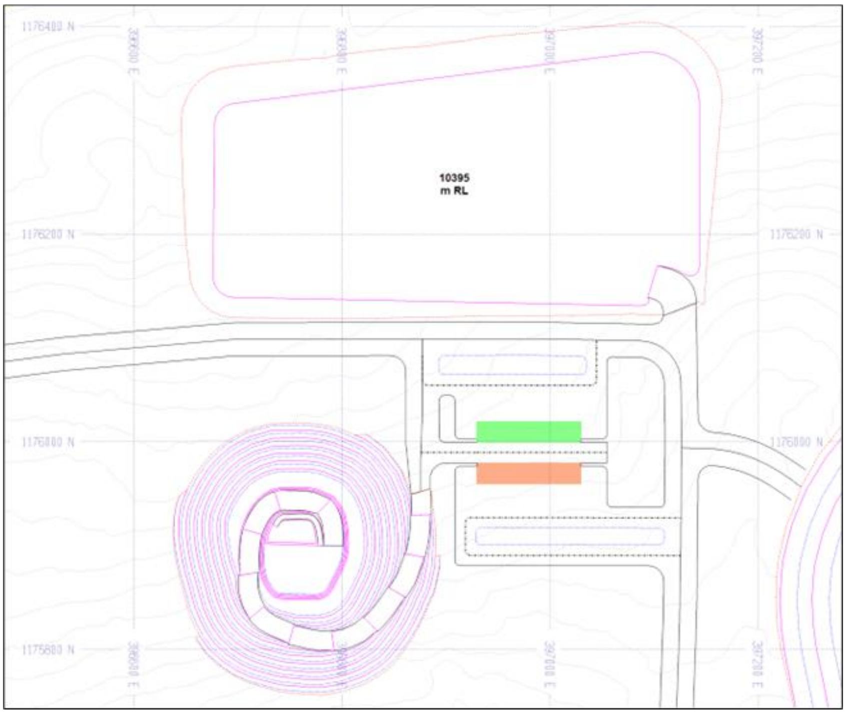

Figure 8.9: GBE Waste Rock Dump Layout

Figure 8.10: BC Waste Rock Dump Layout

Rehabilitation of the waste rock dumps will commence as soon as areas of the dumps are completed to final limits. The WRD faces will be reprofiled to a final landform and then topsoil will be rehandled from stockpiles across the entire dump surface ready for final revegetation and closure.

8.6 Underground Optimisation

Underground stope optimisation was undertaken using Deswik.SO software (SO), a strategic mine planning tool to analyse different stoping and design parameters to maximise the value of an orebody.

The optimisation was undertaken on Indicated Mineral Resources and a portion of the Inferred Mineral Resources. The Inferred Mineral Resources for potential inclusion were selected by the resource Competent Person and wireframes were generated to delineate this portion from other Inferred Mineral Resources. In general, the criteria for selection included:

- Inferred Mineral Resources outside the central high-grade core were generally excluded.
- Where grades were based on extrapolation from assays the Inferred Mineral Resources were generally excluded.
- Where higher levels of uncertainty exist in relation to structure or grade continuity, even though they meet the requirements of an Inferred Mineral Resource, these areas where generally excluded.

Stope optimisation parameters were used to generate transverse stopes for the whole orebody, and as an initial pass to locate the optimal stope shapes. The stopes were then manually split based on geotechnical constraints (stable stope spans and orebody orientation) into mineable shapes and extraction direction.

Key underground optimisation parameters/inputs used for this study are tabled below.

Table 8.9: Stope Optimisation Parameters

Item	Units	Basis
Mining Method	-	Combination of Longitudinal and Transverse Long-hole stoping with Paste Fill
Stope Length		
101600 to 10020 mRL	m	20
10020 to 9880 mRL	m	15
9880 to 9740 mRL	m	15
9740 to 9500 m RL	m	12
Level Height	m	20
Stope width	m	5 m to 100 m
Stope dilution	m	0.5 m Hanging wall and Foot wall
Minimum Pillar	m	30
Dip angles	°	40 / 70 / 2 Minimum / Maximum / Change
Strike angles	°	-0.2
Stope thickness ratio	-	2.4 / 2.4 top to bottom/left to right

The planned dilution, based on an overbreak of 0.5 metres on the hanging wall and footwall of each stope, equates to between 9% and 16% dilution, depending on stope dimensions.

The underground average mining cost and cut-off grade was calculated based on first principles, supported by historical and typical information with the inputs and cut-off grade estimate presented in Table 8.10 and Table 8.11.

Table 8.10: Underground Optimisation Cost Input Parameters

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**

Table 8.11: Underground Cut-off Grade Estimate

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**

A sensitivity analysis was conducted by varying gold price, recovery, mining costs, and processing costs from 80% to 120% of the base case values. This variation gave calculated cut-off grades within a range of 1.64 g/t to 2.46 g/t, with gold price and recovery being by far the largest drivers of cut-off grade variation as shown in Table 8.12 and Figure 8.11.

Table 8.12: 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

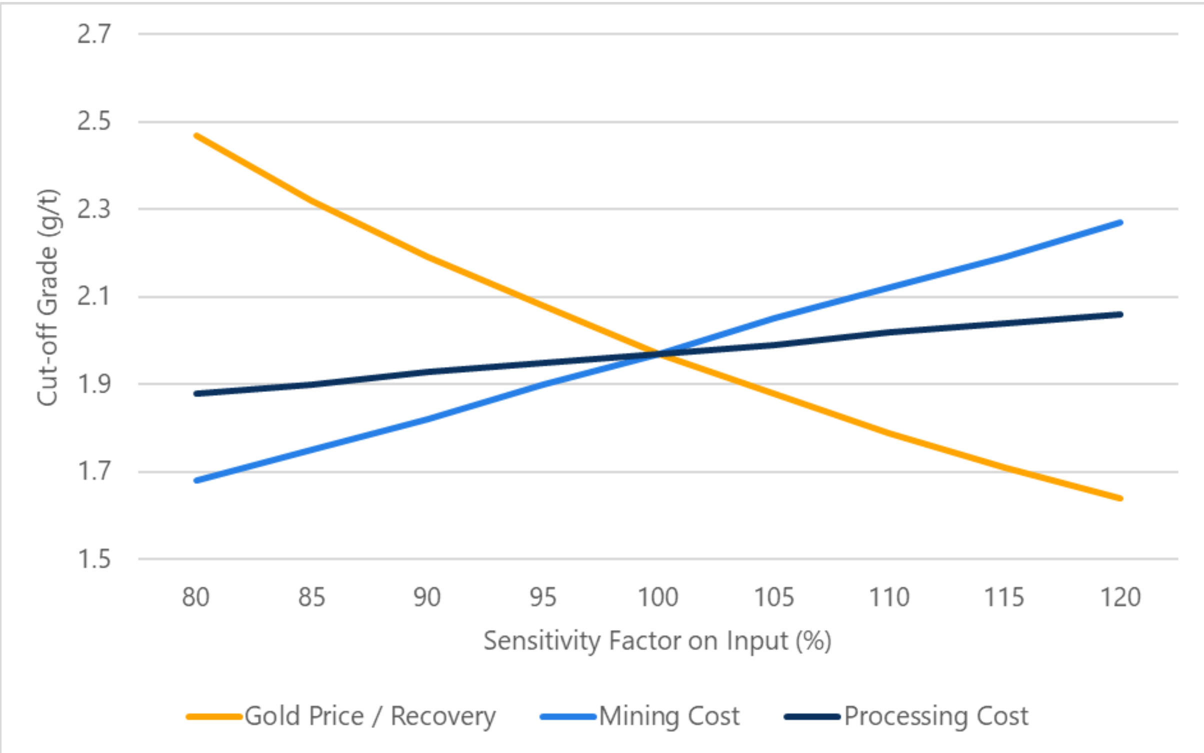

Figure 8.11: Underground Cut-off Grade Sensitivity

8.7 Underground Mine Design

Based on the optimisation, the underground design was developed based on typical mining practices, equipment and designs.

8.7.1 Portal Access and Decline

Access to the underground is through a twin portal arrangement located in the GBE pit at 10335 mRL. This location was constrained by:

- Minimum of 20 m fresh rock on top of the portal.
- Pit wall with 20 m benches and 75° face angle.
- 15 m berm on top of the portal.
- Portal located on the south end of the pit where more fresh rock is present.

The underground mine will be developed with two declines to facilitate the maximisation of production rate from the mine. The two declines will initially be considered as a primary haulage decline and ventilation decline until the main return airway shaft is established, at which point both declines will provide fresh air intake for the life of mine. This secondary decline will also act as an alternative or secondary means of egress for the underground mining operation.

Figure 8.12 shows the elevation of the underground mine with the GBE and NEB pits and Figure 8.13 provides a plan view of the underground mine clearly showing the portal location and decline arrangement.

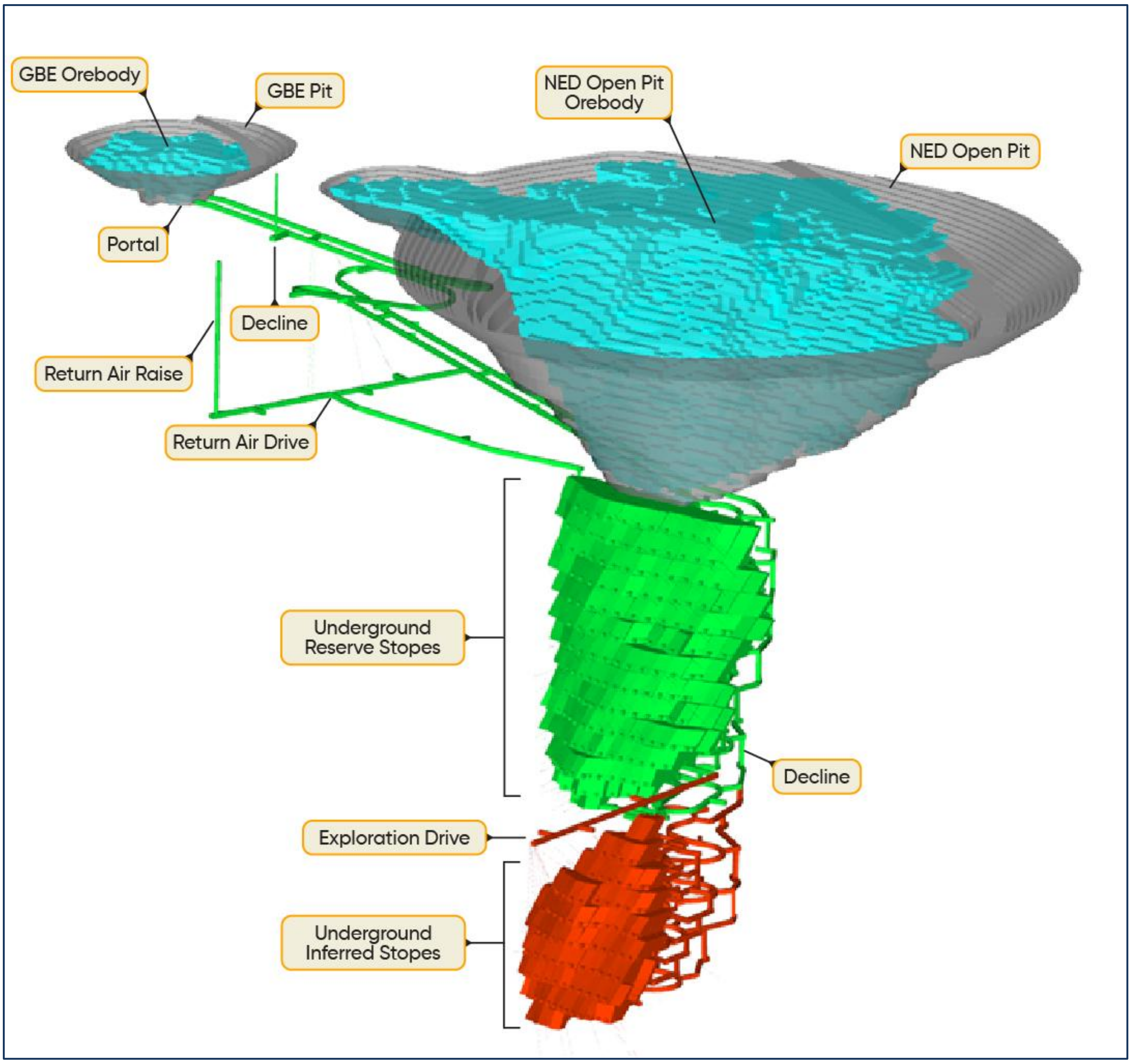

Figure 8.12: Underground Mine Elevation with GBE and NEB Pits looking East

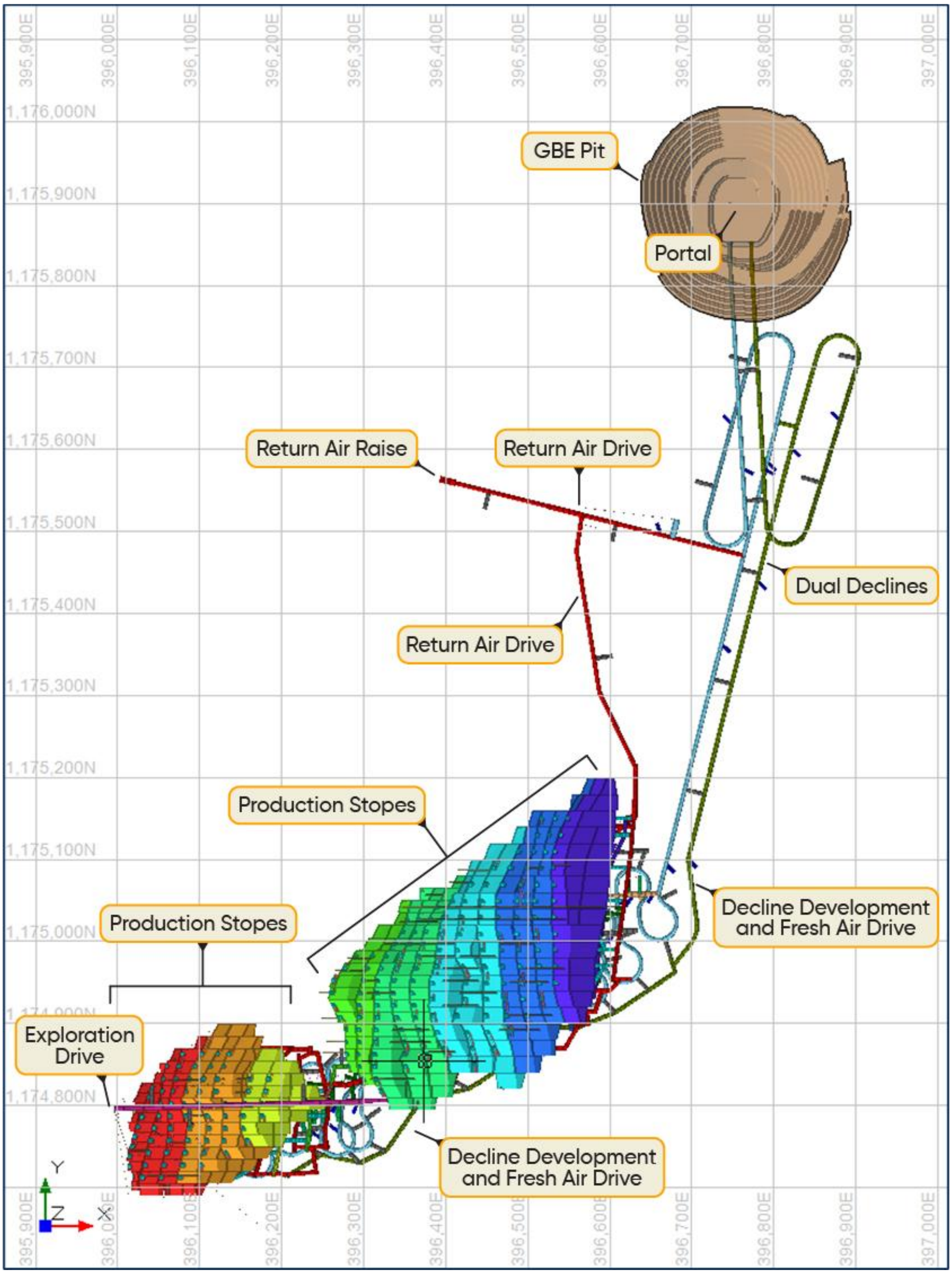

Figure 8.13: Portal Location and Decline Arrangement – Top View

Since the NEB deposit has a good rock qualification (no faults or structures identified) this allows for an uncomplicated development design with the decline being positioned on the footwall to the orebody maintaining 50 m pillar to the NEB pit and 65 m away from the stopes to maintain physical integrity.

The decline access is designed at 1 in 7 down with development stockpiles nominally every 160 m. As the development progresses these empty stockpiles will be utilised as passing bays for various vehicles traversing the decline.

8.7.2 Level Development and Stoping

Level development designs include a footwall drive located 25 m from the orebody providing ore drive access on multiple fronts. A hanging wall drive is designed to support transverse stoping, by offering a free mining face and allowing installation of hanging wall cable bolts to enhance ground stability. Longitudinal stopes are situated along the footwall and at the north and south extremities, with access is provided via dedicated longitudinal ore drives. In addition to production infrastructure, each level is equipped with an escapeway drive, a sump on the access drive, return air connections at both ends of the footwall drive, and a refuge chamber cuddy, which also serves as a location for a substation where required.

Figure 8.14 shows a typical production level design including the features outlined above.

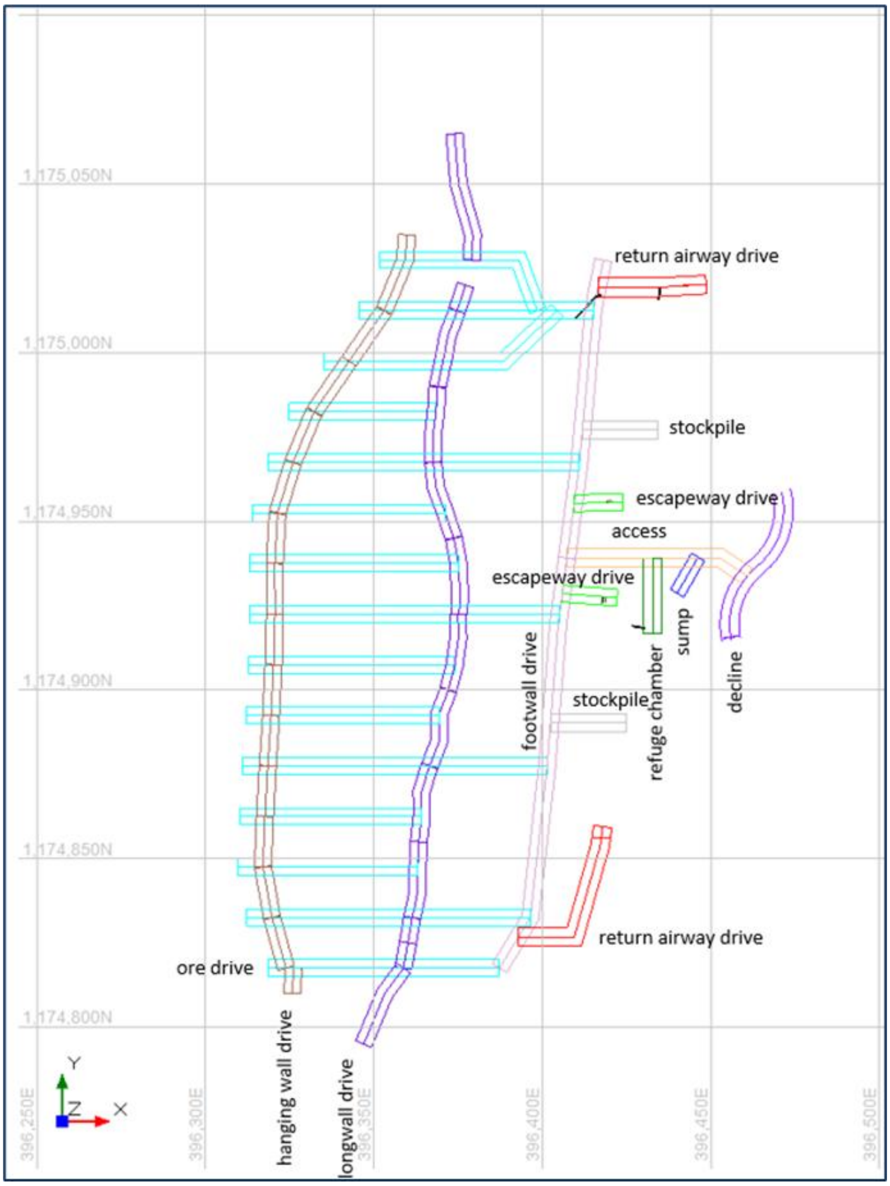

Figure 8.14: Typical Level Plan Design (9960 m RL)

Stope design was based on optimisation results prioritising flexibility, safety, and productivity. The orebody dips between 40° and 45° which influences the level spacing selection of 20 m and design stope length.

The 20 m spacing mitigate risks related to drill hole accuracy and hole deviation, stope recovery and reliable slot performance. It also better respects the orebody's geometry, reducing the potential for internal dilution and maintaining hanging wall stability throughout the operation.

Following stope optimisation, designs were assessed against geotechnical constraints, with focus on hydraulic radius (HR) limits. HR was calculated for both hanging wall (HW_HR) and the stope backs/span (BKS_HR) to ensure that stope geometries remained within acceptable stability thresholds with these results presented in Table 8.13. Stopes were manually reviewed against the geotechnical criteria with those exceeded the acceptable HR limits being modified by splitting in multiple shapes. The final number of stopes, tonnes and grade by mining approach is detailed in Table 8.14.

Table 8.13: 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	10	20	20	3.7	3.3	4.3	4.0	4.3	3.4

Table 8.14: Stope Shapes, Tonnes and Grade by Mining Method

Mining Method	Tonnes Mt	Au Grade g/t	Number of Stopes
Longitudinal long-hole open stoping	4.34	4.33	322
Transverse long-hole open stoping	4.93	4.97	237
Total	**9.27**	**4.67**	**559**

8.7.3 Paste Fill

Paste fill has been selected as the primary backfilling medium for the underground operation, playing a critical role in the mine sequence, production continuity, and overall orebody recovery. Based on stope geometries, tailings characterisation tests, paste fill volumes, binder content, target strengths were established to meet both geotechnical and operational requirements.

Stopes have been classified as either undercut or non-undercut, as shown in Figure 8.15, to differentiate binder strength needs, with undercut stopes requiring higher-strength fill to ensure stability during subsequent mining phases. These undercut stopes will have 10 m of high binder concentrate fill and the rest with a lower concentration.

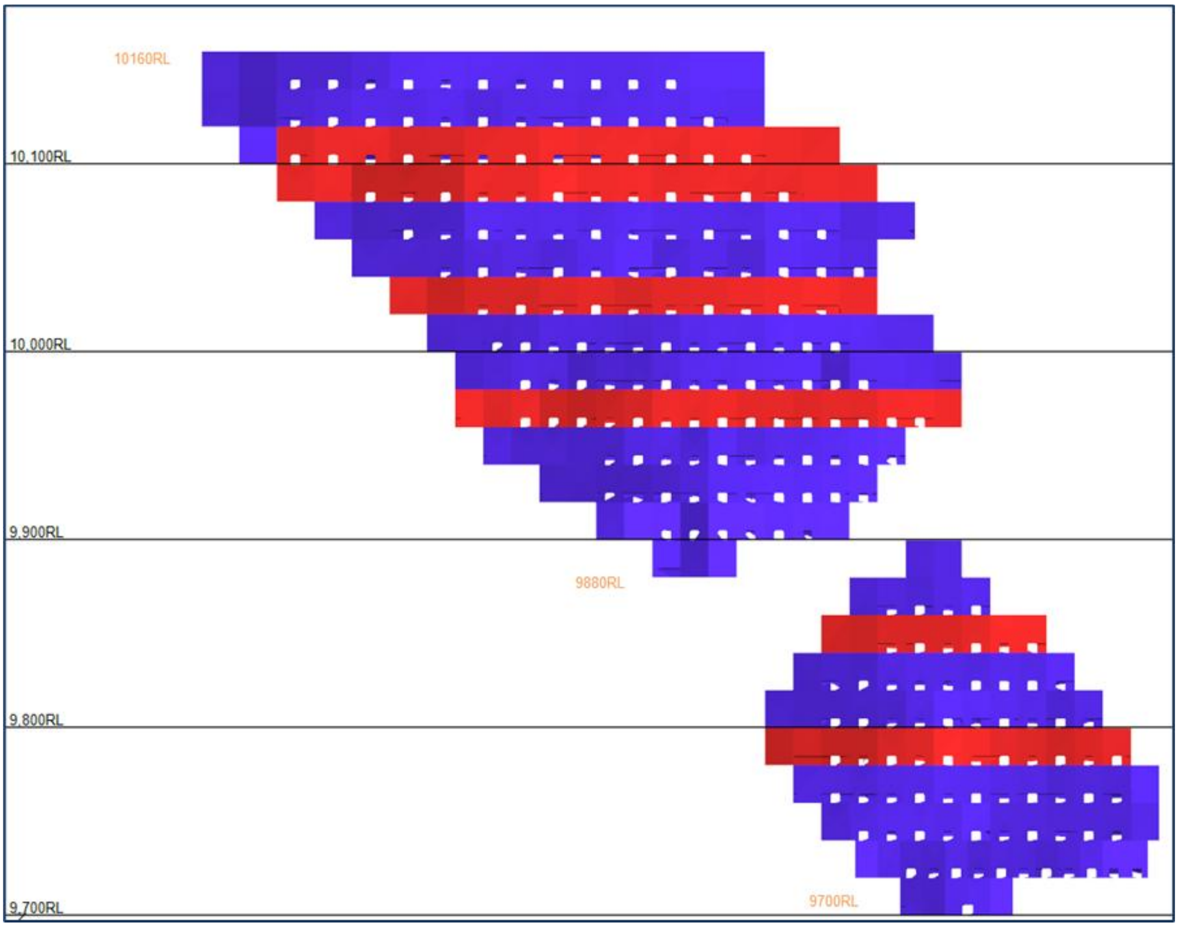

Figure 8.15: Stope Pastefill with Undercut Stopes (Red)

The paste plant is designed with a capacity of 80 m^3/h, and tailings for the paste mix will be transported by truck from the processing plant to the paste plant.

8.7.4 Ground Control

Ground control is fundamental to maintaining a safe and efficient underground mining operation. Development drives will be supported primarily using a combination of rock bolts and mesh with shotcrete only needed for elevated geotechnical requirements as per the following:

- Good quality rock, mesh surface support to within 3.5 m of floor level or less installed with pattern bolting using 2.4 m long friction bolts (or longer) installed at less than 1.3 m bolt spacings in backs and sidewalls to within 2.0 m of floor level or less.
- Bad quality rock, greater than 50 mm fibrecrete shotcrete (with a 28 day strength greater than 32 MPa), surface support to within 1.5 m of floor level or less installed with pattern bolting using greater than 2.4 m long Sandvik D47 MD Bolts installed at less than 1.2 m bolt spacings in backs and sidewalls to within 1.5 m of floor level or less.

