10 August 2026

Robust Definitive Feasibility Study and Project Financing Update

Wia Gold Limited (ASX: WIA) (**Wia** or the **Company**) is pleased to announce results of the Definitive Feasibility Study (**DFS** or **Study**) and agreement of indicative project financing terms for the Kokoseb Gold Project (**Kokoseb** or **Project**) in Namibia. The DFS has confirmed the exceptionally robust technical and economic viability of the Project which combined with the proposed project financing facility, provides a well-defined pathway to develop Kokoseb into Namibia's next major gold mine.

HIGHLIGHTS

- Maiden JORC Probable Ore Reserve of **69.8 Mt at 0.87 g/t Au for 1.95 Moz** of gold, estimated at a conservative gold price assumption of US$2,600/oz.
- Average annual gold production of approximately **150 koz** over the first 10 years for a total production of **1,836 koz** over the **14-year** mine life from an open pit mining inventory of **72.9 Mt @ 0.87 g/t Au for 2.03 Moz**.
- Compelling returns with post-tax Project NPV$_{5\%}$ of **US$1.2bn (A$1.7bn)**, IRR of **41%** and a payback period of **1.8 years** at a gold price assumption of **US$3,600/oz**. At **US$4,075/oz** (average July 2026 spot price), post-tax NPV$_{5\%}$ increases to **US$1.6bn (A$2.3bn),** and IRR to **49%**.
- Capital cost of **US$475m**, which includes US$27m pre-production mining, US$40m contingency and US$50m of additional strategic water and power infrastructure investment enabling future expansion optionality.
- Life of mine (**LOM**) C1 cash operating cost of **US$1,519/oz** and All-in Sustaining Cost (**AISC**) of **US$1,696/oz**
- **Indicative terms have been agreed with Sprott Resource Lending Corp. (Sprott) for a project financing facility, comprising a US$360m senior secured debt facility, and a subscription of US$15m in any future equity raise (together, the Facility). The proposed Facility is subject to customary conditions, including completion of detailed due diligence and definitive documentation.**
- Supported by a cash balance of A$119m at 30 June 2026, the proposed Facility underpins Kokoseb's development and provides a well-defined development pathway.
- Completion of DFS paves way for the grant of Mining Licence and environmental approvals anticipated in H2 2026.
- DFS is based on a 5.25Mtpa open pit development, with studies to commence on potential plant expansion and underground development to unlock additional value.

The production target and forecast financial information in this announcement and the DFS Report is based on 96.2% Probable Ore Reserves, and 3.8% Inferred Mineral Resources (contained gold basis). Inferred Mineral Resources are processed in years 6 to 14 only. 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 target will be realised.

Commenting on the outcomes of the DFS, Wia's Managing Director and CEO, Henk Diederichs said:

"Completion of the DFS marks a major milestone for Wia. In just over five years, Kokoseb has advanced from discovery to a development-ready gold project and one of Africa's most compelling undeveloped gold projects. The DFS delivers a maiden Ore Reserve, a