Production stopes will also require robust ground control due to the mining method and orebody geometry. Cable bolting is planned for the stope hanging wall (mafic rock contact), along with brow cables to control stability during blasting and extraction. Engineered paste fill will provide additional ground support by filling voids and reducing ground movement.

8.7.5 Ventilation

The planned ventilation system will comprise four surface connections:

- Main portal.
- Ventilation drive.
- Fresh air rise (FAR) dedicated to mine cooling.
- Main return air rise (RAR), which will serve as the primary exhaust pathway for the underground mine.

The underground mine layout will support a negative pressure ventilation circuit, with three primary fans installed in parallel on surface at the main return rise. A requirement for mine cooling will see a dedicates rise established to feed into the primary ventilation intake over the life of mine.

The ventilation circuit (shown in Figure 8.16) was modelled/simulated using Ventsim® software to determine operability and estimate the required primary fan duties.

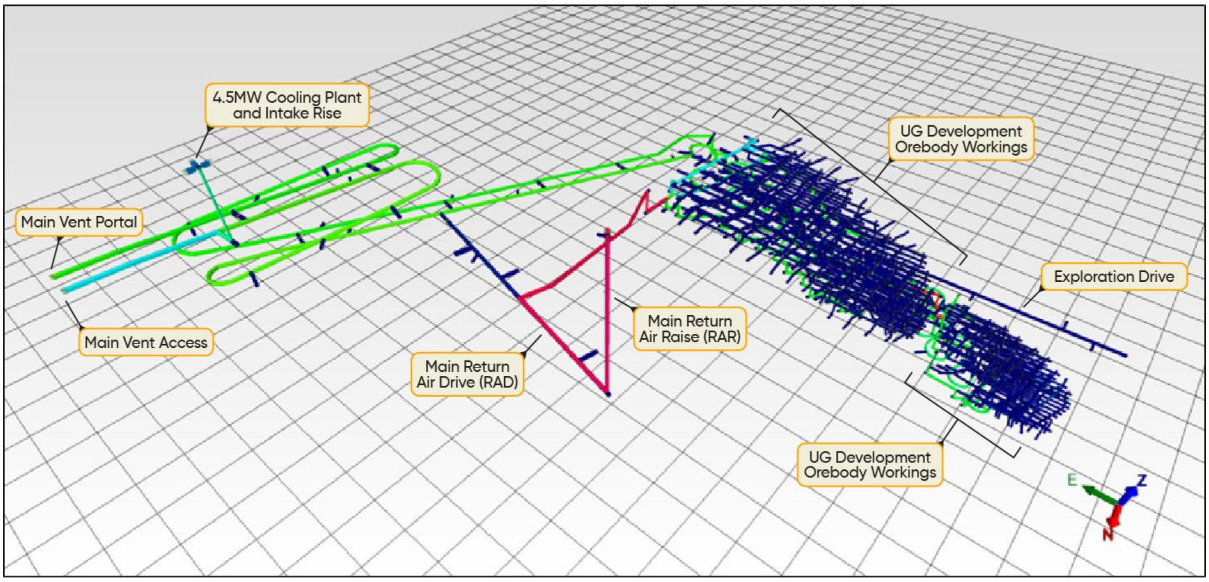

Figure 8.16: Mine Ventilation Circuit (from Ventsim® Software)

A leakage allowance of 20% has been included within the ventilation modelling which is considered realistic for a mining operation with reasonable controls and plans. No consideration has been given to mine gases beyond those associated with blasting fumes.

Airflow demand will vary throughout the life of the project, dependent on variables such as scheduled activities, the number of equipment units in use and equipment utilisation. The peak airflow requirements reach 435 m^3/s in year 6 of the project schedule as shown in Figure 8.17 with a peak primary airflow designed at 460 m^3/s.

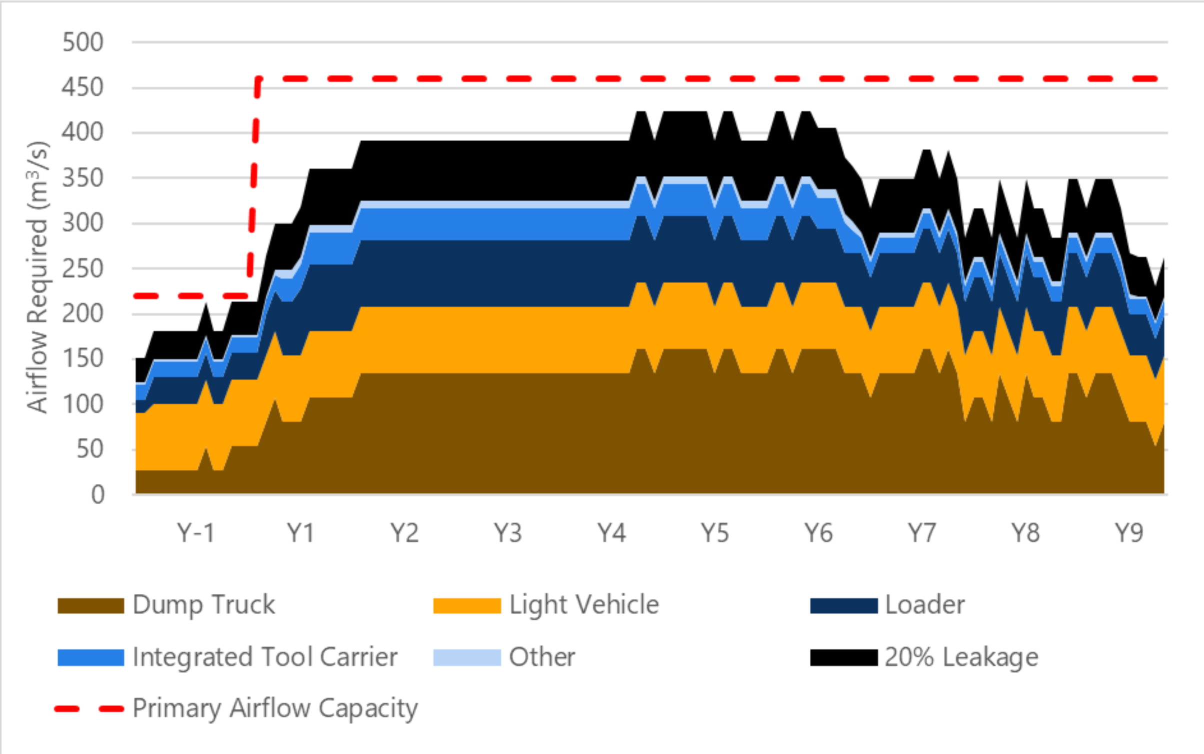

Figure 8.17: Mine Airflow Demand

The primary fan duties for the underground will change throughout the mine's life, increasing as mining progresses. The initial ventilation strategy for the underground operation 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 fresh air decline and will no longer function as a return air decline.

Each production level will have a dedicated secondary ventilation installation to supply fresh air to the working face and ore drives. Auxiliary fan requirements will vary throughout the project lifecycle depending on the stage of the mine and the specific activities being conducted. An indication of the operational auxiliary fan requirements are shown in Figure 8.18.

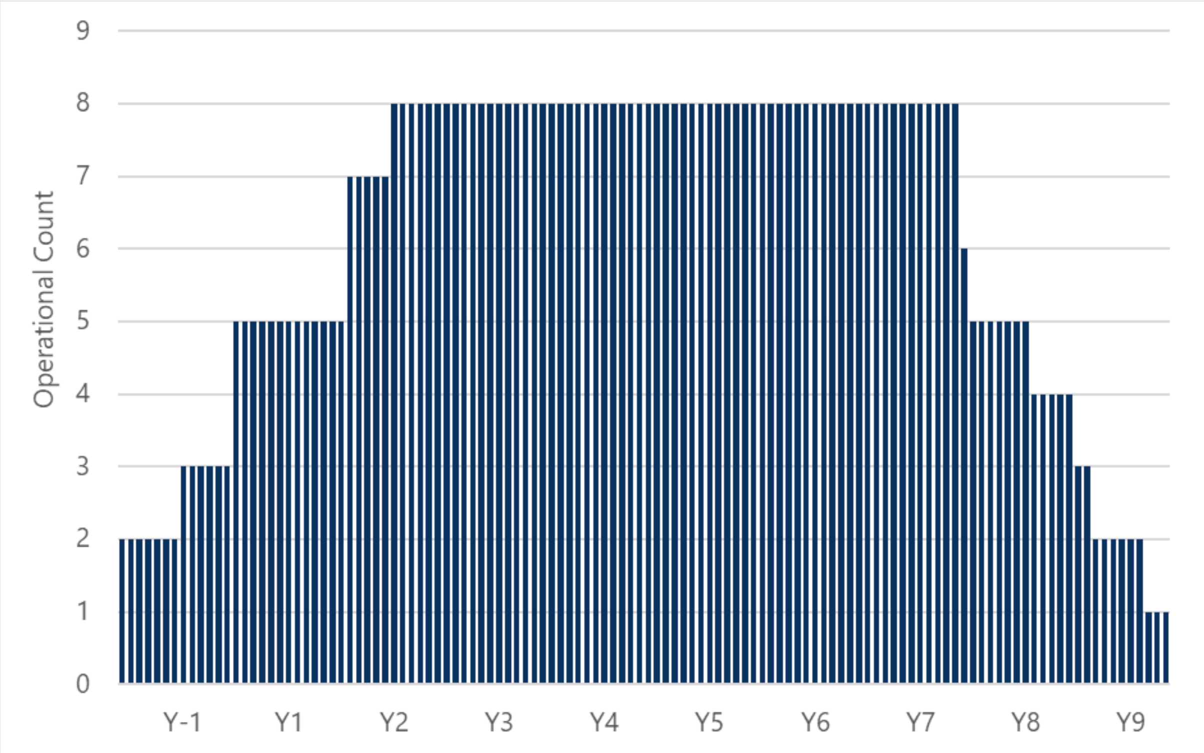

Figure 8.18: Operational Auxiliary Fan Requirements

This system will be located on the decline with a twin 110 kW fan (single stage used) delivering fresh air via 1200/1400 mm round ventilation ducting. Waste air will be returned through in-level regulators to the main RAR. Minimum airflow standards of at least 0.3 m/s are accommodated for in the underground design. This typical ventilation layout is shown in Figure 8.19.

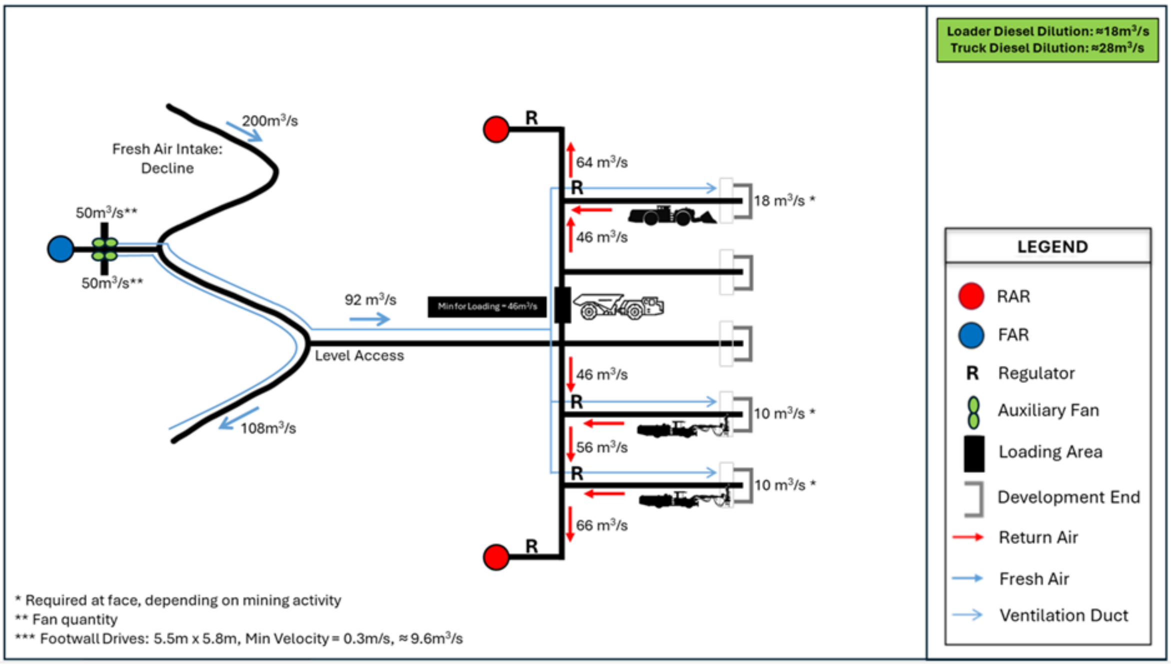

Figure 8.19: Typical Secondary Ventilation Layout

8.7.6 Mine Cooling

The overall heat load for an underground mechanism mine is calculated through the assessment of:

- Mobile equipment, which converts input energy into mechanical power, with any unused output and system losses dissipated as heat.
- Surrounding rock, 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.
- Broken rock from mining activities exposes a large surface area at virgin rock temperature (VRT) and releases more heat with each handling event as additional surfaces are exposed.
- As intake air descends through the mine, it undergoes auto-compression, where potential energy is converted into internal energy, resulting in a rise in air temperature.
- Secondary fans, convert all supplied electrical energy into heat.

Analysis of the mining schedule, equipment fleet and weather data reveal that the underground mine will require a maximum of 4.5 MW bulk air cooling (BAC) over the LOM.

A surface R134a refrigeration plant comprising modular chillers and condenser cooling towers to supply chilled water to a BAC unit is proposed for the Project. Initially designed to meet a 1.5 MW cooling demand, the system is modular and scalable to accommodate increased requirements as the project progresses with the BAC divided into individual cells, capable of delivering 1.5 MW of cooling each.

The cooling requirements for the Project are shown in Figure 8.20.

As shown, there are periods of time in which a shortfall in cooling capacity is expected during certain months of the year due to increased production demand and equipment usage, and the surface climatic conditions, during that period. During this time, heat management strategies will be implemented to manage the heat risk and maintain safe working conditions. It is expected, and confirmed through Ventsim modelling, that the underground wet bulb temperatures should not exceed 29.0°C during these periods.

Sensitivity of cooling demand to small changes in surface conditions is highlighted below through the heat load at reject temperatures of 28.5°C and 29.0°C, respectively.

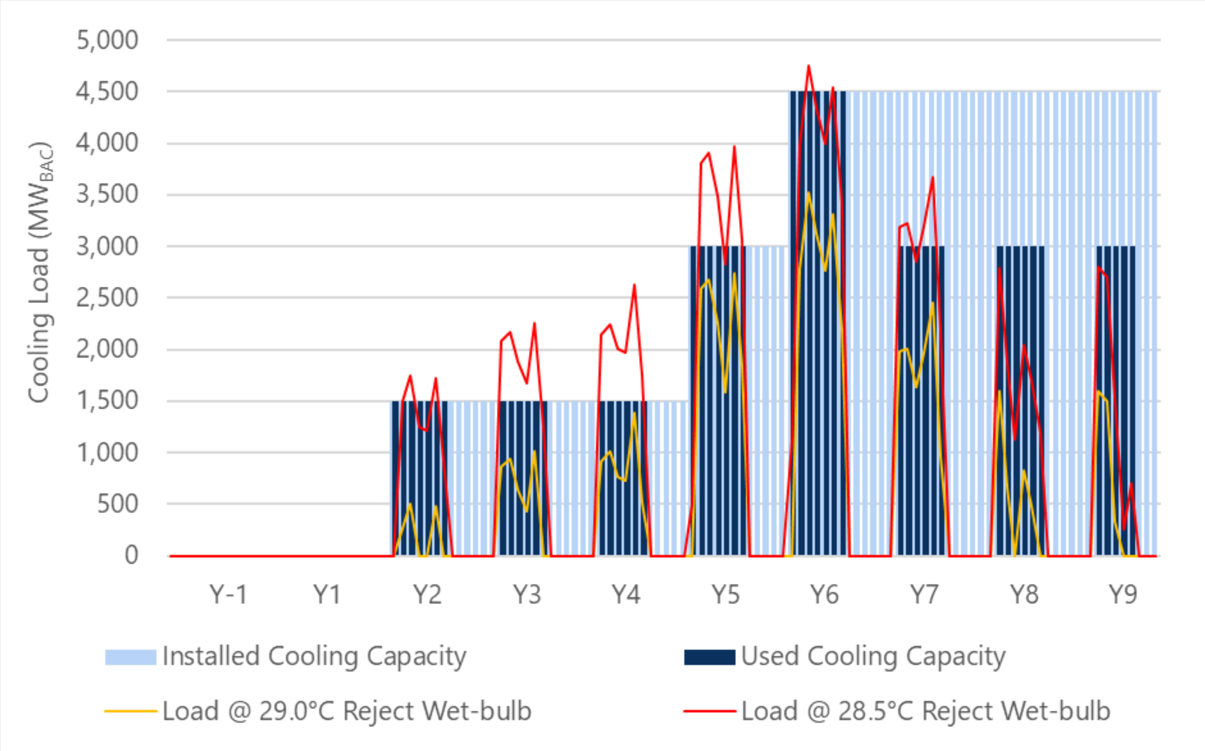

Figure 8.20: Monthly Air-Cooling Requirements and Installed BAC Capacity

8.7.7 Dewatering

The dewatering system was designed to remove a flowrate 40 L/s of water, with redundancy to accommodate higher inflows, due to variability within the estimate of water inflow from the numerical groundwater model and the seasonal weather pattern.

The underground mine dewatering system, shown in Figure 8.21, will consist of three primary underground pump stations:

- Pump station 1 (10,200 mRL level RAD), with three helical rotor pumps (2 duty/1 standby) and a duty flowrate of 40 L/s and maximum flowrate of 99 L/s.
- Pump station 2 (9,908 mRL), with three helical rotor pumps (2 duty/1 standby) and a duty flow of 40 L/s and maximum flowrate of 60 L/s.
- Pump station 3 (9,760 mRL), with three helical rotor pumps (2 duty/1 standby) and a duty flowrate of 40 L/s and a maximum flowrate of 60 L/s.

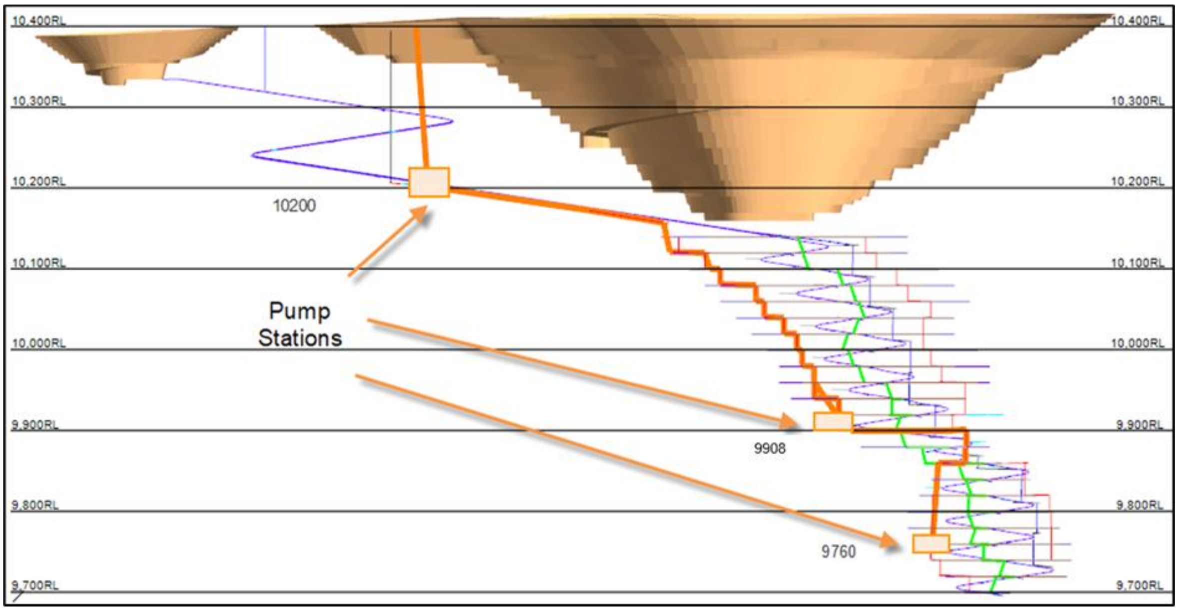

Figure 8.21: Primary Mine Dewatering Design

Primary pump stations will be mined to a size of 8.0 mW x 6.0 mH at a gradient of 1:40 up and have a dedicated sump installed adjacent to the pump station, to settle fines before delivery to the pumps.

Secondary pumping systems, consisting of sumps throughout the mine, are located along the decline spaced at approximately 40 vertical metres, and on each mining level, situated in the access.

Once the water is pumped from the primary pumps via the main rising main, it will exit near the paste plant and discharge into the dirty water pond for settling of silt, recovery of any hydrocarbons and re-use in mine dust control.

8.7.8 Power

Initially power for the mine, including surface vent fans and other infrastructure, will be supplied by diesel generators with connection to the site power grid scheduled once the power station at the process plant becomes operational stable. The diesel generators will remain in place to provide emergency power in the event of a mains power outage. While the diesel generators will not provide sufficient power to continue full mining operations, facilities will be installed in the substation to allow connection of additional generators to supply full mine load if required.

The underground mine 1,000 V feed will be supplied by a 11/1kV step-down transformer installed at surface and delivered underground via service holes connecting to the 10,200 level.

Four 11/1 kV 2 MVA substations are required underground and will supply eight 6-way distribution boards providing the required power for up to eight levels, including primary pumping, ventilation and refuge chambers. All refuge chamber drives have planned service holes to connect them, which will be used to distribute power between levels.

Underground power requirements, infrastructure and reticulation needs were calculated in range of 40 to 45 GWh following a ramp-up over the first three years of underground mining operations as shown in Figure 8.22.

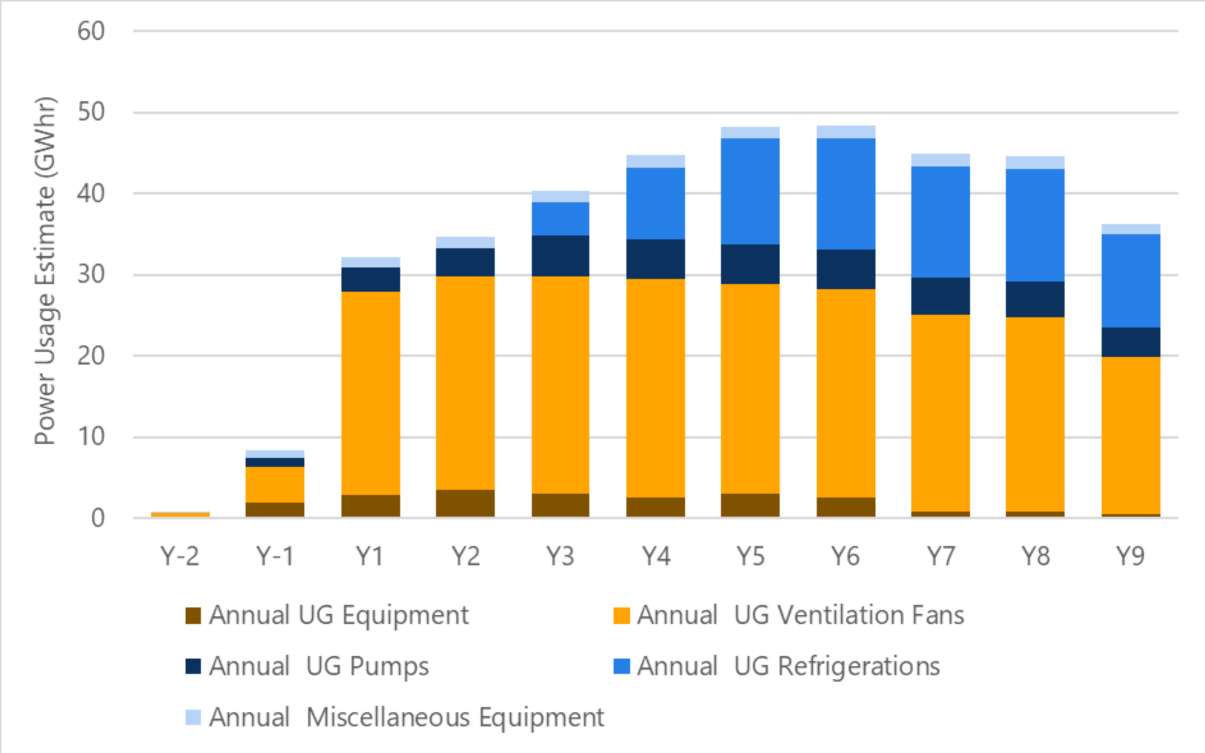

Figure 8.22: Underground Mine Annual Power Usage Estimate

8.7.9 General

Other underground mine design aspects include:

- Communications, via a leaky feeder, typical in many underground mines, will be established throughout mine to enable radio communication for the entire mine.
- Emergency, or secondary means of egress system, designed to help personnel exit the mine if main access routes are blocked due to ground support failure or equipment issues is provided, through dedicated travel ways and raises connecting to other levels and primary exhaust airways serving each mining zone.
- Self-contained refuge chambers, offering a safe haven if the underground atmosphere becomes hazardous, has been included based on best practise guidelines that recommend placing refuge chambers no more than 750 meters walking distance from either the working face or another refuge chamber.

8.8 Production Scheduling

8.8.1 Scheduling Strategy

As detailed previously, the mining sequence and timeline was established prior to the DFS scheduling commencing by overall Project strategy. The process for developing the combined open pit and underground mining and production schedule for the DFS was as follows:

- Underground schedule was developed first as the underground ore has the highest grade and Project returns are optimised if this ore is prioritised. This underground schedule showed that a consistent and sustainable fresh underground ore 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 planned to be 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. It is mined in three stages to prioritise access to higher grade ore and allow material movement to be balanced over the life of mine.
- BC, while being a very profitable pit, containing 10% of the total open pit ounces for 5% of the total material, is located in the Bankan Creek water course and relatively close to the Niger River. These surface water management requirements add additional cost and complexity for this pit leading to it being scheduled at the end of the mine life.

8.8.2 Scheduling Objectives and Constraints

The key targets and constraints of the overall LOM schedule included:

- Ramp up the feed rate to the processing facility as outlined in Section 12.3 and then sustain the plant feed at 4.5 Mtpa over the LOM.
- Maintain a minimum of 25% fresh 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.

Key objectives of the open pit LOM schedule included:

- Minimise material movement in the early years of production.
- Maximise head grade wherever possible by stockpiling lower grade ore.
- Limit vertical rate of advance to 12, 5 m benches per year.

Key objectives of the underground LOM schedule included:

- Accessing high-grade ore as early as possible while also supplementing the mill feed with fresh rock which is required for the efficient comminution circuit operation.
- Accelerated underground development through establishing two portals. This dual-portal approach enabled simultaneous development of the haulage drive and the return airway, accelerating access to the orebody without relying on the completion of the long-term return air raise from surface.

- Stope sequencing aimed at delivering high-grade feed as early as possible. Whilst mining will generally occur top-down, stopes were grouped into three-level blocks to be mined in a bottom-up sequence which is ideal for long-term stability and efficient paste backfilling.
- Targeting the delivery of the maximum possible production from the underground mine to supplement the open pit production to meet the LOM production plan, with an aim to deliver an average of 1.4 Mtpa of underground ore following the underground mining ramp-up period.
- Maintaining high equipment utilisation where possible.

8.8.3 Underground Schedule

The underground production schedule, which forms the basis on which the open pit production schedule has been developed with the aim of maintaining full mill feed and maximising gold production within the blend requirements, has been developed from the mine design. This ore production schedule, providing development ore tonnes and grade as well as stoping ore tonnes and grade, is provided in Table 8.17.

The underground development schedule forms the basis of the underground production schedule, supporting timely access to ore zones, infrastructure installation, and backfill placement. Development activities have been sequenced to align with the production schedule, ventilation requirements, and geotechnical constraints. The development activities by Project year and activity, as well as the split between capital and operating costs, are outlined in Table 8.18.

In addition, the implementation of paste fill is a critical component of the underground mining, enabling maximum stope recovery while maintaining geotechnical stability. Table 8.19 presents the projected annual paste fill requirements over the life of mine, based on stope geometry, extraction sequence, and backfill cycle timing.

Stope sequencing also played a critical role in delivering early gold production from the underground mine. Stopes were grouped into three-level blocks to be mined in a bottom-up sequence, as shown in Figure 8.23, which is ideal for long-term stability and efficient paste backfilling.

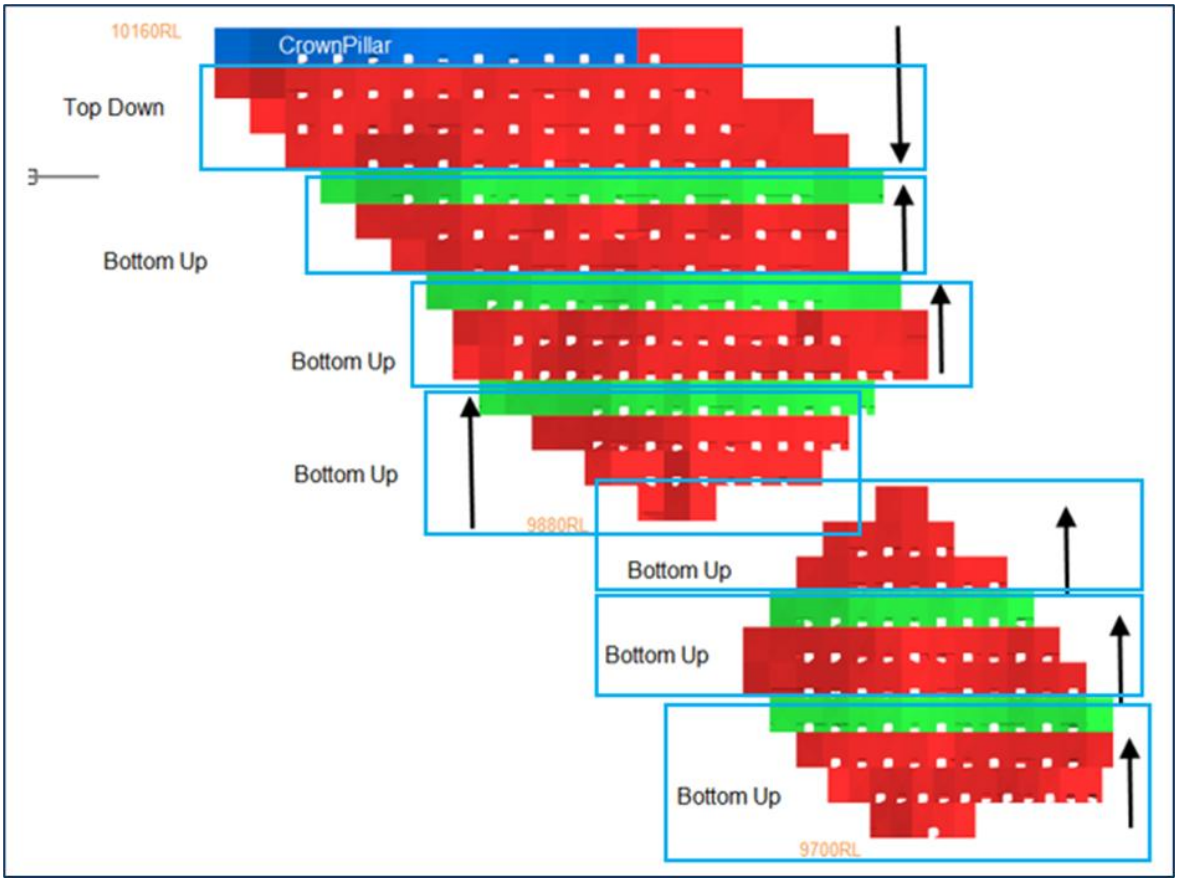

Figure 8.23: Underground Panel Stoping Sequence (Sill Pillars in Green, Stopes in Red)

The development schedule was based on the production rates provided by the potential mining contractor. The average development advance rates used are shown in Table 8.15 for various activities and in Table 8.16 for various equipment types.

Table 8.15: Underground Scheduling Parameters – Advance Rates by Activity

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
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 Drillling	m/d	280
RC Drill	m/d	200
Raise Borer	m/d	4

Table 8.16: Underground Scheduling Parameters – Advance Rates by Equipment

Activity	Units	Rate
Jumbo	m/month	250
Production drill	m/day	280
Bogger	t/day	1,200
RC drill	m/day	200

8.8.4 Open Pit Schedule

The open pit schedule has been developed to completement the production from the underground mine and maintain full ore feed to the mill while maximising value and overall gold production. The open pit production schedule is presented in Table 8.20. The general mining schedule is as follows:

- GBE pit is mined and completed in the first 9 months following commencement of the Project delivery, approximately 15 months before the commencement of ore processing.
- Pre-strip of NEB Stage 1 commences three months prior to commencement of ore processing, with 1.2 Mt of material mined and 195 kt or ore placed on stockpile prior to the commencement of processing, which is driven by the timing for the development of the underground to deliver stable fresh ore feed.

- NEB Stage 1 is completed in final quarter of the third year of ore processing.
- NEB Stage 2 commences in the third quarter of the second year of ore processing and finishes in year seven.
- NEB Stage 3 commences late in the third year of processing and finishes in year 12.
- BC commences in year 9 of processing and finishes in year 11.

Bench turnover is generally less than one bench per month on average for all pits except GBE. A bench turnover rate of between one and two, five metre benches per month is required to complete GBE in the timeframe, which is still considered achievable given the material mined is predominantly weathered. It is also proposed, and included in the DFS cost estimate, to complete grade control of the entire GBE pit prior to commencement of mining, to facilitate this advance rate.

The total ex-pit material movement is approximately 7.5 Mtpa to 8 Mtpa for the first two years. This movement rate then ramps up to between 13 and 14 Mtpa from year three to year six. Following this, the movement rate falls to approximately 10.5 Mtpa from year seven to year nine before gradually reducing over the final three years of mining.

Low-grade stockpiles will generally be between 1.5 Mt and 2 Mt for the first five years, peaking at approximately 2.5 Mt. During this period, the stockpiled material is predominantly low-grade saprolite, with some high-grade saprolite also being stockpiled when required. This is necessitated by the lithology blending requirements for process plant feed. There is also some low-grade fresh ore stockpiled, strictly for production maximisation purposes.

From year six to year 11 the stockpile steadily increases to a maximum of approximately 3.5 Mt, consisting almost entirely low-grade fresh material, which is subsequently reclaimed to the process plant in the last years of operation.

Table 8.17: Underground Ore Tonnes, Grade and Contained Gold

Source	Units	Total	Pre-Prod.	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Stoping	kt	**8,865.2**		816.7	1,099.4	1,062.2	1,102.0	1,104.7	1,119.8	1,198.0	784.0	578.4			
	g/t	**3.95**		4.34	4.10	3.97	3.78	4.31	3.84	3.93	3.58	3.46			
	koz	**1,125.5**		113.9	144.9	135.5	133.9	153.1	138.3	151.2	90.4	64.3			
Development	kt	**1,924.1**	74.1	344.6	304.6	341.8	300.1	299.3	259.7						
	g/t	**2.97**	1.00	3.40	2.94	3.05	3.16	2.43	3.27						
	koz	**183.5**	2.4	37.6	28.8	33.5	30.5	23.4	27.3						
Total	**kt**	**10,789.3**	74.1	1,161.2	1,404.0	1,404.0	1,402.0	1,404.0	1,379.6	1,198.0	784.0	578.4			
	g/t	**3.77**	1.00	4.06	3.85	3.75	3.65	3.91	3.73	3.93	3.58	3.46			
	koz	**1,309.0**	2.4	151.6	173.7	169.1	164.3	176.5	165.6	151.2	90.4	64.3			

Table 8.18: Underground Development Drive Schedule

Development Drive Type	Units	Total	Pre-Prod.	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Access & Safety															
Decline	m	**4,625**	2,026	678	519	232	485	513	171						
Access Drives	m	**946**	164	175	169	38	120	133	147						
Escape Way Drive	m	**872**	43	65	215	122	169	196	63						
Refuge Chamber	m	**454**	70	74	92	39	59	72	46						
Ventilation															
Fresh Air Drive	m	**2,347**	1,600	126	116	210	147	106	41						
Return air Drive	m	**2,679**	808	320	603	244	132	276	297						

Development Drive Type	Units	Total	Pre-Prod.	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Level Drives															
Footwall Drive	m	**3,560**	533	584	1,124	334	220	504	261						
Hanging wall Drive	m	**2,620**	0	532	303	444	699	284	357						
Longitudinal ore drive	m	**4,502**	15	788	796	803	687	584	829						
Ore drive	m	**14,414**	251	2,785	2,432	2,989	1,882	2,120	1,955						
Slot Drive	m	**280**				76	111	42	51						
Dewatering															
Pump Station Drive	m	**68**	23		23			23							
Sump	m	**570**	224	79	68	24	59	69	48						
Other															
Stockpile	m	**1,858**	524	232	348	155	242	256	101						
Diamond Drill Drive	m	**370**	0	0	0	23	334	13	0						
Total	m	**40,165**	**6,282**	**6,436**	**6,808**	**5,734**	**5,345**	**5,192**	**4,368**						
Capital	m	**16,176**	5,937	2,214	3,022	1,121	1,648	1,448	786						
Operating	m	**23,989**	345	4,223	3,786	4,613	3,698	3,744	3,581						

Table 8.19: Paste Fill Requirements Schedule

Paste Fill	Units	Total	Pre-Prod.	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Paste Volume	'000 m^3	**3,197**		279	394	386	400	397	415	423	296	207			
Paste Tonnes	'000 t	**4,225**		518	733	717	745	738	773	788	550	386			
Binder	kg	**357,836**		33,931	58,218	58,737	44,680	31,942	39,557	47,118	26,952	16,702			
Tailings Tonnes	'000 t	**4,373**		537	759	743	771	764	800	815	569	400			
Water	'000 m^3	**15,112**		1,854	2,622	2,566	2,664	2,641	2,765	2,818	1,967	1,381			
Walls	No.	**997**		74	90	99	112	131	141	151	113	86			

Table 8.20: Open Pit Mining Schedule

Metric	Unit	Total	Pre-Prod.	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												
Ore Grade	g/t	**0.73**	0.73												
Contained Gold	koz	**13.1**	13.1												

Metric	Unit	Total	Pre-Prod.	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
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					
Waste Mined	Mt	**44.5**				1.3	7.0	6.2	8.4	6.1	6.3	4.5	2.8	1.7	0.2
Ore Mined	Mt	**21.1**				0.0	0.0	1.0	2.1	1.9	4.1	3.4	3.2	3.8	1.6
Strip Ratio	-	**2.1**				52.8	230.2	6.0	4.0	3.3	1.5	1.3	0.9	0.5	0.1
Ore Grade	g/t	**1.60**				0.70	0.97	0.96	0.88	0.92	1.00	1.25	1.48	2.29	4.69
Contained Gold	koz	**1,086.8**				0.5	0.9	31.4	58.8	55.4	133.8	136.0	149.9	276.6	243.5

Metric	Unit	Total	Pre-Prod.	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	7.0	6.3	4.5	2.8	1.7	0.2
Ore Mined	Mt	**39.6**	0.2	3.4	4.1	3.4	2.1	3.1	3.8	3.5	4.1	3.4	3.2	3.8	1.6
Strip Ratio	-	**1.9**	5.2	1.2	1.0	2.8	5.9	3.7	2.8	2.0	1.5	1.3	0.9	0.5	0.1
Ore Grade	g/t	**1.36**	0.93	1.09	1.15	1.12	0.89	0.94	0.97	1.13	1.00	1.25	1.48	2.29	4.69
Contained Gold	koz	**1,738.3**	5.8	119.7	151.1	122.7	58.9	92.5	120.3	127.6	133.8	136.0	149.9	276.6	243.5
BC Pit															
Waste Mined	Mt	**4.5**										2.0	2.0	0.5	
Ore Mined	Mt	**3.5**										0.7	1.8	1.0	
Strip Ratio	-	**1.3**										2.9	1.1	0.5	
Ore Grade	g/t	**1.78**										1.39	1.70	2.18	
Contained Gold	koz	**199.8**										29.8	98.0	72.1	
Total Open Pit Mining															
Waste Mined	Mt	**81.7**	3.0	4.2	3.9	9.6	12.1	11.3	10.6	7.0	6.3	6.4	4.8	2.2	0.2
Ore Mined	Mt	**43.7**	0.8	3.4	4.1	3.4	2.1	3.1	3.8	3.5	4.1	4.0	4.9	4.8	1.6
Strip Ratio	-	**1.9**	8.7	1.2	1.0	2.8	5.9	3.7	2.8	2.0	1.5	1.6	1.0	0.5	0.1
Ore Grade	g/t	**1.39**	1.66	1.09	1.15	1.12	0.89	0.94	0.97	1.13	1.00	1.28	1.56	2.27	4.69
Contained Gold	koz	**1,951.3**	18.9	119.7	151.1	122.7	58.9	92.5	120.3	127.6	133.8	165.8	247.8	348.7	243.5

8.8.5 Combined Mining and Production Schedule

The overall combined LOM production schedules are presented by source in Figure 8.24. The process plant feed is presented by lithology in Figure 8.25, by Mineral Resource classification, in terms of feed tonnes in Figure 8.26 and in terms of contained gold in Figure 8.27. Stockpile closing balances are presented in Figure 8.28. The LOM mining schedule is provided in Table 8.21 and the mill feed and production schedule in Table 8.22.

The target plant throughput rate is maintained through the life of mine, inclusive of the ramp-up in the first year. The lithological blend requirements, associated with the comminution and tailings filtration circuit operating requirements in the process plant, of minimum 25% fresh and maximum 50% saprolite, are also achieved. The feed grade is approximately 2.0 g/t for the first three years before gradually reducing to an average of approximately 1.55 g/t from year eight to year 10. This coincides with the high-grade feed from the underground reducing, with underground feed production finishing in year nine. The plant feed grade then increases to over 2.1 g/t for years 11 and 12 as the high-grade fresh material at the base of the NEB pit is mined. The final year of production, consisting of low-grade stockpiled material, has an average plant feed grade of 0.7 g/t in year 13.

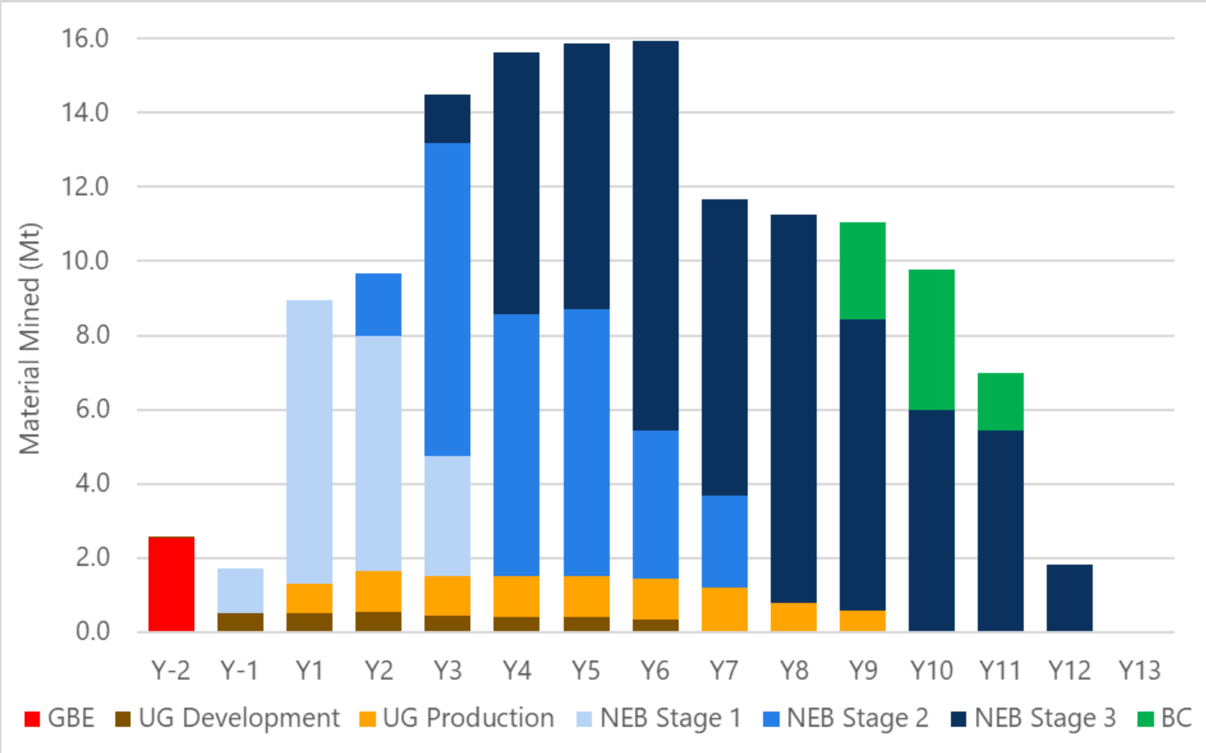

Figure 8.24: Life of Mine Schedule – Total Material Mined

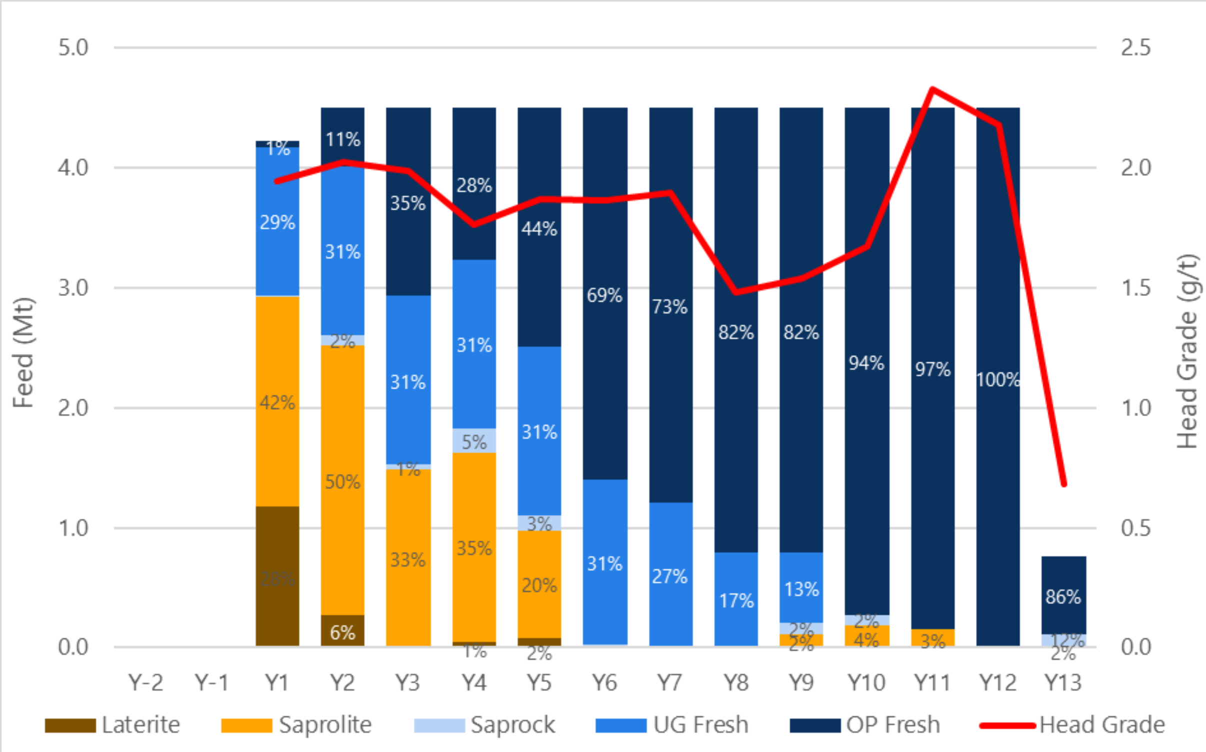

Figure 8.25: Life of Mine Schedule – Mill Feed by Lithology and Grade

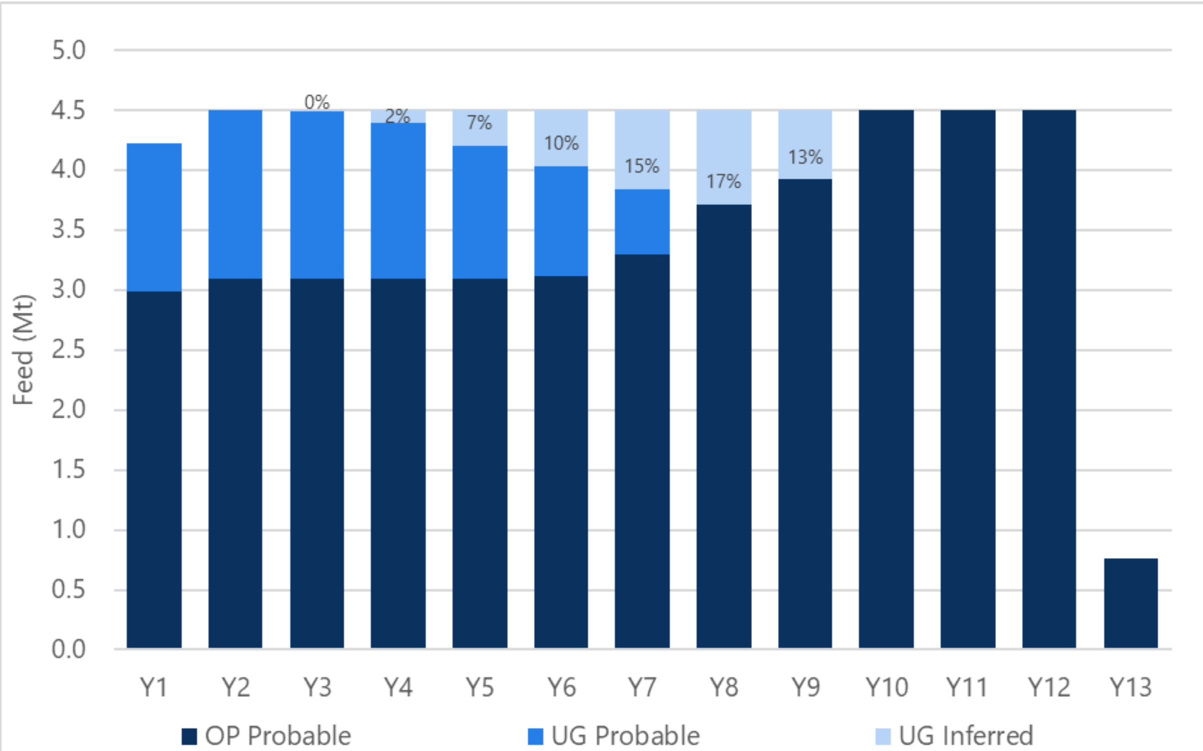

Figure 8.26: Life of Mine Schedule – Mill Feed Tonnes by Mineral Resource Classification

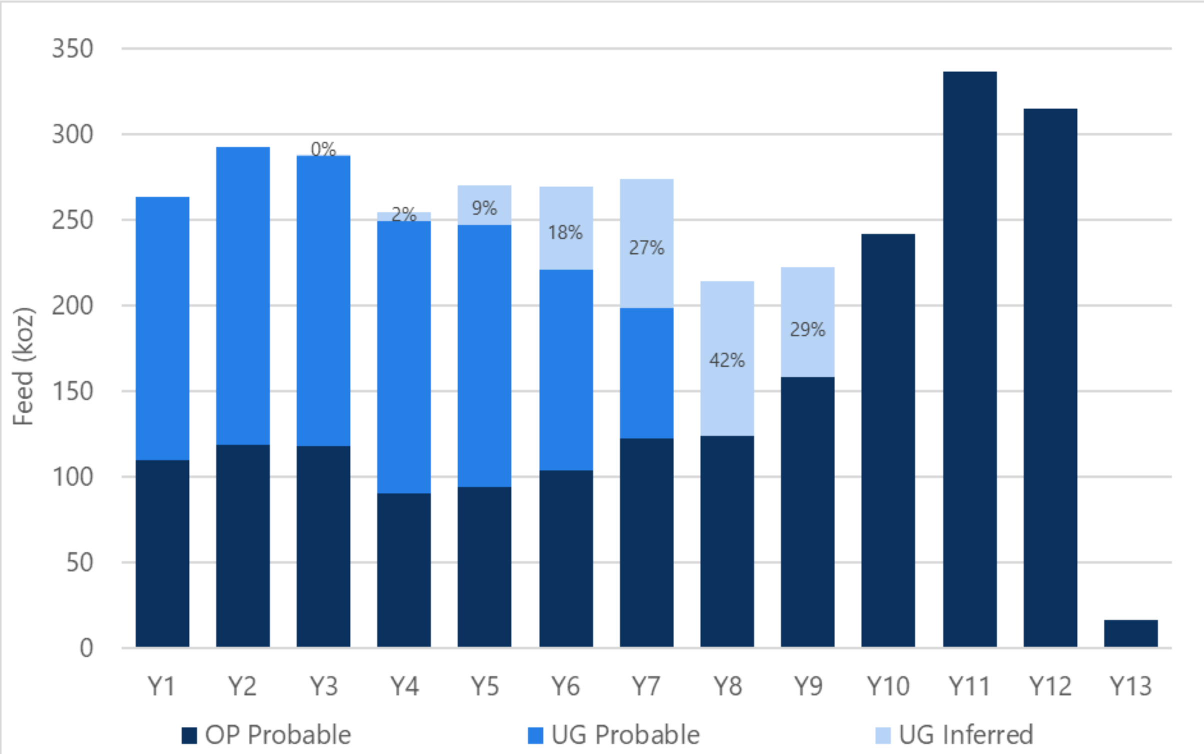

Figure 8.27: Life of Mine Schedule – Mill Feed Contained Gold by Mineral Resource Classification

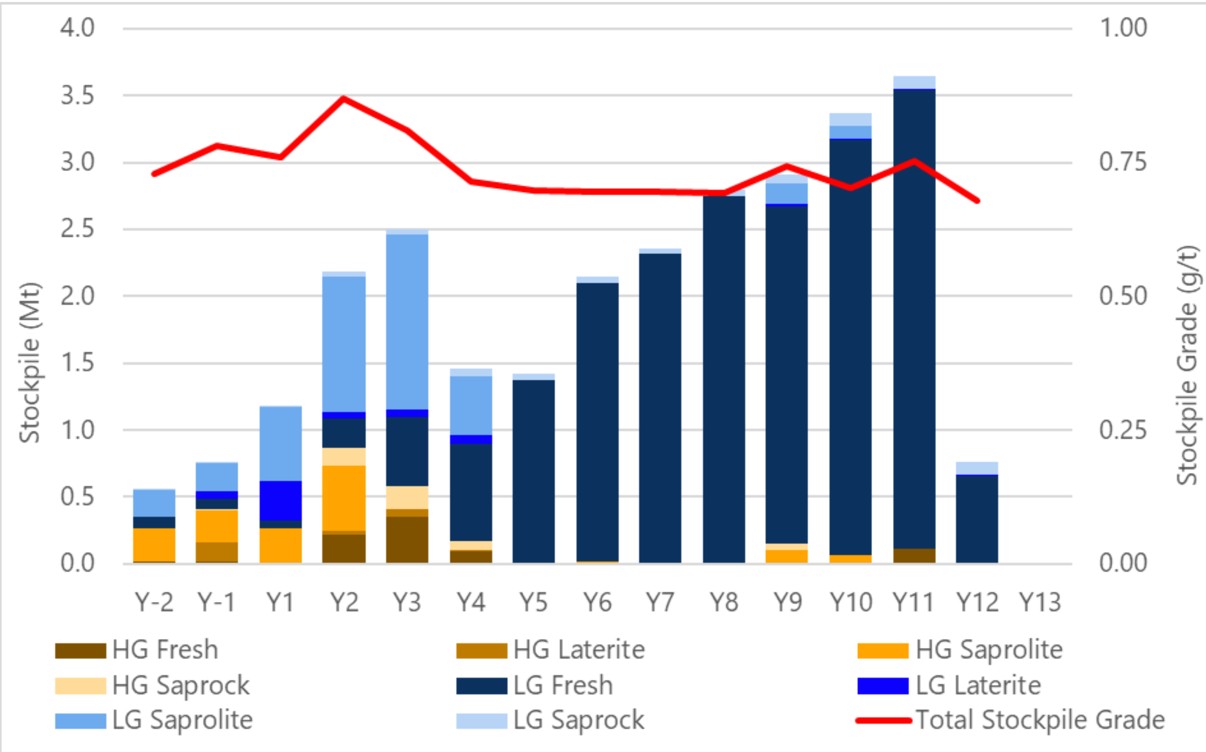

Figure 8.28: Life of Mine Schedule – Stockpile Balances by Lithology

Table 8.21: Life of Mine 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	3.16	0.55	0.49	0.54	0.43	0.40	0.41	0.33							
Production	Mt	8.87		0.82	1.10	1.06	1.10	1.10	1.12	1.20	0.78	0.58				
Underground Total	**Mt**	**12.03**	**0.55**	**1.31**	**1.64**	**1.50**	**1.51**	**1.51**	**1.45**	**1.20**	**0.78**	**0.58**				
Open Pit Mining																
GBE	Mt	2.55	2.55													
NEB Stage 1	Mt	18.45	2.21	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	8.01	10.49	7.87	5.99	5.44	1.80	
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**	**10.49**	**10.49**	**10.49**	**9.79**	**6.99**	**1.80**	
Total Mining																
Total	**Mt**	**137.43**	**4.31**	**8.94**	**9.66**	**14.48**	**15.65**	**15.87**	**15.93**	**11.68**	**11.27**	**11.07**	**9.79**	**6.99**	**1.80**	

Table 8.22: 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	1.18	0.27		0.04	0.08				0.02	0.00	0.00		0.01
	g/t	0.98	1.07	0.61		1.16	0.71				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	37.4	4.8		1.5	1.7				0.7	0.0	0.0		0.2
Saprolite	Mt	8.43	1.75	2.25	1.48	1.58	0.90	0.02	0.01	0.01	0.10	0.19	0.15		
	g/t	1.03	1.20	1.24	0.99	0.70	0.79	0.68	0.84	0.84	1.46	1.03	0.94		
	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	32.8	21.0	0.4	0.2	0.2	4.2	5.7	4.3		
Saprock	Mt	0.76	0.01	0.09	0.05	0.21	0.13	0.00			0.10	0.08			0.09
	g/t	1.34	0.89	1.20	1.21	1.14	0.99	0.82			2.47	2.19			0.60
	Recovery (%)	92.4	92.0	92.2	92.2	92.1	92.1	92.0			92.8	92.7			91.8
	Production (koz)	30.0	0.2	3.1	1.7	6.9	3.7	0.1			7.2	5.3			1.7
OP Fresh	Mt	32.91	0.05	0.49	1.57	1.27	1.99	3.10	3.30	3.71	3.71	4.23	4.35	4.50	0.65
	g/t	1.50	0.79	1.31	1.38	1.12	1.02	1.04	1.16	1.04	1.22	1.69	2.37	2.18	0.69
	Recovery (%)	92.4	91.9	92.2	92.2	92.1	92.1	92.1	92.1	92.1	92.2	92.4	92.8	92.7	91.9
	Production (koz)	1,468.3	1.1	19.1	64.2	42.0	60.2	95.3	113.0	114.2	133.9	212.2	307.7	291.9	13.4
UG Fresh	Mt	10.79	1.24	1.40	1.40	1.40	1.40	1.38	1.20	0.78	0.58				
	g/t	3.77	3.88	3.85	3.75	3.65	3.91	3.73	3.93	3.58	3.46				
	Recovery (%)	93.5	93.6	93.6	93.5	93.4	93.6	93.5	93.6	93.4	93.3				
	Production (koz)	1,223.9	144.1	162.5	158.0	153.5	165.2	154.8	141.5	84.4	60.0				
Total	**Mt**	**54.48**	4.22	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	0.76
	g/t	**1.86**	1.95	2.02	1.99	1.76	1.87	1.86	1.89	1.48	1.54	1.67	2.33	2.18	0.68
	Recovery (%)	**92.8**	93.0	93.0	92.9	92.9	93.0	92.9	92.9	92.6	92.5	92.4	92.7	92.7	91.9
	Production (koz)	**3,025.8**	245.4	272.0	267.3	236.8	251.8	250.6	254.7	198.8	206.0	223.3	312.0	291.9	15.3

8.9 Ore Reserve

Orelogy has completed an Ore Reserve estimate for the Project based on the resource block model used in the preparation of the Mineral Resource estimate published on 7 August 2023 and reported in Section 4. The Ore Reserve estimate has been prepared in accordance with the guidelines of the JORC Code (2012). The estimate comprises an open pit and underground Ore Reserve, for a total of 51.6 Mt at 1.78 g/t for 2.95 Moz of gold.

PDI and Orelogy confirm that they are not aware of any new information or data that materially affects the information included in this DFS and the corresponding market announcement, and that all material assumptions and technical parameters underpinning the Ore Reserve estimates continue to apply and have not materially changed. At the time of completing this DFS, there are no known environmental, legal, socio-economic, marketing or other relevant conditions, that would materially affect the estimated Ore Reserves.

The Ore 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 Ore Reserve estimate is based on Indicated Mineral Resources and no Inferred Mineral Resource was included in the Ore Reserve. The Ore Reserve represents the economically mineable part of the Indicated Mineral Resources, and the reported Mineral Resource estimate is inclusive of the Ore Reserves.

The modelled ore loss and dilution varies by location with the block model, but average 4.4% and 3.6% respectively within the Ore Reserve (refer to).

The Ore Reserve was reported using variable cut-off grades as processing costs vary by material type and metallurgical recovery varies by diluted head grade.

Therefore, ore was flagged on a block-by-block basis within the model. Table 8.23 below provides the average breakeven cut-off grade by material type based on the average mined head grade.

Table 8.23: Ore Reserve Cut-off Grade

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
Fresh – Underground	As per Table 8.11			2.00

The proposed mine plan is technically achievable. All technical proposals made for the operational phase involve the application of conventional technology that is widely utilised in the gold industry in West Africa. Financial modelling completed as part of the DFS shows that the Project is economically viable under current assumptions. Material modifying factors (mining, processing, infrastructure, environmental, legal, social and commercial) were considered during the Ore Reserve estimation process.

A LOM mining and production schedule was also developed with the Inferred component from the underground mining schedule removed and the open pit mining schedule adjusted to maintain full mill feed capacity. This was then assessed using the project financial model to ensure the combined open pit and underground Ore Reserves were economically viable without the inclusion of the inferred material. The financial model based on this Ore Reserve only production schedule indicated robust financial metrics Ore Reserve evaluation gold price noted below, confirming the Ore Reserve as valid and supportable. The open pit Ore Reserve was not assessed independent of the underground Ore Reserve as the open pit mine plan was fundamentally constrained by the underground mine.

The Bankan Gold Project Ore Reserve, which is classified as Probable Ore Reserves with no Proven Ore Reserves is outlined in Table 8.24.

Table 8.24: Bankan Gold Project Ore 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	Probable	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 Ore Reserve conforms with and uses the JORC Code (2012) definitions.
2. The Ore Reserve was evaluated using a gold price of US$1,800/oz.
3. The Ore Reserve was evaluated using variable cut-off grades as described in Table 8.23.
4. Ore block grade and tonnage dilution was incorporated into the model.
5. All figures are rounded to reflect appropriate levels of confidence.
6. Apparent differences may occur due to rounding.
7. The Competent Person responsible for the Reserve is Mr Ross Cheyne, Principal Consultant with Orelogy Consulting Pty Ltd. Mr Cheyne visited the site in January 2025.

9 PROCESSING

9.1 Design

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.

A design blend of 75% fresh ore and 25% weathered ore was selected for the comminution circuit design and the 85th percentile ore physical properties was used for the design and sizing of the grinding mills and other comminution equipment. Under average operating conditions, based on the 50th percentile ore physical properties, the selected equipment will accommodate 100% fresh ore producing the design grind size within the capacity of the SAG and ball mills. The average ore physical properties were also used for calculation of typical operating SAG and ball mill power.

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.
- Blend of 50% weathered and 50% fresh ore.

The worst-case value (typically highest) was used as the design value. Table 9.1 lists the key process design criteria and equipment sizing.

Table 9.1: 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	$t/h/m^2$	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	m^3	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	m^3	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

9.2 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.

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 9.1.

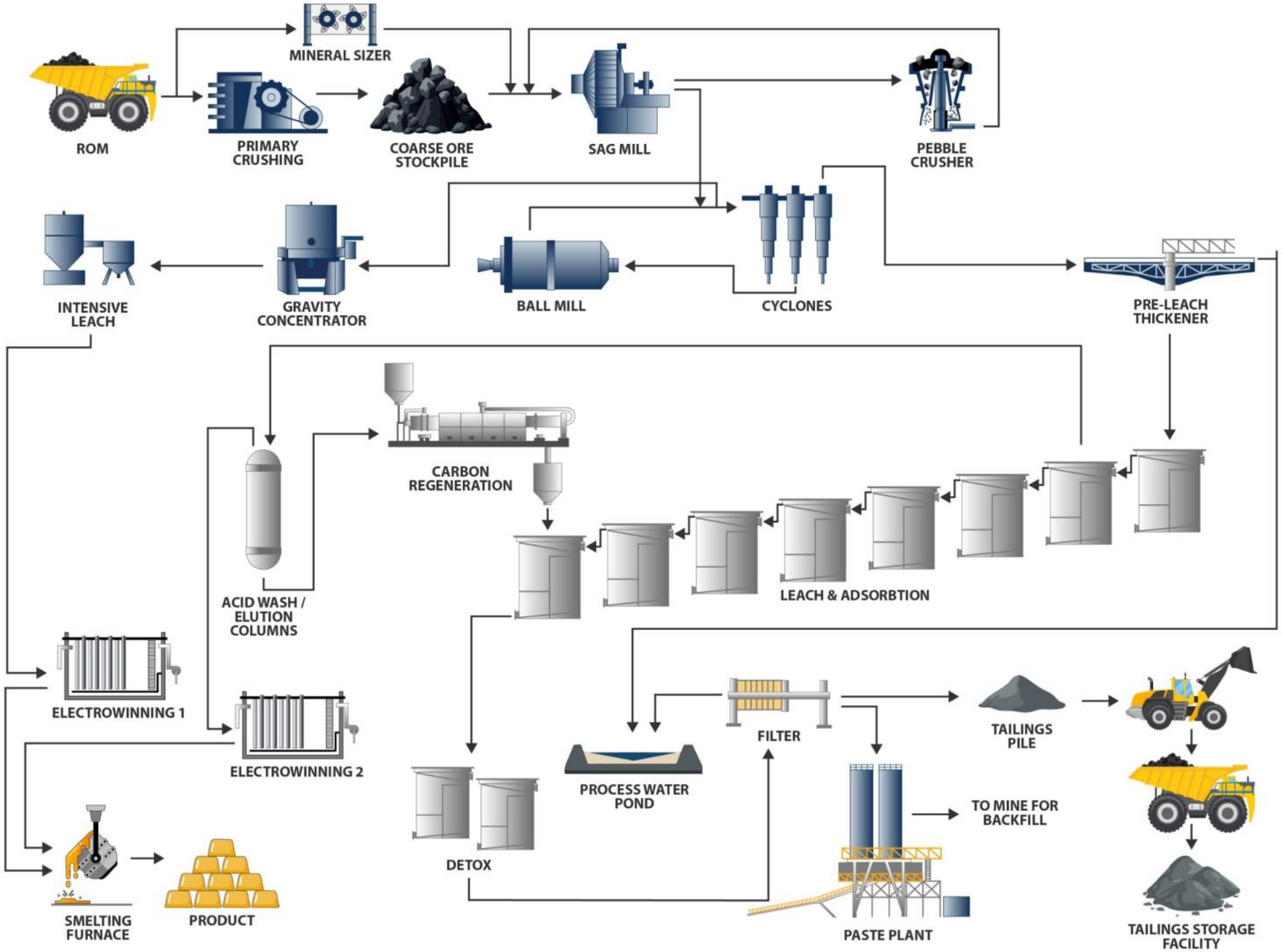

Figure 9.1: Process Flowsheet

10 TAILINGS DISPOSAL

10.1 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 10.1.

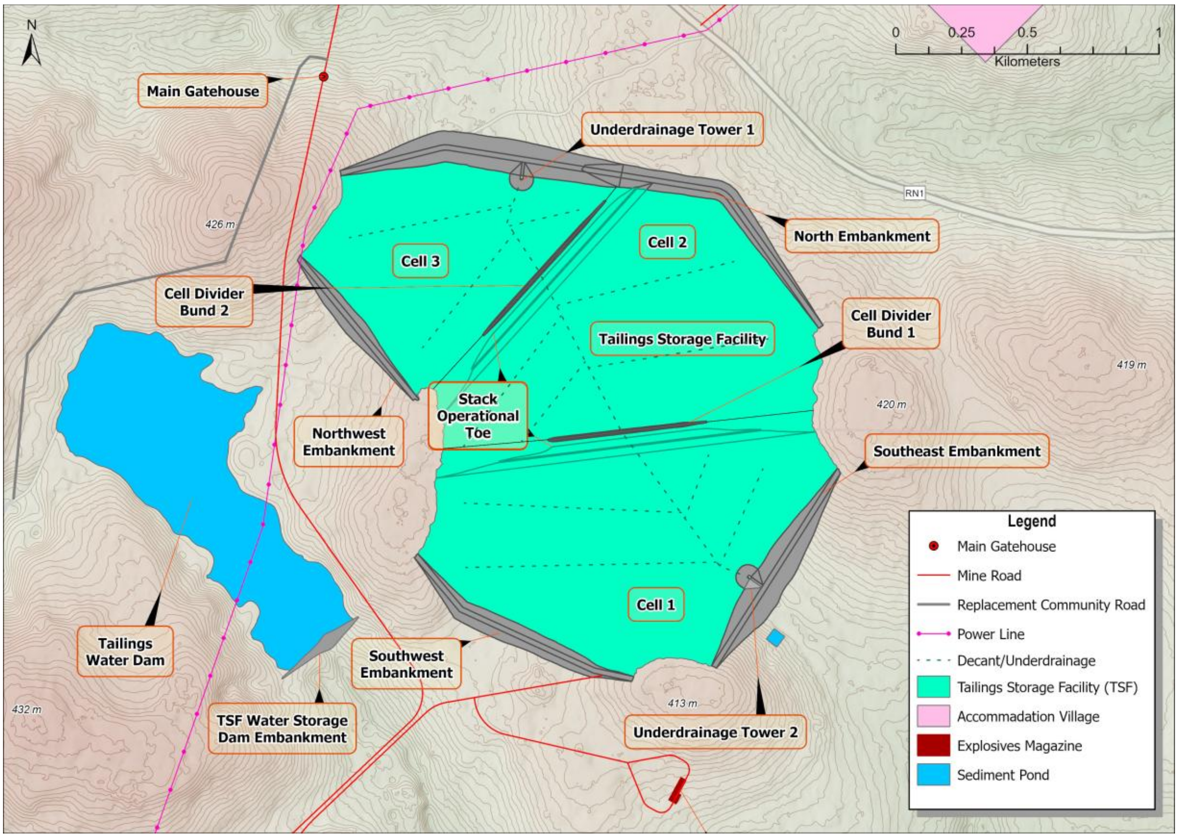

Figure 10.1: TSF and Tailing Water Storage Dam

10.2 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.

10.3 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 10.2 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.

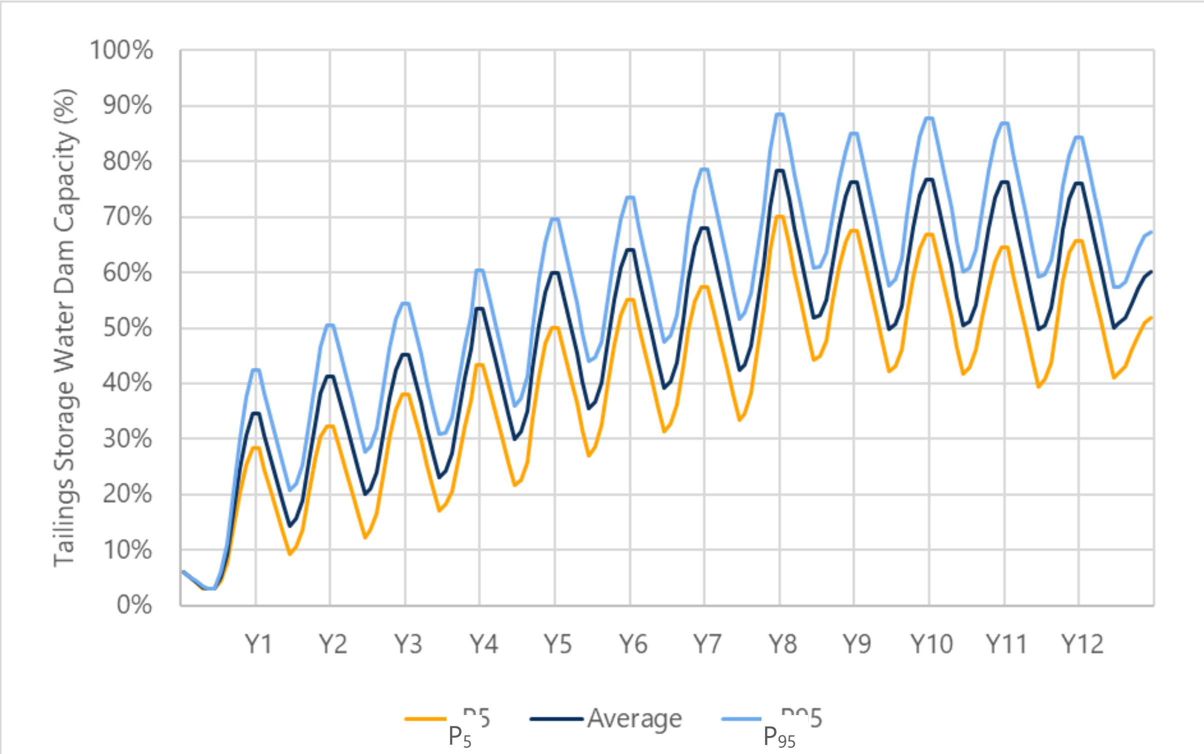

Figure 10.2: TSF Water Balance Probabilistic Analysis Results

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.

11 INFRASTRUCTURE

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

- Access roads.
- Offices, warehouses, workshops and other buildings.
- Accommodation village.
- Power supply and site power distribution.
- Water supply and management.
- Mining infrastructure.
- Fuel storage and supply.
- Paste plant.
- Other infrastructure.

An overall site layout is provided in Figure 11.1, which shows the supporting infrastructure and services, mining locations, and processing plant infrastructure.

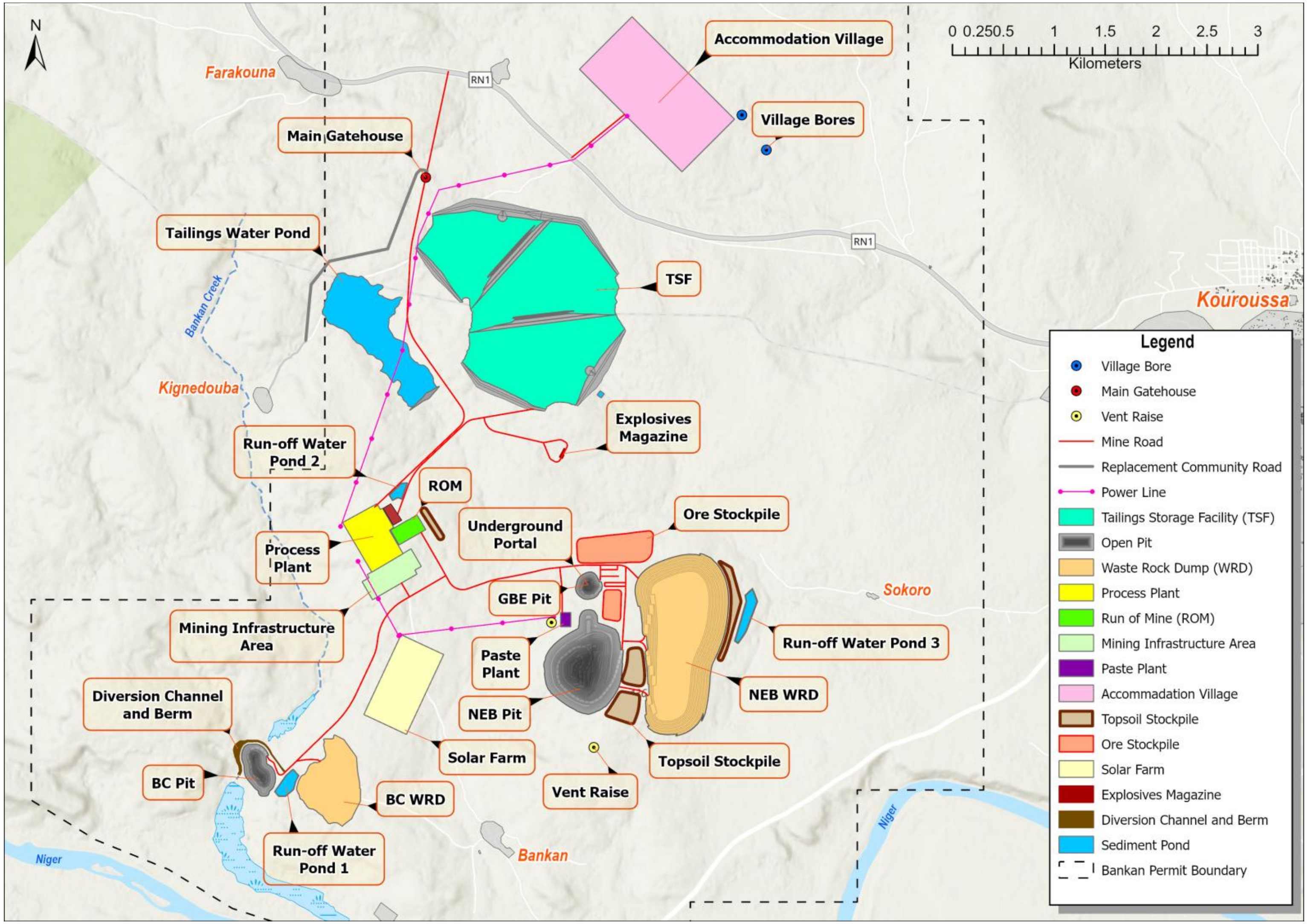

Figure 11.1: Overall Site Layout

11.1 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.

11.2 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.

11.3 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.

11.4 Power Supply and Distribution

The estimated load for the Project is outlined in Table 11.1.

Table 11.1: 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.

11.4.1 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.

11.5 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.

11.6 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.

11.7 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.

11.8 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 11.2.

Table 11.2: 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

11.9 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.

12 OPERATIONS

12.1 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.

12.2 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).

12.3 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 12.1 has been adopted for the DFS.

Table 12.1: 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.

12.4 Human Resources

The high-level organisation chart for the Project is presented in Figure 12.1.

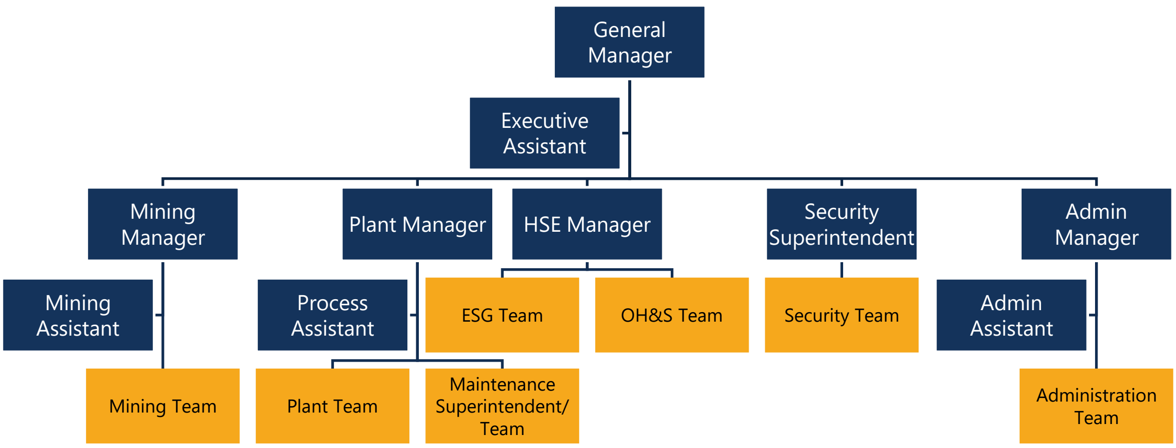

Figure 12.1: High Level Organisation Chart

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 12.2 provides the breakdown of the total personnel, broken down by department, staff and contractors, and expatriate and local for the Project operations.

Table 12.2: 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**
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.

12.5 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.

13 ENVIRONMENTAL, SOCIAL, APPROVALS AND LAND ACCESS

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 MEDD approved the ESIA and issued the ECC (CCE/00070) for the Project on 17 January 2025.

The Company is committed to complying with all relevant Guinean national laws and regulations, international standards, such as the IFC Performance Standards on Environmental and Social Sustainability (2012) (IFC-PS), and best practice standards in the Environmental, Health and Safety (EHS) Guidelines for Mining (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. 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.

13.1 Environmental and Social Baseline Studies

The existing physical, biological and social conditions of the area were assessed and classified as an area of influence 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.

13.2 Environmental and Social 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 the ESIA.

13.3 Environmental and Social Management Plans

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.

13.4 Mine Closure

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.

As described in the International Council on Mining and Metals (ICMM) and the IFC Environmental Health and Safety Guidelines for Mining, an effective mine closure (including planning and implementation), takes into consideration the views, 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 to 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.

13.4.1 Mine Closure and Post Closure Cost Estimate

The closure cost estimate was generated from first principles and tabulated into a basis of estimate. This details assumptions, rates and quantities for the closure of each domain. The closure cost has been estimated at US$36.5m, excluding mine WRD reprofiling which is carried out as part of the mining costs, and has been updated from the ESIA to reflect the current Project design and closure costs in 2025.

In addition, based on the objectives 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.

13.5 Land Access

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.

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.

14 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.

14.1 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.

14.2 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 14.1.

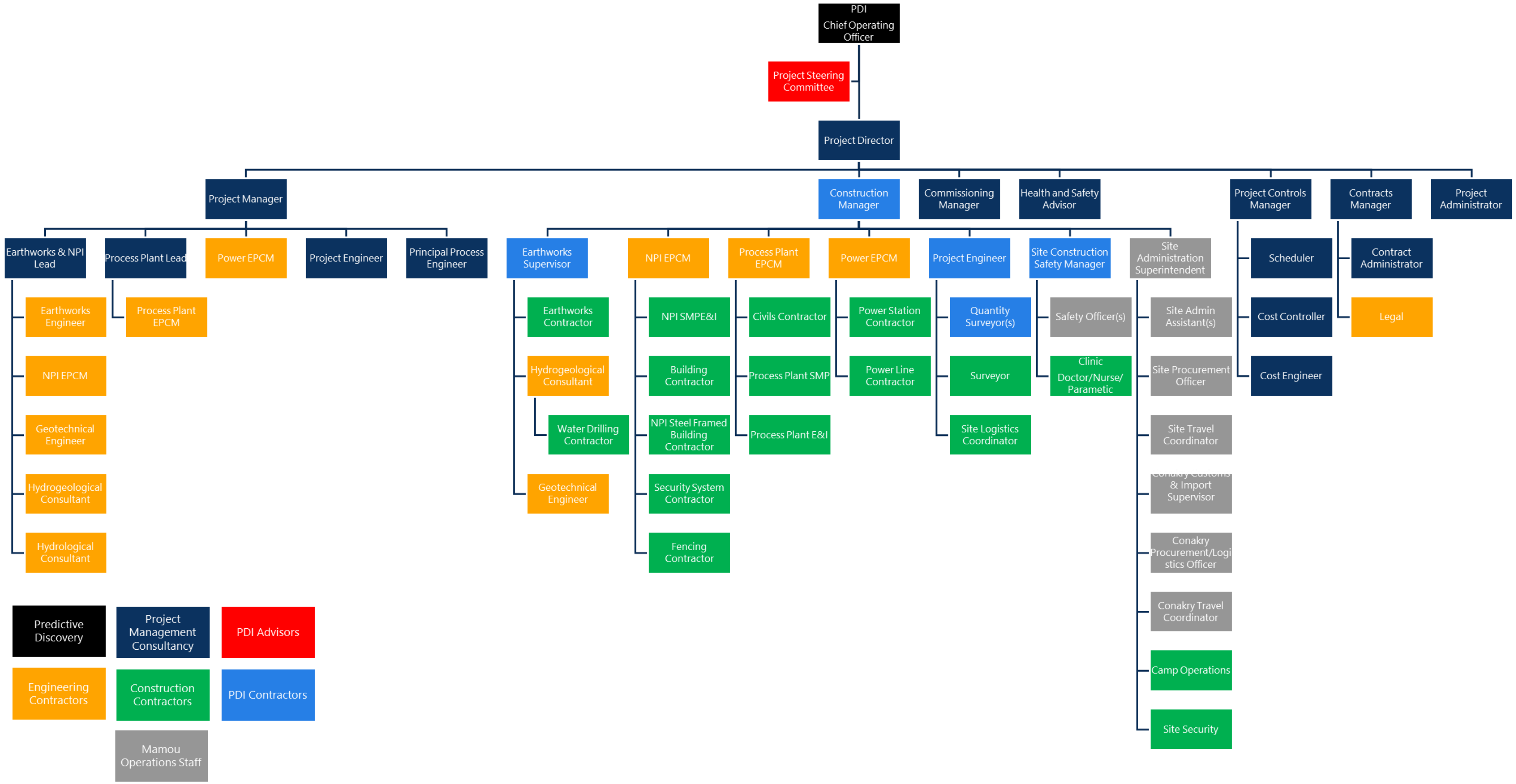

Figure 14.1: Owners Team Organisation Structure

14.3 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.

14.4 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.

14.5 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.

14.6 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 14.2 shows the build up in workforce, excluding contractors, prior to the commencement of production.

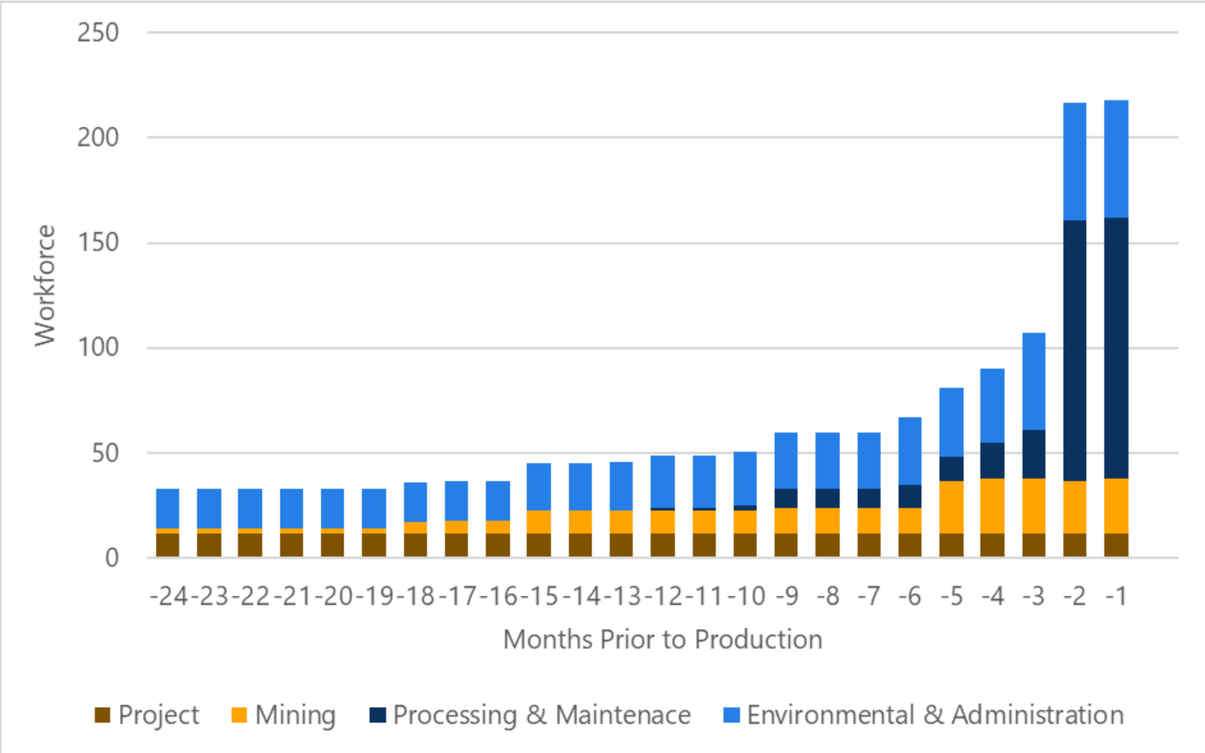

Figure 14.2: Pre-Production Workforce Ramp-up

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.

14.7 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.

14.8 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 14.5 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 14.1, to be considered to not be on the critical path.

Table 14.1: Schedule Float

Area	Days Float
Mining	0
Process Plant	23
Power Station	86

Figure 14.3 provides a summary implementation schedule.

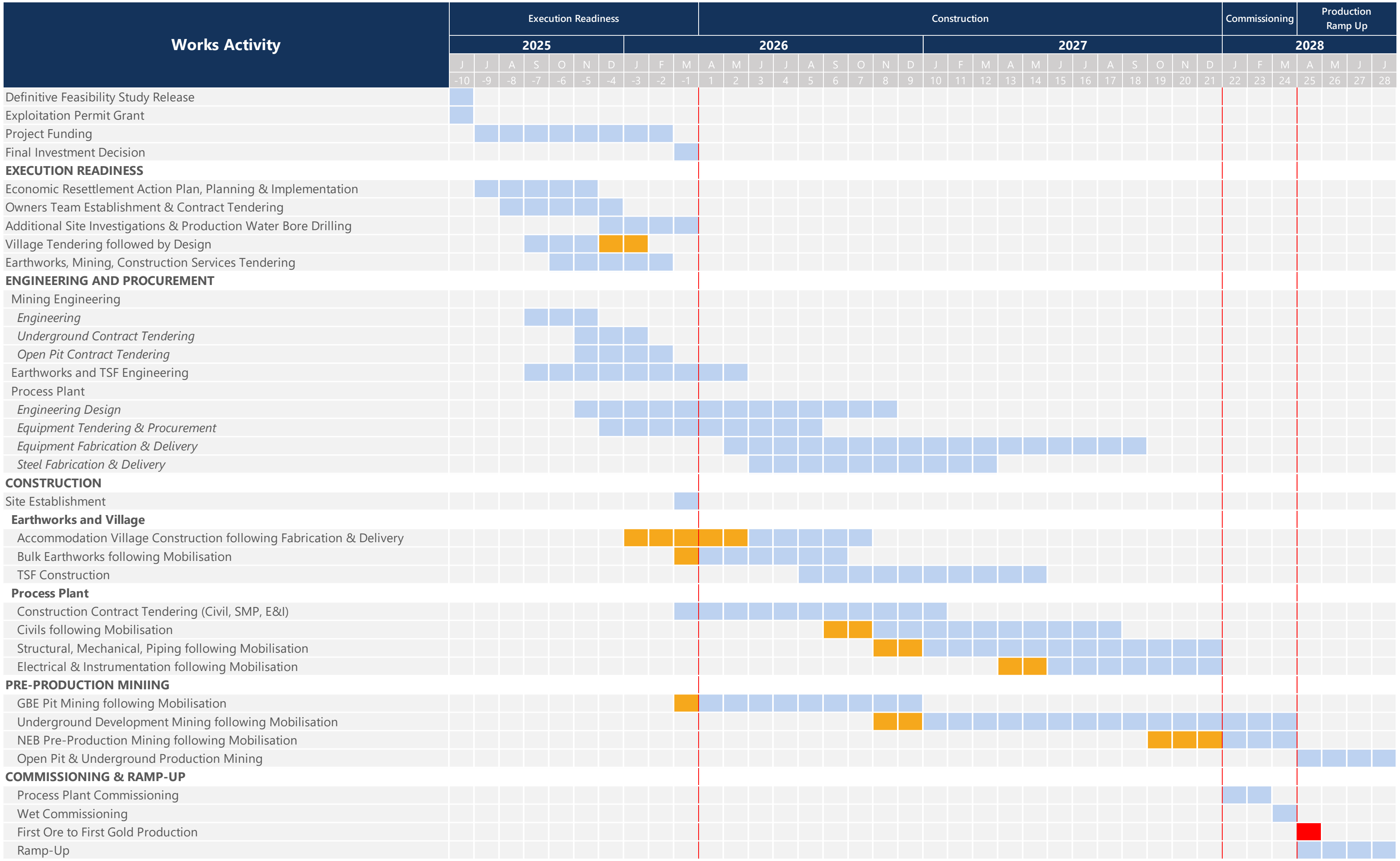

Figure 14.3: Summary Project Implementation Schedule

15 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 14. 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.

15.1 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 14 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).

15.2 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 15.1 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 15.1: 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

15.3 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.

15.4 Capital Cost Estimate

As summarised in Table 15.2 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$463m, 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 15.2: Capital Cost Estimate

Area	US$m[1]
Mining	
Pre-production Open Pit Mining	42.9
Pre-production Underground Mining	62.7
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 15.5.
- 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 mining inventory identified in this study.

15.5 Execution Readiness Costs

The execution readiness costs outlined in Table 15.3 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 14.5.

Table 15.3: 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**

15.6 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 15.4, is US$208.6m.

Table 15.4: Life of Mine Sustaining and Deferred Capital Estimate

Area	US$m
Underground Development	135.7
Refrigeration Plant	4.6
BC Pit Establishment	5.7
TSF Expansion	54.9
TSF Progressive Closure	3.6
General Sustaining Capital	4.2
Total	**208.6**

15.7 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.

16 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 16.1 presents the life of mine operating costs for the Project.

Table 16.1: Life of Mine Operating Costs

Area	LOM Cost US$m	Unit Costs	Unit Cost US$/oz
Open Pit Mining	604.5	US$4.97/t material mined	199.77
Underground Mining	643.3	US$60.04/t ore mined	212.60
Processing	913.9	US$16.78/t ore milled	302.05
Tailings Handling	181.7	US$3.33/t milled	60.00
General and Administration	174.4	US$14.3m per annum	57.60
Transport & Refining	25.6	US$8.45/oz	8.45
C1 Cash Costs	**2,543.3**		**840.50**
Royalties	435.7	6 % of Revenue	144.00
Sustaining Capital & Closure Costs	220.2	-	73.00
All-in Sustaining Costs	**3,199.2**		**1,057**

The basis for estimating these costs is outlined in the sections below.

16.1 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.

16.2 Mining Costs

16.2.1 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 contractors 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 contractors 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 16.2, which provides the open pit unit mining rates, Figure 16.1 showing the fixed contractor costs over the LOM, Table 16.3 and Figure 16.2 which provide the fixed inputs into the owner supplied consumption and the variation in consumption over the LOM respectively.

Table 16.2: 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

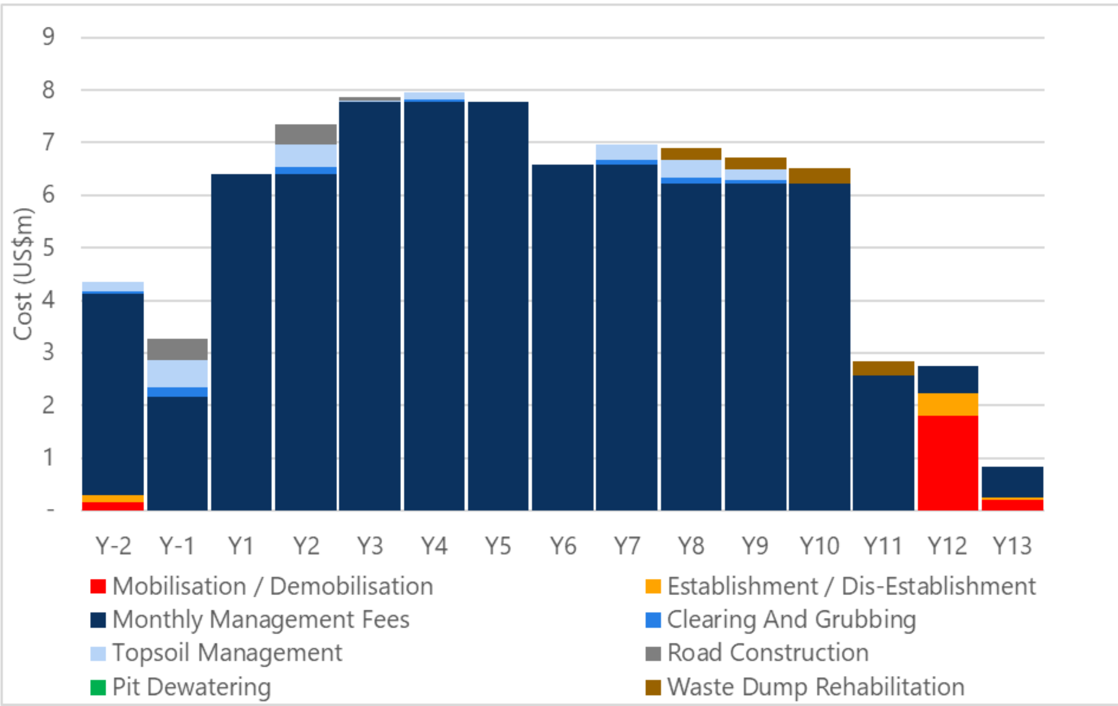

Figure 16.1: Fixed Open Pit Contractor Costs over LOM

Table 16.3: 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

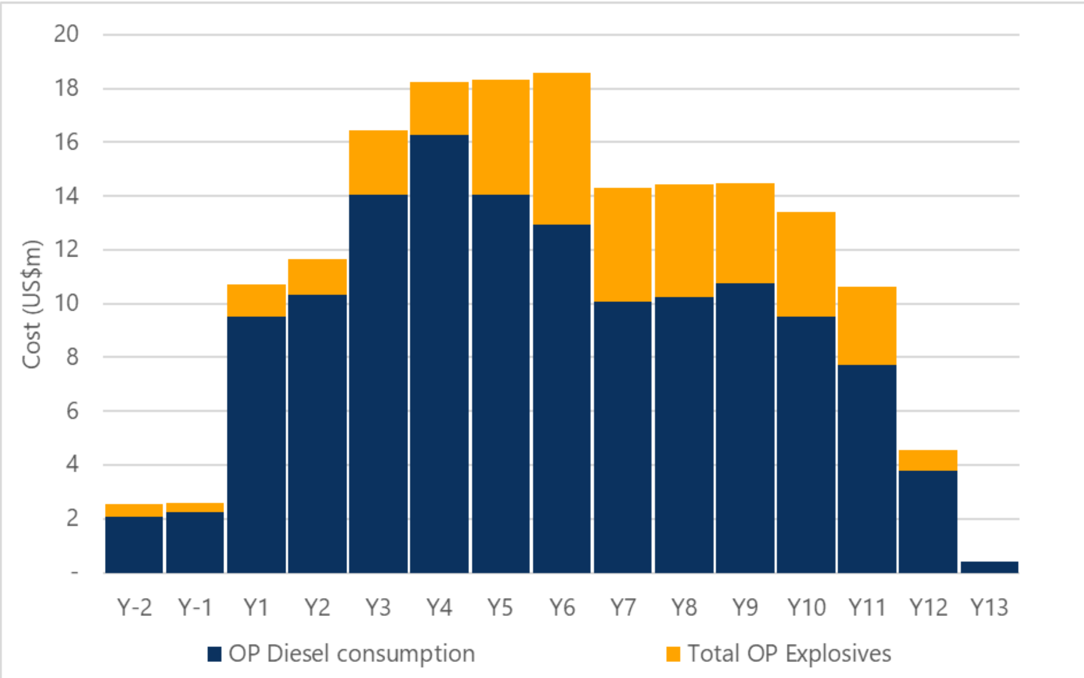

Figure 16.2: Open Pit Diesel and Explosives Usage over LOM

16.2.2 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 put 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 contractors infrastructure.

In addition to the contractors 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 16.4, which provides the underground mining rates, Figure 16.3 showing the fixed contractor costs over the LOM and Table 16.5 and Figure 16.4 which provide the fixed inputs into the owner the variation in owners supplied consumption over the LOM respectively.

Table 16.4: 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.

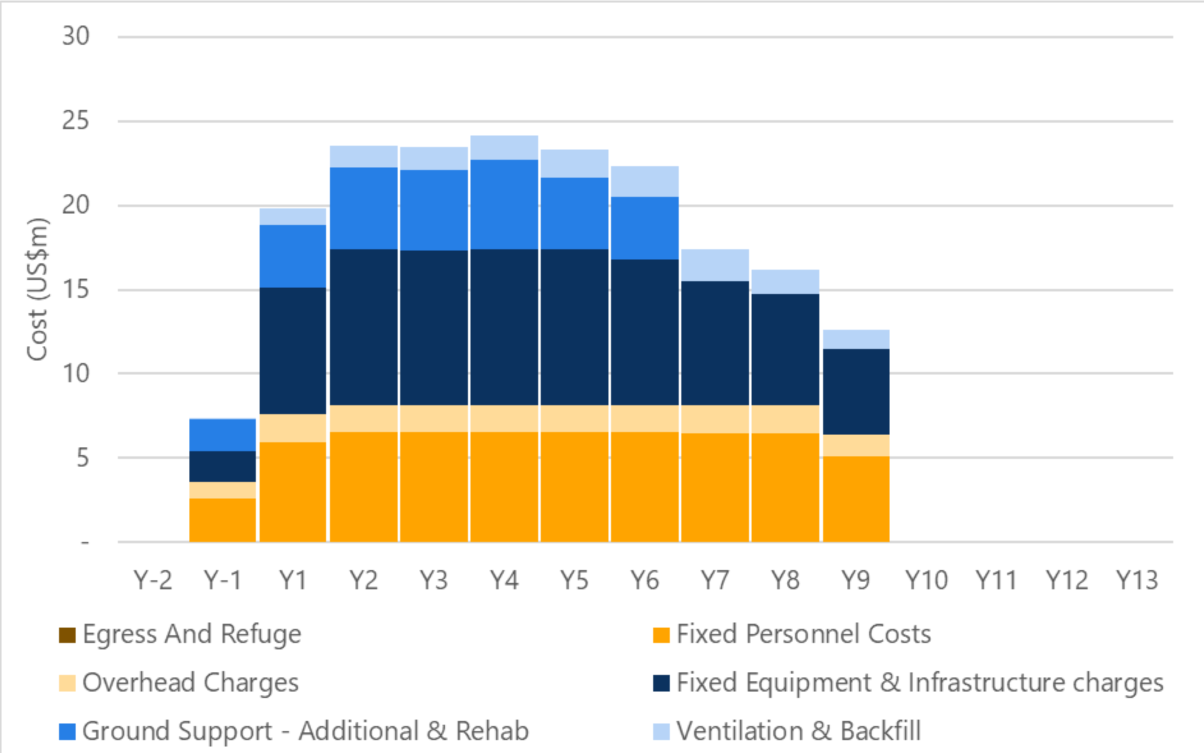

Figure 16.3: Fixed Underground Contractor Costs over LOM

Table 16.5: 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

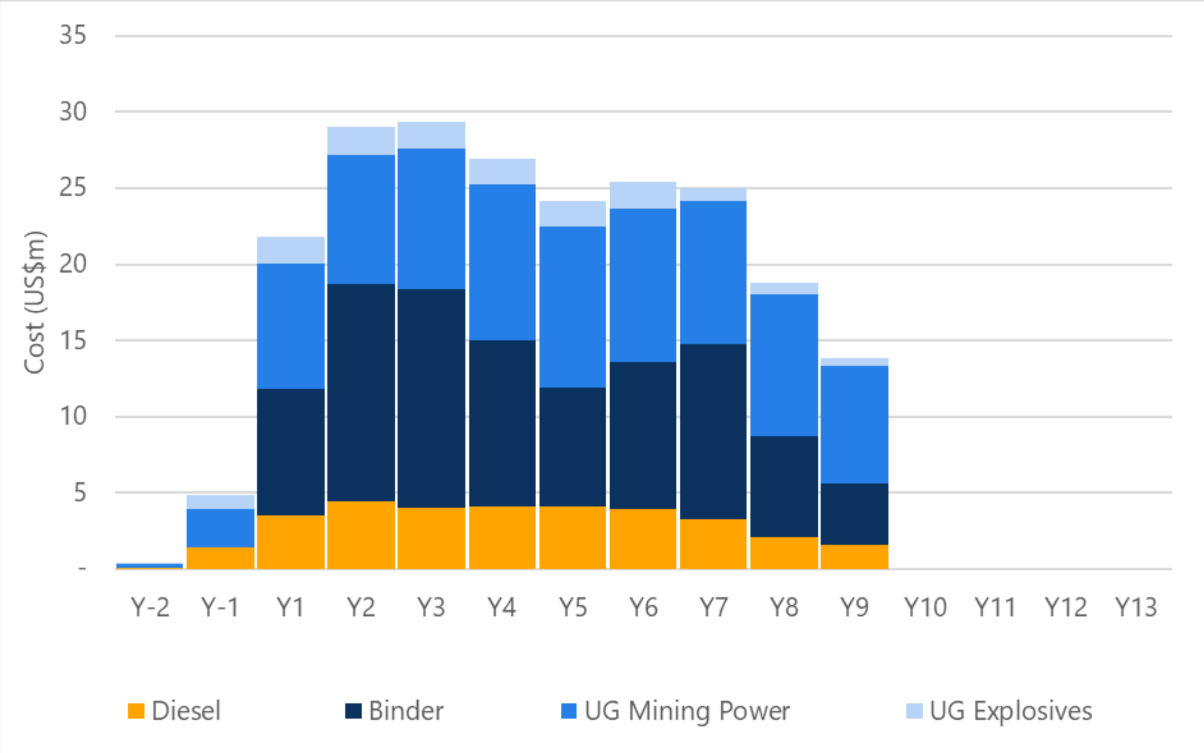

Figure 16.4: Underground Owner Supplied Diesel and Explosives Costs over LOM

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.

16.3 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 a 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 16.6 based on operations with both open pit and underground mining underway.

Table 16.6: 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

16.4 Power

16.4.1 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.

16.4.2 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 16.7.

Table 16.7: 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

16.5 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 16.8 for the average operating blend that includes 20% weathered and lime consumption consistent with that blend.

Table 16.8: 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

16.6 Consumables

Specialised consumables for mechanical equipment have been included as presented in Table 16.9. 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 16.9: 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

16.7 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.

16.8 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.

16.9 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.

16.10 General and Administration

General and administration costs are considered fixed costs. A summary of the general and administration costs included are presented in Table 16.10.

Table 16.10: 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**

16.11 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 16.11, was developed.

Table 16.11: 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 16.12, were back calculated for use in the cost estimation.

Table 16.12: 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.

17 FINANCIAL 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. A 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.

17.1 Key Assumptions

Physical assumptions are based on the mine and production schedule presented Section 8. Capital, sustaining capital, operating and closure costs are as per Section 15 and Section 16. Other key financial assumptions are set out in Table 17.1

Table 17.1: 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.

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 15.1.

17.2 Key Financial Outcomes

The financial analysis was developed based on mining inventory presented for the NEB open pit, NEB underground and BC open pit. Production averages approximately 249,000 oz annually over 12.2 years at an AISC of US$1,057/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$1,637m, IRR of 46% and a payback period of 1.9 years.

Financial outcomes improve significantly if the spot gold price of US$3,300/oz is adopted, with post-tax $NPV_{5\%}$ of US$2,893m, IRR of 73% and a payback period of 1.1 years.

The key Project outcomes for the Project are provided in Table 17.2.

Table 17.2: Key Project Outcomes

<table>
<tr><th></th><th>Units</th><th>Base Case US$2,400/oz</th><th>Spot Price US$3,300/oz</th></tr>
<tr><td colspan="4">Production</td></tr>
<tr><td>Mine Life</td><td>years</td><td colspan="2">12 years and 2 months</td></tr>
<tr><td>Total Gold Production</td><td>koz</td><td colspan="2">3,026</td></tr>
<tr><td>Average Gold Production</td><td>koz/a</td><td colspan="2">249</td></tr>
<tr><td>Proportion Inferred (contained gold)</td><td>%</td><td colspan="2">9.4</td></tr>
<tr><td colspan="4">Costs</td></tr>
<tr><td>Pre-Production Capital Costs</td><td>US$m</td><td colspan="2">463.0</td></tr>
<tr><td>Sustaining Capital Costs</td><td>US$m</td><td colspan="2">208.6</td></tr>
<tr><td>Mine Closure Costs (excluding salvage)</td><td>US$m</td><td colspan="2">39.6</td></tr>
<tr><td>C1 Cash Costs</td><td>US$/oz</td><td colspan="2">841</td></tr>
<tr><td>All-in Sustaining Costs (AISC)</td><td>US$/oz</td><td>1,057</td><td>1,111</td></tr>
<tr><td colspan="4">Financial</td></tr>
<tr><td>Pre-Tax $NPV_{5\%}$</td><td>US$m</td><td>2,279</td><td>4,022</td></tr>
<tr><td>Pre-Tax IRR</td><td>%</td><td>58</td><td>89</td></tr>
<tr><td>Pre-Tax Payback Period</td><td>Years</td><td>1.6</td><td>1.0</td></tr>
<tr><td>Post-Tax $NPV_{5\%}$</td><td>US$m</td><td>1,637</td><td>2,893</td></tr>
<tr><td>Post-Tax IRR</td><td>%</td><td>46</td><td>73</td></tr>
<tr><td>Post-Tax Payback Period</td><td>Years</td><td>1.9</td><td>1.1</td></tr>
</table>

Annual production (Figure 17.1), operating costs (AISC) (Figure 17.2) and project cashflows (Figure 17.3) are presented graphically below.

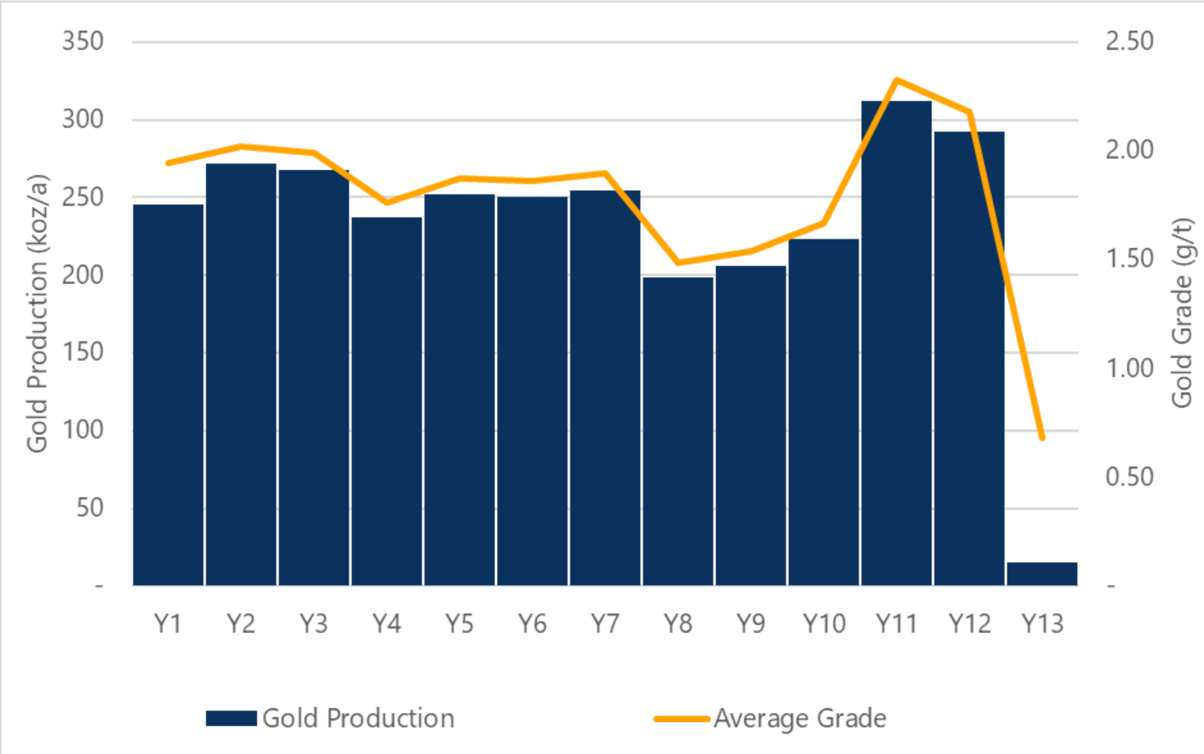

Figure 17.1: Gold Production and Grade

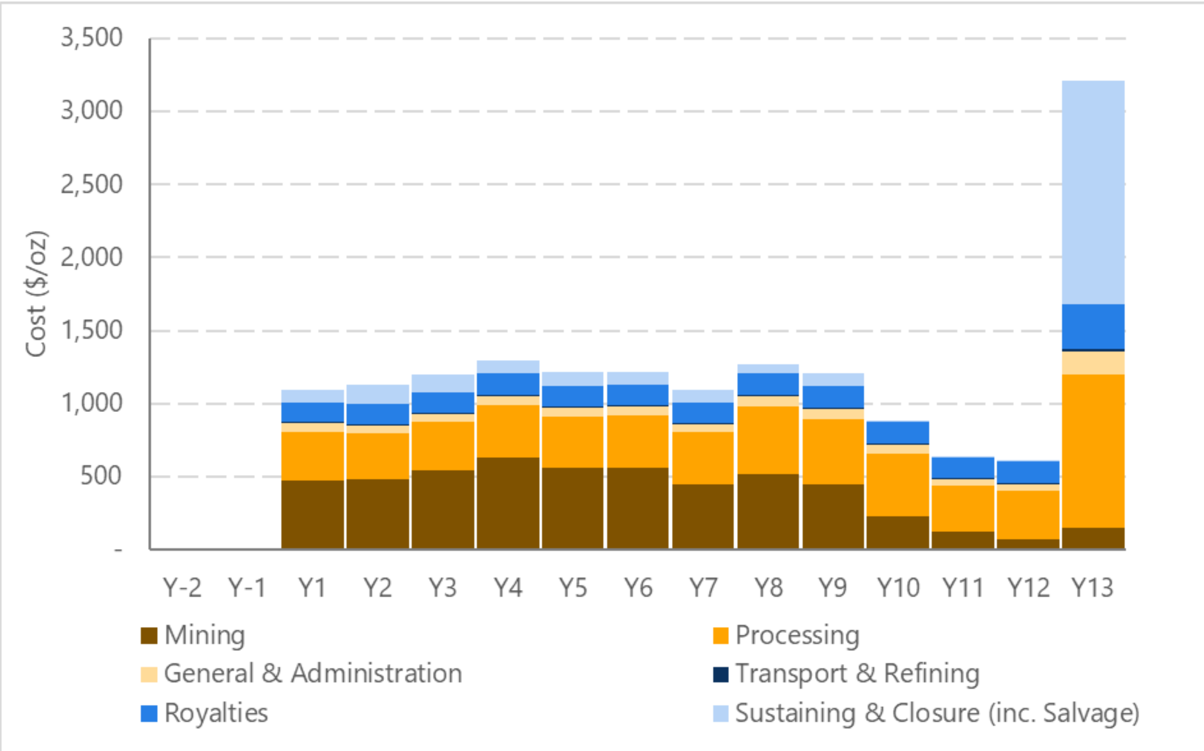

Figure 17.2: All-in Sustaining Cost per Oz

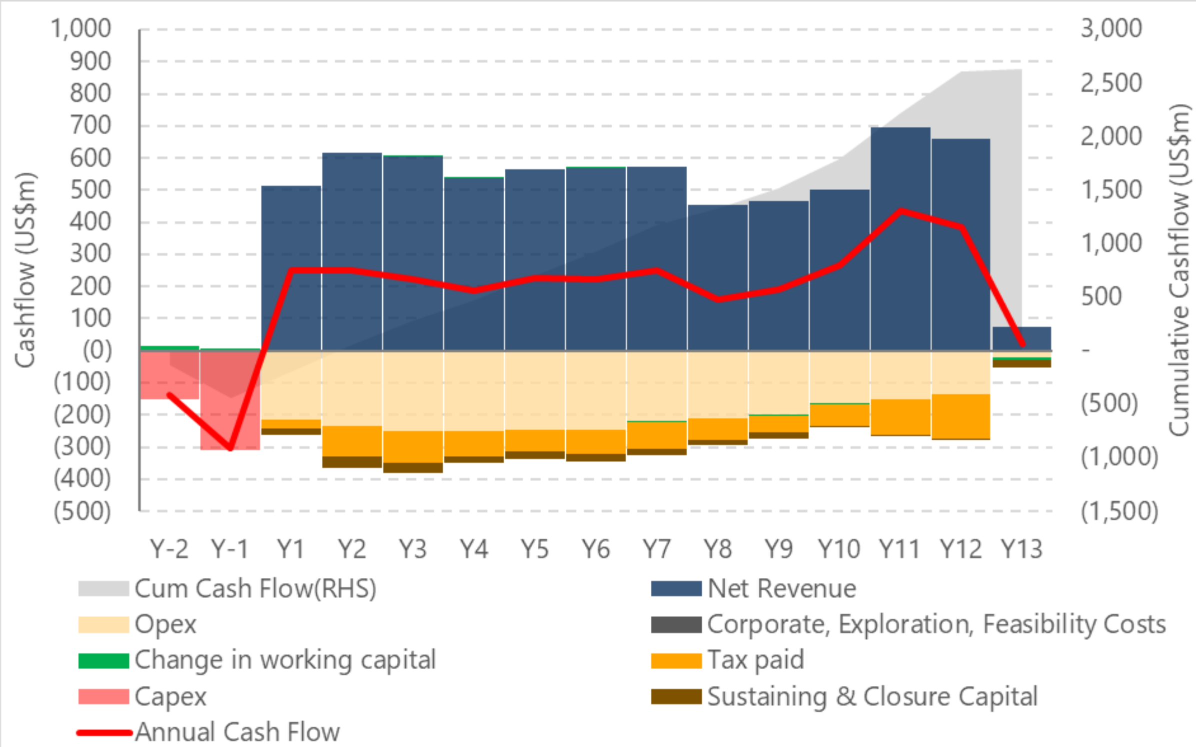

Figure 17.3: Project Cash Flow

17.3 Sensitivity Analysis

The sensitivity of the post-tax $NPV_{5\%}$ to changes in key assumptions are shown in Figure 17.4. 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.

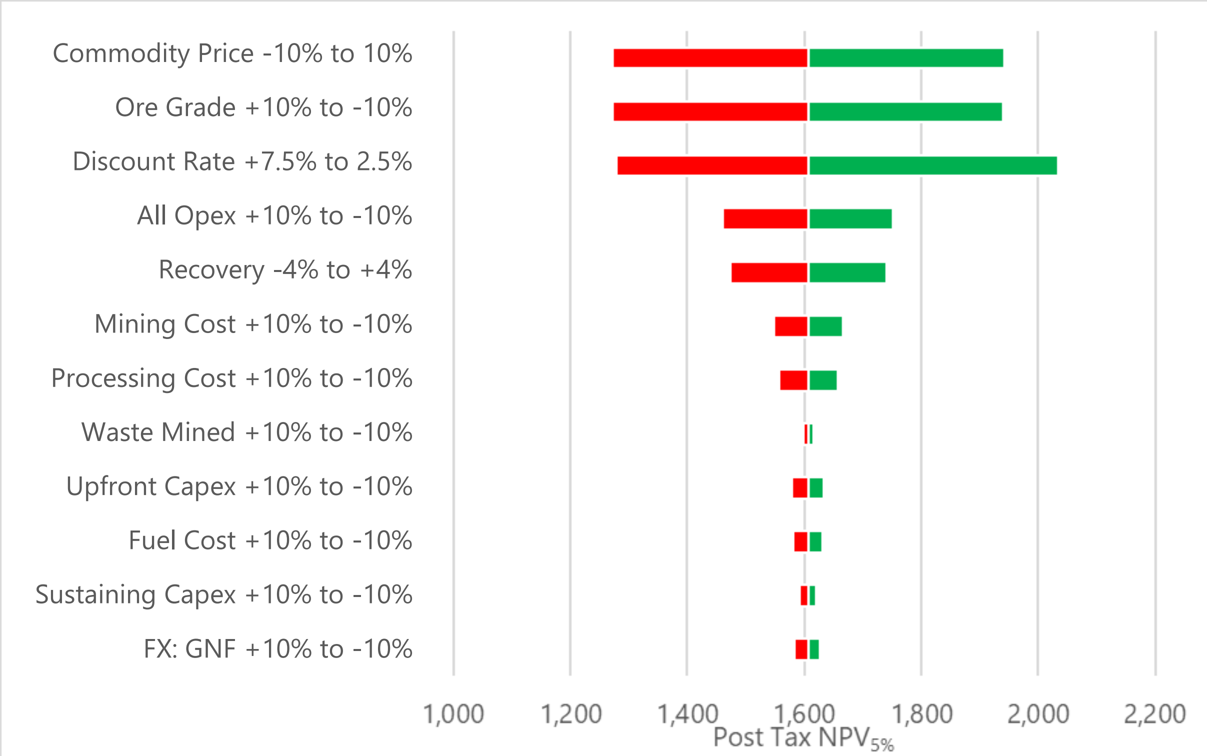

Figure 17.4: Post Tax $NPV_{5\%}$ Sensitivities

17.4 Funding Requirement and Strategy

The DFS confirms that the Project has strong financial fundamentals. The upfront capital requirement is estimated at US$463m. 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 royalites, 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$1B, 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 market capitalisation of around A$1B;
- 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.

18 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.

18.1 Risks

A summary, which is not exhaustive, of the key risks identified during the DFS, including mitigating strategies and actions, is presented in Table 18.1.

Table 18.1: 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.
Inferred Mineral Resources included in the mining inventory have a low level of geological confidence and may not convert to Ore Reserves following additional drilling and exploration impacting on the stated Project outcomes.	Inferred Mineral Resources included in the mining inventory only represent 9.5% of the contained gold and this is all from the deeper sections of the proposed underground mine only. The Inferred Mineral Resources are only included from year six of production and peak at 35% in any one year. Not all Inferred Mineral Resources included in the underground were included in the mining inventory based on an assessment of the current geological confidence around conversion (interpolated versus extrapolated grades). The Project has also been assessed based on the Ore Reserves only and found not to rely on the inclusion of Inferred Mineral Resources to still provide robust project returns.
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.

Risk	Mitigation
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.

18.2 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.
- 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 Ore 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.

19 CONCLUSIONS AND RECOMMENDATIONS

The extensive amount of Project work completed to date, including exploration, site investigations, site development, processing and other testwork, as well as associated studies leading to the completion of the DFS and an update of the Ore Reserve estimate, has demonstrated the technical and economic viability of the Project.

The DFS demonstrates strong economics, with a post-tax NPV5% of US$1,637m and an IRR of 46% at a consensus long-term gold price forecast of US$2,400/oz. The pre-production capital cost of US$463m will be paid back within 1.9 years based on post-tax cashflows. In addition, significant upside exists on the financial returns when considering the current spot gold price.

The deposits are well defined, and the mine design and equipment will suit conventional drill, blast, truck and shovel mining of the open pits and hybrid longitudinal/transverse long hole open stoping of the NEB underground deposit. The ore is free milling and amenable to conventional CIL processing to realise high recoveries (average of 92.8%), with gravity recoverable gold averaging 32%.

Environmental approval has been granted for the Project with all environmental and social risks being managed through the implementation of an extensive health and safety, environmental and social management plans implemented under an umbrella of an environment and social management system. The Project is expected to create significant benefits for local communities and Guinea more broadly through employment, the development of service businesses, and the creation of revenues from taxes, royalties, and the Local Development Contribution fund.

Based on the positive outcomes of the DFS, PDI intends to progress the Project to development and commence execution readiness activities in parallel with Project funding activities.

predictive discovery

JORC Table 1
Bankan Gold Project

24 June 2025

Section 1: Sampling Techniques and Data		
Criteria	**JORC Code Explanation**	**Commentary**
Sampling Technique	*Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as downhole gamma sondes, or handheld XRF instruments, etc.). These examples should not be taken as limiting the broad meaning of sampling.*	Samples were acquired by a mixture of aircore ("AC"), reverse circulation ("RC") and diamond drilling ("DDH"). The majority of samples are 1 m downhole, with DDH core sampling intervals breaking at lithological contacts where appropriate. Only RC and DDH drilling were used to estimate the resource.
	Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.	
	Aspects of the determination of mineralisation that are Material to the Public Report. In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.	
Drilling	*Drill type (eg core, reverse circulation, open- hole hammer, rotary air blast, auger, Bangka, sonic, etc.) and details (eg core diameter, triple or standard tube, depth of diamond tails, face- sampling bit or other type, whether core is oriented and if so, by what method, etc.).*	The NEB and BC estimates include assays received up to 29 July 2023 and are based on 205 DDH holes for 73,043 m, 62 RC/DDH holes for 25,711 m and 162 RC holes for 49,521 m, for a total 429 holes for 148,275 m of drilling. This includes the results of the close spaced grade control RC drilling completed in early 2022. For Fouwagbe, 10 DD drillholes for 2,599.5m and 42 RC drillholes for 5,434m were drilled; and for Sounsoun 32 DD drillholes for 7,116m and 76 RC drillholes for 10,032m. Core is orientated by a downhole orientation tool. Core diameters used are mostly NQ with minor HQ and HQ triple tube; 140 mm RC face sampling bits were used; and 90 mm aircore.
Drill Sample Recovery	*Method of recording and assessing core and chip sample recoveries and results assessed.*	Core recoveries were recorded by dividing the total length of core returned from each run by the length of the run. NEB and BC core recoveries averaged 92%, with the poorest recoveries (averaging 82%) in the first 40 m of the drillholes. Overall core recoveries for Fouwagbe average 84%, however the poorest recoveries are between 50 and 100m downhole, which averaged 73%; this is due to the deeply weathered and soft profile at Fouwagbe. For Sounsoun recoveries were excellent, averaging 98%, with only slightly poorer recoveries near surface. Overall RC recovery appears good, however the total recovery is not measured. The rig cyclones are regularly cleaned (several times during drilling and between drilling) in order to minimise sample accumulation and contamination, and to increase the recovery rate.

Section 1: Sampling Techniques and Data		
Criteria	**JORC Code Explanation**	**Commentary**
	Measures taken to maximise sample recovery and ensure representative nature of the samples.	The splitters are regularly checked to ensure sample build up is minimised.
	Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	No relationship between sample recovery and grade has been analysed.
Logging	*Whether core and chip samples have been geologically and geotechnical logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.*	All drill samples were logged systematically for lithology, weathering, alteration, veining, structure and minor minerals. Minor minerals were estimated quantitively. The Competent Person considers that the availability of qualitative and quantitative logging has appropriately informed the geological modelling, including weathering and oxidation, water table level and rock type.
	Whether logging is qualitative or quantitative in nature. Core (or costean/Trench, channel, etc.) photography.	The Competent Person considers that the availability of qualitative and quantitative logging has appropriately informed the geological modelling, including weathering and oxidation, water table level and rock type.
	The total length and percentage of the relevant intersections logged.	All drillhole intervals have been logged.
Sub-Sampling Technique and Sample Preparation	*If core, whether cut or sawn and whether quarter, half or all core taken.*	DDH core was cut with a diamond saw. Routine samples were half-core, with predetermined diamond core duplicates being quarter-core.
	If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.	RC and AC drillholes were split using a cone sampler. The majority of chip samples are dry or only slightly damp.
	For all sample types, the nature, quality and appropriateness of the sample preparation technique.	The Competent Person considers these methods appropriate for this style of mineralisation.
	Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.	For RC and AC samples, sample weights are recorded as are the weights of the rejects.
	Measures taken to ensure that the sampling is representative of the in-situ material collected, including for instance results for field duplicate/second-half sampling.	Field duplicate results for RC and DDH core samples demonstrated no bias in the sample results. There is a moderate scatter in the RC duplicate pairs and considerable scatter in the DDH duplicate pairs suggesting 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.
	Whether sample sizes are appropriate to the grain size of the material being sampled.	Sample sizes are considered to be appropriate to the grain size of the material being sampled.

Section 1: Sampling Techniques and Data		
Criteria	**JORC Code Explanation**	**Commentary**
Quality of Assay Data and Laboratory Tests	*The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.*	Samples were assayed using industry standard fire assaying with a 50 g charge; this method is a total method that should recover all gold in a sample. Several commercial laboratories have been used, including SGS in Bamako, SGS in Ouagadougou, MSA in Yamoussoukro and BVI in Conakry. All use slightly different procedures, but typically the sample is dried, crushed to -2 mm, split to 200 g and pulverised to -75 microns, before a 50 g aliquot is taken for assay.
	For geophysical tools, spectrometers, handheld XRF instruments, etc., the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.	Not applicable.
	Nature of quality control procedures adopted (e.g. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (i.e. lack of bias) and precision have been established.	Predictive Discovery Limited ("PDI") inserts routine blanks, certified reference materials and field duplicates into the sample stream submitted to the laboratories. The field duplicates are either second splits of chips (RC and AC) or quarter core (DDH) duplicates. The laboratories also insert their own CRMs and perform duplicate assays. Analysis of this QAQC data demonstrated that the data is of acceptable quality to be used for resource estimation.
Verification of Sampling and Assaying	*The verification of significant intersections by either independent or alternative company personnel.*	At this stage, the intersections have not been verified independently.
	The use of twinned holes.	No twinned holes have been drilled at NEB, BC or Fouwagbe. One pair of twin holes was drilled at Sounsoun: DD hole RBNDD0001 and RC hole RBNRC0059. The mineralised intersections (>0.15g/t) were 9.8m @ 0.45g/t and 6m @ 0.35g/t respectively, at a distance of approximately 8m apart. The similar tenor but differing widths may be a reflection of short-scale structural complexity that cannot be resolved at the current drill spacing.
	The verification of significant intersections by either independent or alternative company personnel.	Drillhole logging is completed on paper sheets and manually entered into a database on site. The data is managed by a company employee, who checks for data validation. Assay results are returned electronically from the assay laboratory and are merged into the assay table of the database.
	Discuss any adjustment to assay data.	No adjustments or calibrations have been made to any assay data.
Location of Data points	*Accuracy and quality of surveys used to locate drill holes (collar and down- hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.*	Collar surveying is by contracted surveyors using 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; the Champ Gyro or the Reflex EZ Shot depending on the contractor.
	Specification of the grid system used.	All surveying is completed on the WGS84 grid.

Section 1: Sampling Techniques and Data		
Criteria	**JORC Code Explanation**	**Commentary**
	Quality and adequacy of topographic control.	The topography of the project area was acquired in June 2022, comprising 3m gridded satellite topographic data. Collar pickups by DGPS are typically within 3m of this surface. The Competent Person considers that the surface is suitable for this Mineral Resource estimate.
Data Spacing and Distribution	*Data spacing for reporting of Exploration Results.*	NEB and BC have been drilled on variable grids, with 40m by 80m in the core Indicated area widening out to more than 100m in the Inferred areas. Fouwagbe has been drilled on approximately 50m by 50m spacing. Sounsoun has been drilled on two separate grids. The initial holes were drilled to 135° (to the SE) on an 100m by 50m grid, infilled to 50m by 50m 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 50m by 50m.
	Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.	The Competent Person believes the mineralised zones have sufficient geological and grade continuity to support the classification applied to the Mineral Resources given the current drill pattern.
	Whether sample compositing has been applied.	Drillholes were composited to 3 m downhole for saprolite and fresh mineralisation, and 1 m downhole for the laterite domain.
Orientation of Data in Relation to Geological Structure	*Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.*	The mineralisation at NEB and BC is mainly controlled by west-dipping shears, with a steep high grade plunge control to the west-southwest. The majority of the drilling is drilled at 55° to the east and intersects these shear and plunge directions at a high angle. The relatively consistent angle of intersection between the mineralisation and the drilling leads to an unbiased sampling. Earlier NEB holes drilled at lower angles to the mineralisation direction have not been used in the estimation of Mineral Resources. Drilling at Fouwagbe is orientated towards 135° (to the SE) on NW-SE lines. Three NW-dipping trend surfaces have been interpreted and the drilling intersects these at near right angles. These may be part of a network of shear zones. An interpretation of drillhole logging at Sounsoun has defined a E-W trending shear zone in the north of the target area. Within the E-W structure, the higher grade mineralisation is hosted in shoots which plunge steeply to the north. The drilling orientated to the south intersects these at a high angle. The drilling orientated to the SE intersects these structures at a low angle. All data has been used, as there is insufficient data to determine any bias between the two drilling orientations. The current resource classification of Inferred conforms with this use of multiple drillhole orientations.

Section 1: Sampling Techniques and Data		
Criteria	**JORC Code Explanation**	**Commentary**
	If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	At NEB, programs were initially oriented to the west; when it was recognised that the mineralisation dips west, the drilling was switched to east drilling and most areas were re-drilled. An analysis of the data from east and west dipping holes showed: • The 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; • The west dipping holes in the saprolite have a larger population > 2g/t Au; • The mean and median of the west dipping holes are lower than east dipping in the fresh; • In the saprolite, the composites in the west dipping holes are less variable. The west dipping data was filtered from the composite dataset before further processing, except for the laterite domain.
Sample Security	*The measures taken to ensure sample security.*	Samples are stored onsite with a 24-hour security presence. 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. Coarse rejects and pulps will eventually be recovered from SGS and stored at PDI's office in Kouroussa or at the core shed.
Audits or Reviews	*The results of any audits or reviews of sampling techniques and data.*	ERM has reviewed the sampling techniques and chain of custody procedures at the Project during the site visits. The Competent Person considers the techniques suitably designed and implemented.

Section 2 Reporting of Exploration Results		
Criteria	**JORC Code Explanation**	**Commentary**
Mineral Tenement and Land Tenure Status	*Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.*	The Bankan Gold Project consists of four *Permis de Recherce Industrielle (Or)* (exploration permits) as follows: Permit Name / Area (km^2) / Holder: Kaninko / 98.2158 / Mamou Resources SARLU Saman / 99.74845 / Mamou Resources SARLU Bokoro / 99.9785 / Kindia Resources SARLU Argo / 57.5422 / Argo Mining SARLU The permits are located between 9 51'00"W and 10 03 24W and between 10 32'26"N and 10'52"00N, situated to the northwest, west and southwest of the town of Kouroussa in Guinea. The main Project area, and all the Mineral Resources on which the Definitive Feasibility Study (DFS) is based, are situated on parts of the Kaninko and Saman exploration permits which are both held by Mamou Resources SARLU (Mamou), a 100% owned subsidiary of PDI. On 31 January 2025, PDI and Mamou submitted exploitation permit applications for 50% of the Kaninko and Saman permits areas to the Ministry of Mines and Geology (MMG) and the Centre de Promotion et de Développement Minier (CPDM) in accordance with Guinean mining law. PDI has indicated that the applications are at an advanced stage and are still being processed and is not aware of any immediate obstacles to the granting of the exploitation permits. The remaining portions of the Kaninko and Saman permit areas will be relinquished upon grant of the exploitation permits.

Section 2 Reporting of Exploration Results		
Criteria	**JORC Code Explanation**	**Commentary**
	The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.	Parts of the Kaninko and Saman permits, including the NEB and BC deposits, which are the subject of the DFS, are situated in the Peripheral Zone of the Upper Niger National Park. The NEB and BC deposits are 21 km and 18 km, respectively, away from the closest point of the Core Conservation Area. The Argo and Bokoro permits are situated entirely outside the Peripheral Zone of the Upper Niger National Park. There are overlapping regulations and decrees governing mining activities in natural protected areas in Guinea, including the Upper Niger National Park's management plan, and, as a result, the framework and conditions for the development of the Project are being developed in collaboration with the Ministère de l'Environnement et du Développement Durable (MEDD), park authorities, conservation groups and other stakeholders, and ultimately will be set out in the mining convention to be entered into in connection with the Project. PDI has taken a robust approach to address the sensitivities associated with the location of the Project within the Peripheral Zone of the Upper Niger National Park and has completed an Environmental & Social Impact Assessment (ESIA) and an Environmental & Social Management Plan framework with the support of ERM. In January 2025, MEDD approved the ESIA and issued the Certificate of Environmental Compliance (ECC) for the Project, confirming MEDD's support for development of the Project in the Peripheral Zone. PDI has also committed to: • Relinquishing the portion of the Kaninko permit that overlaps the Buffer Zone of the Upper Niger National Park (contains no resources, exploration targets or proposed infrastructure); • Reforestation of the area within the Bankan exploitation permit (which is under application) along the boundary with the Buffer Zone during the development phase of the Project. The Argo permit is subject to a joint venture, whereby PDI has earned a 90% interest and can acquire the remaining 10% at a decision to mine in exchange for a 2% net smelter royalty on production. 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.

Section 2 Reporting of Exploration Results		
Criteria	**JORC Code Explanation**	**Commentary**
Exploration Done by Other Parties	*Acknowledgment and appraisal of exploration by other parties.*	No previous significant modern exploration has been performed in the NEB and BC area. Previous exploration work has been completed in the Argo area by Cassidy Gold, including soil sampling, AC and RC drilling. Artisanal miners have extracted an unknown quantity of gold from shallow hand dug pits and shafts, with panning and loaming used to identify mineralisation areas.
Geology	*Deposit type, geological setting and style of mineralisation.*	The Bankan deposits are hosted in Paleoproterozoic rocks of the Birimian Supergroup in the Siguiri Basin, which is host to several significant large active gold mining operations. Mineralisation consists of wide zones of structurally controlled chlorite, silica and sericite alteration with associated pyrite and quartz veining, emplaced during deformation of anastomosing north-south shears on the hangingwall of a tonalitic felsic intrusive, which has intruded a mafic and sedimentary greenstone sequence. NEB mineralisation is found largely in a corridor between two moderately west dipping shears (the Main and Eastern Shears) with shallower dipping linking structures. The mineralisation is preferentially developed at the Main Shear, especially around the contact between the footwall tonalite and the overlying mafic/metasediment package. Higher grades are found in a steeply SW plunging shoot; a second high grade shoot down plunge of the main High Grade has been identified by three drillholes and is the target of current extensional drilling. North of a NE/SW striking wrench fault, the Gbengbeden mineralisation is similar to NEB, and is controlled by three anastomosing shears BC mineralisation is controlled by moderately west-dipping shears in a tonalite/skarn package with mafic hangingwall. Preliminary analysis suggests that the higher grade mineralisation plunges steeply to the SW, similar to NEB. Weathering has formed a deep saprolite profile, with a pisolitic and nodular lateritic cover which hosts remobilised gold, generally above the primary deposits or dispersed a few tens of metres laterally. Fouwagbe: The mineralisation has broadly developed along a main deformation corridor dipping ~50° to the NW, 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 characterised by brecciated, foliated quartz veining with traces of sulphides. The Fouwagbe area is very deeply weathered, and structural measurements and interpretation are difficult. At greater depth, beneath the oxidised felsic formations, a formation of mafic volcanic rocks was encountered, in which the main sulphide-rich formation zone is hosted. Sounsoun: The main mineralised trend consists of a E-W sheared zone dipping 70° to the north, developed 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. The mineralisation seems to be preferentially developed along this E-W shear zone.

Section 2 Reporting of Exploration Results		
Criteria	**JORC Code Explanation**	**Commentary**
Drill Hole Information	*A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:* • *easting and northing of the drill hole collar* • *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar* • *dip and azimuth of the hole* • *down hole length and interception depth* • *hole length.*	Exploration Results are not being reported.
	If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.	Exploration Results are not being reported.
Data Aggregation Methods	*In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated.*	Exploration Results are not being reported.
	Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.	Exploration Results are not being reported.
	The assumptions used for any reporting of metal equivalent values should be clearly stated.	Exploration Results are not being reported.
Relationship Between Mineralisation Widths and Intercept Lengths	*These relationships are particularly important in the reporting of Exploration Results.*	Exploration Results are not being reported.
	If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.	Exploration Results are not being reported.
	If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').	Exploration Results are not being reported.
Diagrams	*Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	Exploration Results are not being reported.

Section 2 Reporting of Exploration Results		
Criteria	**JORC Code Explanation**	**Commentary**
Balanced Reporting	*Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	Exploration Results are not being reported.
Other Substantive Exploration Data	*Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results;* *bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	Not applicable.
Further Work	*The nature and scale of planned further work (eg tests for lateral extensions or large scale step out drilling.*	Additional drilling will be undertaken in the future with the aim of increasing and upgrading the classification of the Mineral Resources at all deposits.
	Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.	Not applicable.

Section 3 Estimation and Reporting of Mineral Resources		
Criteria	**JORC Code Explanation**	**Commentary**
Database Integrity	Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.	Data is manually entered on site into Excel spreadsheet files, using a standardised format. Original forms are archived on site for reference.
	Data validation procedures used.	PDI employs a database administrator who performs standard database validation checks including incorrect XYZ locations, missing surveys, missing logging, missing assays and data out of range. The Competent Person checked the drillhole files for errors prior to Mineral Resource estimation. The Competent Person found no material errors and deemed the database was fit for the purpose of Mineral Resource estimation.
Site Visits	Comment on any site visits undertaken by the Competent Person and the outcome of those visits.	The Competent Person visited the site from 10th to 15th June 2022, from the 10th to the 21st November 2022, from the 11th to the 27th January 2023, and from 28th August to 5th September 2024. During these visits the following were inspected: • The general site layout, including the NEB and BC deposits, the Fouwagbe and Sounsoun deposits, Bankan village and surrounding areas; • Diamond core drilling; • Drillhole setup;

Section 3 Estimation and Reporting of Mineral Resources		
Criteria	**JORC Code Explanation**	**Commentary**
		• Core orientation and markup; • Core logging; • Core sampling; • Density measurement procedure; • PLT measurement procedure; • XRF measurement procedure; • 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.
	If no site visits have been undertaken, indicate why this is the case.	Not applicable.
Geological Interpretation	Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.	All drillholes have been geologically logged for weathering and lithology. A standardisation and relogging program in April 2021 ensured consistency of logging and allowed lithologies to be simplified into a few main types. An inspection of historic logging, core photos and core resulted in the identification of numerous intersections of the footwall shears, as well as hangingwall lamprophyre dykes; these were added to the appropriate database fields and used for geological modelling.
	Nature of the data used and of any assumptions made.	No material assumptions have been made which affect the Mineral Resource reported herein.
	The effect, if any, of alternative interpretations on Mineral Resource estimation.	The Competent Person is confident any alternative interpretations would result in globally immaterial differences in the Mineral Resource estimate.
	The use of geology in guiding and controlling Mineral Resource estimation. The factors affecting continuity both of grade and geology.	The interpreted anastomosing shear systems for all four deposits have been used as a primary control in the interpretation of the mineralised domains, and as an anisotropy for the Leapfrog grade shells. In addition, the interpreted base of the laterite cover has been used as an upper constraint of the in situ mineralisation, and as the base of lateritic mineralisation where present. The NEB High Grade domain is located at and in the immediate footwall of the Main Shear. BC mineralisation is controlled by shears in a tonalite/skarn package with mafic hanging wall. Preliminary analysis suggests that the higher-grade mineralisation plunges steeply to the SW, similar to NEB.

Section 3 Estimation and Reporting of Mineral Resources		
Criteria	**JORC Code Explanation**	**Commentary**
		For Fouwagbe, interpreted mineralised trends were used to produce mineralised shells; a SW plunge of 30° was interpreted from the continuity of the mineralisation and the disposition of the artisanal workings on the surface. 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.
Dimensions	The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.	NEB resource covers a strike length of approximately 1,500 m, and has been estimated to approximately 1,100 m below the natural surface. The plan width varies from 50 m to more than 220 m wide. The laterite mineralisation is near the natural surface, with saprolite mineralisation directly below the base of the laterite. BC covers approximately 650 m long in strike and to approximately 350 m below the natural surface, with a width of the Low Grade domain of up to 240 m. Fouwagbe mineralisation covers a strike length of approximately 400m, and has been estimated to approximately 300m below the natural surface. The plan width varies from 5m to more than 40m wide. The laterite mineralisation is near the natural surface, with saprolite mineralisation directly below the base of the laterite. The Mineral Resource is constrained by an optimised open pit; this does not capture the entire mineralisation but is approximately 220m deep. Sounsoun mineralisation covers a strike length of approximately 550m, and has been estimated to approximately 250m below the natural surface. The plan width varies from 5m to more than 25m wide. The Mineral Resource is constrained by an optimised open pit; this does not capture the entire mineralisation but is approximately 150m deep.
Estimation and Modelling Techniques	*The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen, include a description of computer software and parameters used.*	Gold grades have been estimated using Ordinary Kriging using Surpac software. 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 Au (Northern) 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 1m downhole composites. 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. For the laterite mineralisation, a 0.5 g/t Au cut-off domain was defined from 1m downhole composites. These domains were used as hard boundaries. High Grade composites were cut to 40 g/t, Medium Grade and Laterite to 30 g/t. The Northern and Low Grade domains were uncut. Search ellipses and kriging parameters were chosen following Kriging Neighbourhood Analysis.

Section 3 Estimation and Reporting of Mineral Resources		
Criteria	**JORC Code Explanation**	**Commentary**
		For Fouwagbe and Sounsoun, 1m downhole composites of DD and RC assays were extracted from the database and constrained to within the wireframes. High grades were cut to 40g/t (Fouwagbe) and 30g/t (Sounsoun); these top cuts were chosen after inspection of the raw statistics and the distributions; the topcuts reduce the variance without reducing the raw mean by more than 5%. For both deposits, the maximum search distance of 200m, and minimum of 8 and maximum of 24 composites, was chosen to fill all of the domains in a single estimation pass.
	The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.	The previous resource estimate for the NEB and BC deposits was dated 21 January 2023 and totalled Indicated + Inferred 76.8 Mt @ 1.69 g/t Au for 4.1M oz.; the current model total Indicated plus Inferred is 100. 5Mt @ 1.66 g/t Au for 5.4 Moz. The changes are • In the NEB open pit, the completion of the infill drilling programme has upgraded the majority of the Inferred resource to Indicated; • The revised NEB structural and mineralisation model has produced additional Inferred resources in the footwall to STMZ that has been captured by the resource open pit optimisation • Further extensional drilling at depth has increased the underground resource; the revised structural interpretation has also identified two new resource zones that added incremental resources. • At Gbengbeden, additional resources have been produced by extensional and infill drilling • At BC, a relogging programme in early 2023 lead to a new geological model. In conjunction with the additional infill and extensional drilling, this has increased the resource These differences are result of the greater level of data and the more detailed interpretation that has been possible with it. In particular, the infill drilling has demonstrated a greater number of internal higher and lower grade structures, as well as restricting the distance that grade shells are extended past the edge of the database. The resource estimate for the Fouwagbe and Sounsoun deposits is a maiden estimate and therefore there are not previous estimates. Previous artisanal mining production is minor in scale and not formally recorded.
	The assumptions made regarding recovery of by-products.	No by-products have been modelled or are expected.
	Estimation of deleterious elements or other non-grade variables of economic significance (e.g. sulphur for acid mine drainage characterisation).	No elements other than gold have been estimated.
	In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.	For NEB and BC, the estimation block size is 20 m Y by 10 m X by 5 m Z, approximately half the sample spacing in the best drilled parts of the deposits. The search ellipses range from

Section 3 Estimation and Reporting of Mineral Resources		
Criteria	**JORC Code Explanation**	**Commentary**
		140 m to 300 m with a minimum of 8 and a maximum of 14 to 24 composites adopted. For Fouwagbe and Sounsoun, the estimation block size is 20m Y by 10m X by 5m Z, approximately half the sample spacing in the best drilled parts of the deposits. The search ellipses range of 200m with a minimum of 8 and a maximum of 24 composites adopted.
	Any assumptions behind modelling of selective mining units.	SMU units were not modelled.
	Any assumptions about correlation between variables	No assumptions have been made regarding the correlation of variables.
	Description of how the geological interpretation was used to control the resource estimates.	The interpretation of the Main Shear, Footwall Shears and other shears were used as an anisotropy for the Leapfrog shells. The logged base of laterite was used as a limit of the data used for the Mottled Zone, Saprolite Zone, Saprock and Fresh mineralisation. The interpretation of the structures at Fouwagbe and Sounsoun were used as an anisotropy for the Leapfrog shells. The logged base of laterite was used as a limit of the data used for the Mottled Zone, Saprolite Zone, Saprock and Fresh mineralisation.
	Discussion of basis for using or not using grade cutting or capping.	For the estimate of grades, high-grade cuts were applied to composites to reduce the influence of extreme outliers. These values, determined by statistical analysis including review of coefficient of variation values, histograms, log-probability plots, and mean-variance plots. The aim of choosing topcuts was to reduce the coefficient of variability without unduly affecting the overall mean grade of the various mineralised domains.
	The process of validation, the checking process used, the comparison of model data to drillhole data, and use of reconciliation data if available.	Standard model validation was completed using numerical methods (histogram and swath plots) and validated visually in section and 3D against the input raw drillhole data, composites, and blocks.
Moisture	*Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	Tonnages have been estimated on a dry basis.
Cut-off Parameters	*The basis of the adopted cut-off grade(s) or quality parameters applied.*	The NEB open pit, Fouwagbe and Sounsoun resources are reported at a 0.5 g/t Au cutoff and BC is reported at a 0.4 g/t Au cutoff. Preliminary open pit economic assessments have suggested that for a bulk mining option the economic cutoff is likely to be in the range of 0.4-0.5 g/t Au, depending on the Au price assumed. For BC, 0.4 g/t Au represents the mineralisation continuity better than 0.5 g/t Au. The NEB underground resource is reported at a 2.0 g/t cutoff based on preliminary economic cutoff grade calculations.
Mining Factors or Assumptions	*Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is*	Open pit mining is considered as the appropriate method for most future studies, and the Competent Person believes that there are reasonable prospects for eventual economic

Section 3 Estimation and Reporting of Mineral Resources		
Criteria	**JORC Code Explanation**	**Commentary**
	always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	extraction based on the outputs of the Whittle optimisations completed. The key assumptions of the optimisations were: • Mill throughput of 4 Mtpa; • Metallurgical recovery of 94% for NEB, BC and Fouwagbe and 90% for Sounsoun; • Ore loss of 4% and dilution of 5%; • Base mining cost of US$1.92/t, incremented with depth; • Processing costs of US$19.90-$24.73/t, depending on material type; • Gold price of US$1,800/oz (NEB and BC) or $2,300/oz (Fouwagbe and Sounsoun); • Discount rate of 5%. The optimisations captured a large proportion of the mineralisation and was largely driven by the extent of the modelled High Grade domains. For NEB underground area, a bulk mining method has been assumed, and the current models are reported at a 2.0 g/t Au cutoff that greater selectivity is not achievable from the current very widely spaced data. Although the Mineral Resource has been reported with the optimal open pit optimisation, a decision has been made to optimise the overall Project by mining a smaller open pit at NEB, and plan mining some of the open pit resource from an underground operation. As the underground cutoff is higher than the open pit cutoff, some of the lower grade parts of the open pit resource will be effectively sterilised and is unlikely to meet the Reasonable Prospects test. When the final decision is made to commence the underground, the Mineral Resource should be re-stated to reflect the planned sterilisation of part of the open pit resource.
Metallurgical Factors or Assumptions	*The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	A scoping level metallurgical testwork program was carried out on eleven samples with a total weight of 305 kg from both NEB and BC, representing softer saprolite and fresh rock mineralisation. All samples were quarter NQ diamond drill core apart from one saprolite sample of reverse circulation chips. The scope of the test work program included: comminution testwork, optimisation of grind size and leaching characteristics, gravity concentration, and cyanide leaching tests. The testwork program was completed by Metallurgy Pty Ltd in Perth, Western Australia. The main results were: • The fresh ore is relatively hard, with a Bond Ball Mill Index of 18-25 kWh/t. • Optimum grind size is approximately 75 microns. • The ore has a moderate proportion of gravity-recoverable gold, ranging from 13% to 37% for the samples. • Using optimum leaching conditions, over 94% of the leach feed gold could be recovered in 24 hours, with a cyanide consumption of 0.7-0.9 kg/t and lime consumption of around 0.1kg/t. These results suggest that relatively high recoveries may be achievable using standard CIL technology.

predictive discovery

Section 3 Estimation and Reporting of Mineral Resources		
Criteria	**JORC Code Explanation**	**Commentary**
		Four composites of saprolite ore were leach tested, two from Sounsoun and two from Fouwagbe. All four had good gravity recovery on the samples, despite the low head grade of three (<1g/t). The cyanide consumptions were consistently around the 0.22-0.24kg/t, and extractions at 24 hours were between 88% and 99%.
Environmental Factors or Assumptions	*Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered, this should be reported with an explanation of the environmental assumptions made.*	No assumptions regarding possible waste and process residue disposal options have been made. Mine waste is planned to be stored in waste rock dumps adjacent to the deposits and process waste in a tailings storage facility. There are other gold mines operating in the Siguiri Basin in Guinea which utilise similar disposal techniques. PDI has taken a robust approach to address the sensitivities associated with the location of the Project within the Peripheral Zone of the Upper Niger National Park and has completed an ESIA and an Environmental & Social Management Plan framework with the support of ERM. In January 2025, MEDD approved the ESIA and issued the ECC for the Project, confirming MEDD's support for development of the Project in the Peripheral Zone.
Bulk Density	*Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.*	The density of selected core samples are measured using an immersion method. Samples of 10-30 cm of competent core are selected, every 30-50 m in waste lithologies and every 5 m in shear zones. The samples are oven dried, then weighed in air and then immersed in water and density calculated using Archimedes' Principle. A total of 9,704 measurements have been recorded. An analysis of the current density database was made, by classifying by the logged weathering and lithology. From a review of these, 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 gcm^{-3}; fresh mafic: 2.9 gcm^{-3}; fresh metasediment: 2.6 gcm^{-3}; saprock, 2.3 gcm^{-3}; saprolite and mottled zone: 1.6 gcm^{-3}; laterite: 2.2 gcm^{-3}. These are typical values for the logged rock types.
	The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc.), moisture and differences between rock and alteration zones within the deposit.	Friable, oxidised or porous samples are first wax coated, with the mass of the wax recorded and taken into account for the density calculation. Lithology and weathering type are recorded for each sample.
	Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.	Densities were applied according to the interpreted lithology and weathering state.
Classification	*The basis for the classification of the Mineral Resources into varying confidence categories.*	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

Section 3 Estimation and Reporting of Mineral Resources		
Criteria	**JORC Code Explanation**	**Commentary**
		footwall, any open pit blocks in the Low Grade domain above the cutoff, the entire underground resource, and the majority of Gbengbeden, 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 80m at the bottom of the model. The core area has been classified 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. At Fouwagbe and Sounsoun, the Mineral Resources are classified entirely as Inferred.
	Whether appropriate account has been taken of all relevant factors (i.e. relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).	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.
	Whether the result appropriately reflects the Competent Person's view of the deposit.	The Mineral Resource classifications applied appropriately reflect the view of the Competent Person.
Audits or Reviews	*The results of any audits or reviews of Mineral Resource estimates.*	Internal audits were completed by ERM which verified the technical inputs, methodology, parameters and results of the estimate.
Discussion of Relative Accuracy / Confidence	*Where appropriate, a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.*	The accuracy of the Mineral Resource is communicated through the classification assigned. The Mineral Resource been classified in accordance with the JORC Code (2012 Edition) using a qualitative approach. All factors that have been considered have been adequately communicated in Section 1 and Section 3 of this table.
	The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.	The Mineral Resource Statement relates to a global estimate of in-situ tonnes and grade. It is suitable for reporting as a resource, however the relatively wide sampling grid has produced a model with only moderately well estimated individual blocks. No reliance should be placed on individual block grade estimates.
	These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	There has been no previous commercial production from the property. Previous artisanal mining production is minor in scale and not formally recorded.

Section 4 Estimation and Reporting of Ore Reserves		
Criteria	**JORC Code explanation**	**Commentary**
Mineral Resource estimate for conversion to Ore Reserves	*Description of the Mineral Resource estimate used as a basis for the conversion to an Ore Reserve.* *Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore Reserves.*	The Open Pit and Underground Ore Reserve estimate is based on the NEB and BC Mineral Resource estimate completed in August 2023 by ERM (formerly CSA Global) and prepared by Mr. Phil Jankowski as the Competent Person. The Mineral Resource estimate was reported using a 0.5 g/t Au cut-off for the NEB open pit, 0.4 g/t Au for BC and a 2 g/t Au cut-off for NEB underground Mineral Resources. The Mineral Resource estimate for the combined NEB and BC open pits has been reported as follows: • Indicated: 83.7 Mt at 1.54 g/t Au • Inferred: 10.0 Mt at 1.03 g/t Au. The Mineral Resource estimate for the NEB underground has been reported as: • Inferred: 6.8 Mt at 4.07 g/t Au. It should be noted that the above open pit Mineral Resource 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 Ore Reserve was considerably smaller as it was based on an open pit / underground trade-off optimisation. Consequently, the portion of Open Pit Indicated Resource below this smaller pit shell was considered as part of the underground mining study. A mining study at a DFS level was carried out on the Indicated portion of the Mineral Resource, including pit optimisation, optimisation of underground stope shapes, mine design (both open pit and underground), production schedules and cost estimation and modelling. The Ore Reserve was then estimated by taking into consideration the mining, processing, metallurgical, economic, marketing, legal, environmental, social, and governmental factors. Some of the Open Pit Ore Reserve economic cut-off grades for NEB are lower than the 0.5 g/t cut-off grade used to report the NEB Open Pit Mineral Resource. However, the NEB Open Pit Ore Reserve designs are well within the RPEEE optimisation shell used to define the NEB Open Pit Mineral Resource. The BC Open Pit Mineral Resource is inclusive of the BC Open Pit Ore Reserve. The NEB Open Pit Mineral Resource is inclusive of the NEB Underground Ore Reserve.
Site visits	*Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* *If no site visits have been undertaken indicate why this is the case.*	Mr Ross Cheyne, an employee of Orelogy Mine Consulting Pty Ltd and the Competent Person for the Open Pit Ore Reserves, conducted a site visit from 25th January 2025 to 29th January 2025. During this visit, the following aspects of the Project were inspected: • Road access to site from Conakry. • The general site layout, including but not limited to: - the NEB/GBE and BC deposits. - Process plant. - Mining infrastructure. - TSF. - Magazine. • Bankan village and surrounding areas.

Section 4 Estimation and Reporting of Ore Reserves		
Criteria	**JORC Code explanation**	**Commentary**
		• Kouroussa township. • Diamond core drilling samples stored at site. Detailed technical discussions were also conducted with PDI staff. No areas of concern were identified as part of the site inspection. Mr Julian Broomfield, an employee of Orelogy Mine Consulting Pty Ltd and the Competent Person for the Underground Ore Reserves, has not visited the site and has relied on the observations of Mr Ross Cheyne.
Study Status	*The type and level of study undertaken to enable Mineral Resources to be converted to Ore Reserves.* *The Code requires that a study to at least PFS level has been undertaken to convert Mineral Resources to Ore Reserves. Such studies will have been carried out and will have determined a mine plan that is technically achievable and economically viable and that material Modifying Factors have been considered.*	The Bankan Gold Project has been completed to a minimum level of a DFS, this being considered as +/- 15%. The work undertaken for the Project has addressed all material Modifying Factors required to convert Mineral Resources to Ore Reserves. It has shown that the mine plan is technically achievable and economically viable. This Ore Reserve estimate applies all material Modifying Factors such as mining dilution, mining recovery, infrastructure, costs, legal, environmental, social and regulatory, in line with normal JORC Code standards. The Ore Reserve estimate and associated mining schedule and financial modelling are underpinned by operating mining cost data, processing costs from proven technology and plant recovery information.
Cut-off parameters	*The basis of the cut-off grade(s) or quality parameters applied.*	An economic definition of ore has been applied for this Ore Reserve estimation. **Open Pit** The Open Pit Ore Reserve is reported using variable cut-off grades as processing costs vary by material type and metallurgical recovery varies by the diluted head grade. Consequently, ore was flagged in the model on a block-by-block basis. Dilution has been applied in the model and therefore the cut-off grades quoted are for a diluted grade. Based on the average diluted grade, the average cut-off grades by material type are: • Laterite = 0.41 g/t diluted • Saprolite / Mottled = 0.38 g/t diluted • Saprock = 0.41 g/t diluted • Fresh = 0.48 g/t diluted All blocks in the pit design that do not satisfy these criteria were classified as waste material. The cut-off grade was based on a fixed long-term gold price of US$1,800/oz. **Underground** An economic cut-off grade for the underground has been applied for this Ore Reserve estimation. The cut-off calculation was based on the following equation: Cut-off Grade (g/t) = Total Cost ($/t) / (Process Recovery (%) * (Metal Price ($/g) – Other Costs ($/g)) The cut-off grade parameters include: • Fixed gold price of US$ 1,800 /oz • Government royalty of 6% • Selling cost of US$ 4.0 /oz

<table>
<tr><th colspan="3">Section 4 Estimation and Reporting of Ore Reserves</th></tr>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>

Processing recovery of 92.6%
Processing cost of US$ 22.34 /t
G&A cost of US$ 2.28 /t
Underground mining cost of US$ 74.47 /t

The underground economic cut-off grade was estimated to be 2.0 g/t, noting that all underground ore is fresh ore.
</td></tr>
<tr><td>Mining factors or assumptions</td><td>
The method and assumptions used as reported in the PFS or FS to convert the Mineral Resource to an Ore Reserve (i.e. either by application of appropriate factors by optimisation or by preliminary or detailed design).
The choice, nature and appropriateness of the selected mining method(s) and other mining parameters including associated design issues such as pre-strip, access, etc. The assumptions made regarding geotechnical parameters (e.g. pit slopes, stope sizes, etc), grade control and pre-production drilling.
The major assumptions made and Mineral Resource model used for pit and stope optimisation (if appropriate).
The mining dilution factors used.
The mining recovery factors used.
Any minimum mining widths used.
The manner in which Inferred Mineral Resources are utilised in mining studies and the sensitivity of the outcome to their inclusion.
The infrastructure requirements of the selected mining methods.
</td><td>
Open Pit
To develop the mine plan for the Bankan Gold Project, optimised pit shells were prepared using GEOVIA's Whittle™ software. Designs and schedules based on optimised pit shells were completed using the Hexagon MinePlan Schedule Optimizer (MPSO) scheduling tool.
Input parameters for the pit optimisations were based on data from PDI and external parties, such as geotechnical reports and metallurgical results.
The gold price is based on a conservative US$1,800/oz.
The operating costs have been based on a mix of first principles estimation, budget submission from suppliers and contractor and input from PDI, all to an acceptable DFS standard.
The mining method is based on three pit stages in NEB and a single pit stage each in GBE and BC. Pits are mined using conventional open cut, drill and blast, and load and haul mining methods. This is considered appropriate for this style of deposit and is the most commonly used approach in West Africa.
The resources below the NEB open pit will be mined by underground methods.
Pit slope parameters were made in accordance with the recommendations provided by geotechnical consultants Peter O'Bryan and Associates, using assessment of resource definition diamond cored drill holes and seven (7) geotechnical-specific diamond cored holes for its analysis. The following pit slope design parameters were provided as input to the mine design:
All Weatherred Zones
<table>
<tr><td>Face Height (m)</td><td>5</td></tr>
<tr><td>Face Angle (°)</td><td>60</td></tr>
<tr><td>Berm Width (m)</td><td>4.5</td></tr>
<tr><td>Inter-ramp Angle (°)</td><td>34.1</td></tr>
</table>
NEB / GBE Fresh Zone
<table>
<tr><th>Parameter</th><th>Footwall (Domain A)</th><th>Hanging wall (Domain B)</th></tr>
<tr><td>Face Height (m)</td><td>10</td><td>20</td></tr>
<tr><td>Face Angle (°)</td><td>90</td><td>75</td></tr>
<tr><td>Berm Width (m)</td><td>>4[1]</td><td>8</td></tr>
<tr><td>Inter-ramp Angle (°)</td><td>68[1]</td><td>56.3</td></tr>
</table>
Note 1: Inter-ramp angle is limited by the dip in mineralisation which is likely to result in berm widths of 12 m and an inter-ramp angle of 39.8°.
BC Fresh Zone
</td></tr>
</table>

Section 4 Estimation and Reporting of Ore Reserves		
Criteria	**JORC Code explanation**	**Commentary**
		<table><tr><th>Parameter</th><th>Footwall (Domain A)</th><th>Hanging wall (Domain B)</th></tr><tr><td>Face Height (m)</td><td>10</td><td>20</td></tr><tr><td>Face Angle (°)</td><td>90</td><td>75</td></tr><tr><td>Berm Width (m)</td><td>>4[1]</td><td>8</td></tr><tr><td>Inter-ramp Angle (°)</td><td>68[1]</td><td>56.3</td></tr></table>Note 1: Inter-ramp angle is limited by the dip in mineralisation which is likely to result in berm widths of 8.5 m and an inter-ramp angle of 49.6˚. The pit optimisations were run including the application of an underground mining cost to determine the optimum depth to transfer from open pit to underground mining methods. Pit designs were validated against an optimised pit shell generated using the final DFS parameters. There was an overall 3% reduction in ore and 2% increase in waste across all designs relative to the optimised shells. Dilution and ore loss was modelled in 3D utilising a proprietary edge dilution approach by Orelogy Mine Consulting. This resulted in a global dilution of 3.6% and ore loss of 4.4% inclusive of operational ore loss of 1%. This is reflective of the broader and more continuous nature of the NEB orebody which makes up 90% of the Open Pit Ore Reserve. Dilution for the smaller deposits (BC and GBE) is much higher, averaging 5.4% and 8.7% respectively. The minimum mining width applied in the design is 50 m and is appropriate for the selected mining equipment fleet. Revenue from open pit Inferred Mineral Resources has not been included in the pit optimisations, or in the LOM scheduling for the Ore Reserve modelling. The base of the GBE pit is utilised for the portal access to the underground operation. The underground mine requires approx. 15 months of development before ore can be sustainably supplied to the process plant. Therefore, GBE pit is required to be mined in the first three quarters of Year -2 of the Project to allow the underground timeline to be achieved. Therefore, the basis of the DFS is that GBE will be mined by the bulk earthworks contractor mobilised as part of the construction phase of the Project. **Underground** The mining method is a combination of transverse and longitudinal long hole open stoping with engineered paste fill. Mining methodology is a top-down mining approach, extracting panels of three mining levels in a bottom-up mining sequence. Selection of the mining method followed the UBC mining method selection process (Miller-Tait, 1995). Mining development and production will be conducted with conventional mechanised mining equipment, (jumbo/ long hole development/production drill and blast, load and haul with LHD and trucks) in line with industry best practices. This is considered appropriate for this style of deposit. The underground orebody will be accessed through mining a conventional decline (1 in 7 gradient) from the base of a small open pit (GBE) as opposed to a box cut with ventilation provided through a secondary/twin decline and vertical shafts.

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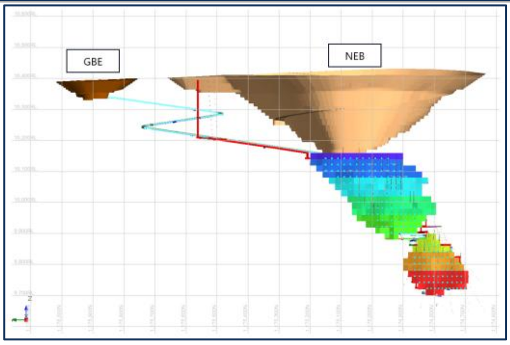
Criteria	**JORC Code explanation**	**Commentary**
		 Key geotechnical parameters provided by external consultants Peter O'Bryan & Associates and Minefill Services are shown below:

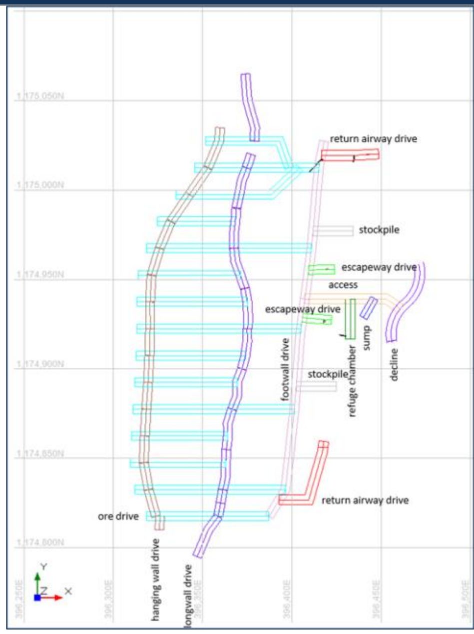
Undergound Mine Design Criteria	
Mining Option	Transverse Long Hole Open Stope with Fill (Paste)
Maximum vertical height (m) before sill pillars	60
Maximum stope length (m)	Orebody Width
Maximum span before rib pillars (m)	Due to orebody width, in-stope rib pillars not practicable/feasible
Rib pillar width (m)	n/a
Sill Pillar thickness (m)	≥ 30 m (expected level spacing)
Crown pillar thickness (m)	1:1 width to height ratio plus 25m
Suggested stope width (m)	10m
Suggested level spacing (m)	30m
Backfill strength (Kpa)	1,700 to 3,800

Level intervals of 20m were adopted to mitigate risks related to drill hole accuracy and hole deviation, stope recovery and reliable slot performance. It also better respects the orebody's geometry (40-45° dip), reducing the potential for internal dilution and maintaining hangingwall stability throughout the operation.

Level development designs include a footwall drive located 25 m from the orebody providing ore drive access on multiple fronts. A hangingwall drive is designed to support transverse stoping, by offering a free mining face and allowing installation of hangingwall cable bolts to enhance ground stability. Longitudinal stopes are situated along the footwall and at the north and south extremities, with access is provided via dedicated longitudinal ore drives. In addition to production infrastructure, each level is equipped with an escapeway drive, a sump on the access drive, return air connections at both ends of the footwall drive, and a refuge chamber cuddy, which also serves as a location for a substation where required.

Section 4 Estimation and Reporting of Ore Reserves

Criteria	JORC Code explanation	Commentary
		(continued below)



Designs and schedules based on optimised stope shapes were completed in Deswik Suite, a proprietary design and scheduling software tool.

Various mining recovery modifying factors were applied within the mine plan based on the following:

Design Type	Recovery (%)
Development	100
Stoping	85
Sill Pillar Stope	75
Crown Pillar	75

The modifying factor of planned dilution, been based on an overbreak of 0.5 metres on the hangingwall and footwall of each stope, equates to between 9% and 16% dilution, depending on stope dimensions.

No revenue from any Inferred Mineral Resource has been included in the underground optimisations or life of mine scheduling for the Ore Reserve estimation.

Underground specific infrastructure includes, (but is not limited to), twin decline, return air ventilation raises to the surface, primary and secondary ventilation fans, surface cooling plant, internal return air raise and escape way raises between levels, paste plant and associated reticulation, dewatering pump stations, footwall drives on each level, power substations, refuge chambers, compressor and mine services.

The establishment of this infrastructure is included in the capital cost estimates for the Project.

General

General mining infrastructure will include a ROM pad, tailings storage facility, topsoil and waste rock dumps, stockpiles, haul roads, workshops, processing plant, power station and offices. The establishment of this infrastructure is included in the capital cost estimate or the operating cost estimate (where assumed to be provided by a contractor as part of a rate for services or a build-own-operate arrangement) for the Project.

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Criteria	**JORC Code explanation**	**Commentary**
Metallurgical factors or assumptions	*The metallurgical process proposed and the appropriateness of that process to the style of mineralisation.* *Whether the metallurgical process is well-tested technology or novel in nature.* *The nature, amount and representativeness of metallurgical testwork undertaken, the nature of the metallurgical domaining applied and the corresponding metallurgical recovery factors applied.* *Any assumptions or allowances made for deleterious elements.* *The existence of any bulk sample or pilot-scale testwork and the degree to which such samples are considered representative of the orebody as a whole.* *For minerals that are defined by a specification, has the ore reserve estimation been based on the appropriate mineralogy to meet the specifications?*	The gold processing plant implements separate crushing circuits for weathered and fresh feed to take into account materials handling issues with weathered feed (particularly the saprolite clays). For fresh feed a single stage jaw crusher is used, stockpiling crushed ore on a stockpile. For weathered feed a single stage mineral sizer feeds directly to the grinding circuit. The grinding circuit is a SAG/Pebble crusher/Ball mill circuit (SABC) to prepare the ore for treatment in a well-tested conventional Gravity/Leach/CIL circuit. The process plant has been designed based on a throughput of 4.5 Mtpa. The process throughput ramps up from 60% of nameplate in Month 1 to 100% nameplate by Month 5 of Year 1. At steady state, mill feed split between Open Pit and Underground will be approximately 70/30. Metallurgical testwork has been conducted across three programs, from 2021 to 2025 as per below: • Program 1: Preliminary testwork managed by Mintrex (2021). • Program 2: Interim testwork managed by IMO (2024). • Program 3: Latest testwork managed by PDI/Dhamana (2024-2025). The first two scoping level metallurgical testwork program were carried out on eleven samples from both NEB and BC, representing softer saprolite and fresh rock mineralisation. All samples were quarter NQ diamond drill core apart from one saprolite sample of reverse circulation chips. These programs defined the key process design criteria, which included: • Grind size, selecting a grind size of (P80) 75µm. • Gravity gold recovery, which demonstrated that there are 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. Program 3, which was 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 to improve the definition of process performance and design envelope, and to define carbon loading and cyanide destruction performance and design parameters, and critical equipment sizing. Program 3 used available samples from the previous programs and a further 32 new samples spanning the range of lithologies and grades from NEB/GBE and BC. All samples were quarter NQ diamond drill core.

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Criteria	**JORC Code explanation**	**Commentary**
		Key design and operating parameters that have been finalised from all testwork to date include: • Materials handling testwork 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. • Fresh ore and fresh/laterite blends are both suitable for jaw crushing and stockpile reclamation. • 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. • Cyanide concentration to target leach residual of 150 to 200 mg/L of NaCN. 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. • For blends containing less than 75% saprolite, thickener underflows can realistically achieve greater than 52% solids (w/w), and rheology 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. • Cyanide consumption is influenced by the presence of cyanide soluble copper), 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 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 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, 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. The median copper in solution is 27 mg/L.

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Criteria	**JORC Code explanation**	**Commentary**
		The effect of the recovery calculation is relatively small with the higher-grade underground ore generating an average 93.5% recovery against the open pit at 92.3%.
Environmental	*The status of studies of potential environmental impacts of the mining and processing operation. Details of waste rock characterisation and the consideration of potential sites, status of design options considered and, where applicable, the status of approvals for process residue storage and waste dumps should be reported.*	PDI completed an ESIA for the Project in April 2024. The MEDD approved the ESIA and issued the Certificate of Environmental Compliance (CCE/00070) for the Project on 17 January 2025. The Project area lies within the Peripheral Zone of the Upper Niger National Park. Over the years, the area in which the Project is to be located has been extensively deforested for agriculture, grazing and charcoal burning, as well as for artisanal mining, degrading the area's biodiversity, water courses, soils and habitats. Key aspects the ESIA addressed include the impact of mining activities on critical habitats and some endangered species. A critical habitat assessment has been completed, and the triggering features identified in the Project area of influence consist of two habitat types (the bowal and gallery forest) and nine species: western chimpanzee (critically endangered), hooded vulture (critically endangered), three fish species; three plants; and a reptile. Design and control measures for biodiversity have been factored into the mine plan to avoid sensitive biodiversity features in the Project area. Mitigation will be managed through the Biodiversity Action Plan to identify and protect set-asides, biological corridors, restoration of habitats, and biodiversity offsets. Other impacts identified are on air quality and noise, where the nearest sensitive receptor is Bankan village. Mitigating these impacts will require good international industry practice measures to control dust and minimise combustion emissions, construction activities and traffic near Bankan. Geochemical test work to assess the propensity for acid metalliferous drainage and metal leaching (AMD/ML) has been completed on the saprolite and bedrock that will be excavated. The sulphur content in most of the samples is low, with the majority of samples falling within the non-acid formation (NAF) category. Leachate samples, combined with mineralogy and drill hole database metals data, indicate that most of the major lithologies tested are non-acid forming or acid consuming. On this basis, only high-level controls are required to manage the risk of acid generation in the waste rock. Proposed Waste Rock Dumps (WRD) have been placed and designed to minimise impact and incorporate boundary sediment traps and sumps to collect surface water runoff. WRD heights have been restricted to not exceed the surrounding regional topography and be geotechnically stable. A filtered TSF has been adopted due to the Project's location within the Peripheral Zone of the Upper Niger National Park and proximity to the regional centre of Kouroussa. The filtered TSF will be fully lined with HDPE, and as such, has a low risk of any leachate from the TSF entering groundwater. A conceptual closure plan has been developed as part of the ESIA, aligned with ICMM guidance and is reflected in the Project cost model. A total closure cost of US$36.5M has been calculated, inclusive of a US$3.2M allowance for post-closure monitoring. AGE Consultants conducted a 1:100-year flood event on surface water across the Project Site, including the Niger River, and prepared a Stormwater Management Plan.

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Criteria	**JORC Code explanation**	**Commentary**
		It is important to note that the Project final execution design aspects may differ from those presented in the ESIA or the DFS. Where these changes are material to environmental and social risks and impacts, management of the change process shall be applied to re-assess these and/or conduct additional assessments (as required by international standards and Guinean law and regulations).
Infrastructure	*The existence of appropriate infrastructure: availability of land for plant development, power, water, transportation* *(particularly for bulk commodities), labour, accommodation; or the ease with which the infrastructure can be provided or accessed.*	Other than some local roads and an exploration camp, there is no existing infrastructure in the Project area. 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. 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. 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 site 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 camp size will be sufficient for the workforce requirements and will be expanded with temporary construction facilities for construction.
Costs	*The derivation of, or assumptions made, regarding projected capital costs in the study.* *The methodology used to estimate operating costs.* *Allowances made for the content of deleterious elements.* *The derivation of assumptions made of metal or commodity price(s), for the principal minerals and co- products.* *The source of exchange rates used in the study.* *Derivation of transportation charges.*	All costs used in the study were based on US dollars, and the conversion of amounts into US dollars, where required, was based on defined exchange rates which approximated spot rates at the time. Capital and operating cost estimates are of sufficient accuracy for a DFS confidence level. The capital cost estimate is a bottom-up estimate generated from designs of sufficient detail to support a Class 3 estimate, market information and a mining contractor quote for both open pit and underground mining. A small percentage of costs were priced on industry norms and typical estimating factors. The operating cost estimate is a bottom-up estimate, incorporating a mining contractor quote for both open pit and underground mining. All significant and measurable items are

Section 4 Estimation and Reporting of Ore Reserves		
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	The basis for forecasting or source of treatment and refining charges, penalties for failure to meet specification, etc. *The allowances made for royalties payable, both Government and private.*	itemised with smaller items factored in as per industry practice. There are no allowances for deleterious elements in product as none have been identified as part of the DFS. Transportation, including sea freight and charges from the port to the site, have been included as separate mobilisation costs and incorporated into the capital and operating cost estimate. For goods sourced in Guinea only transportation costs from the supply source to the Project area have been included. The treatment and refining charges are based on pricing from a global refining company. The gold metal produced is not sold under a specification. The royalty included is based on 5% of the revenue plus a Local Development Contribution of 1% of revenue.
Revenue factors	*The derivation of, or assumptions made regarding revenue factors including head grade, metal or commodity price(s) exchange rates, transportation and treatment charges, penalties, net smelter returns, etc.* *The derivation of assumptions made of metal or commodity price(s), for the principal metals, minerals and co-products.*	The revenue calculations have been based on detailed mine designs, mining schedules and modifying factors. The treatment and refining charges used was US$ 4.00/oz. The gold price used for estimation of the Ore Reserve is a fixed US$ 1,800/oz. This was reviewed against a market consensus long-term forecast from approximately 30 institutions of US$2,400/oz and is considered conservative.
Market assessment	*The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to affect supply and demand into the future.* *A customer and competitor analysis along with the identification of likely market windows for the product.* *Price and volume forecasts and the basis for these forecasts.* *For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.*	A long-term consensus forecast from approximately 30 institutions was reviewed. Market commentators continue to forecast strong gold prices. There is a transparent and liquid market for the sale of gold. It is assumed that gold doré will be air freighted and sold to a global refinery. Price is expected to be based on the LBMA gold price on the day of completion of refining.
Economic	*The inputs to the economic analysis to produce the NPV in the study, the source and confidence of these economic inputs including estimated inflation, discount rate, etc.* *NPV ranges and sensitivity to variations in the significant assumptions and inputs.*	The economic analysis is based on the capital cost and operating costs as driven by the combined production schedule including only the Ore Reserves from the open pit and the underground operations to generate a cash flow. The cash flow forecasts include: • Initial, sustaining and closure capital estimates. • Operating cost estimates for mining, processing, general & administration. • Selling costs for transport and refining. • Royalty costs, and tax Revenue estimates are based on gold dore produced using the estimated metallurgical recoveries. A 5% discount rate has been applied in the financial model. This economic analysis is based on the finalised study parameters, There are minor differences between these inputs and those used to develop the cut-off grade for the Ore Reserve but the impact is immaterial. Review of the results of the economic modelling by the Competent Persons showed that removing the Inferred

Section 4 Estimation and Reporting of Ore Reserves		
Criteria	**JORC Code explanation**	**Commentary**
		Mineral Resources included in the mining inventory considered in the DFS reduced the post-tax IRR by less than 1%. Additionally, when considering the long-term gold price of US$1,800/oz and only revenue from Ore Reserves, the Project provides substantial returns, with a post-tax IRR in excess of 25%, which demonstrates the economic robustness of the Ore Reserves. Further evaluation by the Competent Persons on the sensitivity of the economic returns from the Ore Reserves to various factors such as mined grade, mined tonnes, metal recovery, mining/processing/G&A costs, capital/sustaining/closure costs and discount rate demonstrated sufficient returns in all instances to justify the Ore Reserves estimate. Further details of assumptions used in the economic analysis can be found in Chapter 17 of the DFS Summary Report.
Social	*The status of agreements with key stakeholders and matters leading to social licence to operate.*	There were no agreements in place at the time of releasing the Ore Reserve. The Project's social area of influence is mainly over a low-density rural area with three small villages (Bankan, Kignédouba, and Sokoro), extending to Kouroussa to the east, the closest large town, with a population of around 50,000. The rural areas are impoverished, with low levels of education and poor access to electricity, services and healthcare. Artisanal gold mining and agriculture are the main occupations. The Project is expected to generate positive social impacts at a local, regional and national level through the generation of direct and indirect jobs during the different project phases, the creation of long-term benefits associated with capacity enhancement of local Guinean labour force through on-the-job and formal training, increased spending capacity, a growing market to cover supply chain needs, and tax and royalty payments. These positive social impacts are expected to be long-term in nature and of moderate to major significance. Negative social impacts have been identified from land acquisition and access restrictions that will not only affect agricultural land, land used for grazing and access to ecosystem services but will also affect all the artisanal mining sites. PDI has established a Resettlement and Compensation Policy Framework (RCPF) which will serve as the basis for the Resettlement Action Plan and a Livelihood Restoration Framework which is being developed to help mitigate these impacts, along with stakeholder engagement to help define the post-mining land use. Impacts on community cohesion and health have been identified from the in-migration of people. PDI is developing and will implement plans and procedures to manage livelihood restoration, stakeholder engagement, community health and other potential impacts to manage and mitigate social risks. Stakeholder consultation will also continue through the Project construction and operations phases. It is important to note that the Project final execution design aspects may differ from those presented in the ESIA or the DFS. Where these changes are material to environmental and social risks and impacts, management of the change process shall be applied to re-assess these and/or conduct additional assessments (as required by international standards and Guinean law and regulations).

Section 4 Estimation and Reporting of Ore Reserves		
Criteria	**JORC Code explanation**	**Commentary**
Other	*To the extent relevant, the impact of the following on the Project and/or on the estimation and classification of the Ore Reserves:* *Any identified material naturally occurring risks.* *The status of material legal agreements and marketing arrangements.* *The status of governmental agreements and approvals critical to the viability of the Project, such as mineral tenement status and government and statutory approvals. There must be reasonable grounds to expect that all necessary Government approvals will be received within the timeframes anticipated in the PFS or FS. Highlight and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.*	PDI has successfully secured approval of the Project ESIA and been issued with a Certificate of Environmental Compliance (CCE/00070). This is pivotal for obtaining several required authorisations from pertinent authorities, such as the MEDD (in accordance with the Environmental Code) and the Ministry of Forests. PDI has developed a strategy and approach to secure the relevant exploitation title and mining convention. The exploitation permit application was lodged in January 2025 and significant progress has been made towards its grant. PDI does not foresee any impediments to securing an exploitation permit and agreeing a mining convention. PDI believes the exploitation permit will be granted imminently allowing development to commence. Development does not require the agreement of a mining convention. To the best of Orelogy's knowledge, there are reasonable grounds to expect that all necessary Government approvals will be received within the timeframes anticipated. It is important to note that design aspects finalised in the future detailed design may differ from those presented in the ESIA. Where these changes are material to environmental and social risks and impacts, management of the change process shall be applied to re-assess these and/or conduct additional assessments (as required by international standards and Guinean law and regulations).
Classification	*The basis for the classification of the Ore Reserves into varying confidence categories.* *Whether the result appropriately reflects the Competent Person's view of the deposit.* *The proportion of Probable Ore Reserves that have been derived from Measured Mineral Resources (if any).*	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. The classification reflects the overall level of confidence in mineralised domain continuity based on the mineralisation drill sample data numbers, spacing and orientation. At NEB, 3.90 Moz or 98% of the Open Pit Mineral Resource is classified as Indicated. The Open Pit Inferred comprises some separate zones in the footwall and any blocks in the open pit Low Grade domain above the cut-off. Approximately 50% of the BC Mineral Resource is Indicated resource which extends to a depth of 70m below the natural surface, with Inferred Mineral Resource lying below and along strike of the mineralisation. Mr Ross Cheyne, the Competent Person for the Open Pit Ore Reserve estimate, has reviewed the work undertaken to date and considers appropriate to the deposit to allow the Ore Reserves to be classified as Probable. The Open Pit Ore Reserve converts approximately 45% of the NEB Indicated Mineral Resource and 82% of the BC Indicated Mineral Resources to Probable Ore Reserve. The lower conversion for NEB is a function of the open pit depth being constrained by the underground mine. Mr Julian Broomfield, the Competent Person for the Underground Ore Reserve estimate, has reviewed the work undertaken and considers it sufficiently detailed and relevant to the deposit to allow these Underground Ore Reserves to be classified as Probable. The Underground Ore Reserve converts approximately a further 26% of the Open Pit Mineral Resource to Probable Ore Reserves. There are no Measured Mineral Resources, so all Probable Ore Reserves are based on Indicated Mineral Resources only, after applying appropriate modifying factors as per the guidelines.

Section 4 Estimation and Reporting of Ore Reserves		
Criteria	**JORC Code explanation**	**Commentary**
Audits or reviews	*The results of any audits or reviews of Ore Reserve estimates.*	The Ore Reserve has been developed by Orelogy Consulting Pty Ltd and has been subject to internal review and due diligence processes. No formal external audit has been undertaken on this Ore Reserve estimate.
Discussion of relative accuracy/confidence	*Where appropriate a statement of the relative accuracy and confidence level in the Ore Reserve estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the reserve within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors which could affect the relative accuracy and confidence of the estimate.* *The statement should specify whether it relates to global or local estimates and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* *Accuracy and confidence discussions should extend to specific discussions of any applied Modifying Factors that may have a material impact on Ore Reserve viability or for which there are remaining areas of uncertainty at the current study stage.* *It is recognised that this may not be possible or appropriate in all circumstances. These statements of relative accuracy and confidence of the estimate should be compared with available production data.*	It should be noted that Ore Reserve estimates are subject to several variables common to mining operations. The Competent Person considers the Ore Reserve estimate can be reasonably justified based on sound geological and economic analysis. All related mining and processing studies at the Project have been undertaken with a relative accuracy appropriate for a DFS confidence level. The Project is not operating, and no production data is available to assess projected Project parameters. Estimates of mining dilution and ore loss have been applied locally based on the geometry of the orebody and assumed operational selectivity. All metallurgical recoveries are based on a global weighted average of testwork results. At the time of the release of this Ore Reserve, there is no known or identified barrier to mining operations proceeding at the Project and the associated validity of the reported Ore Reserve. However, if barriers or risks are identified past the date of this Ore Reserve release, then this Ore Reserve may need to be revised.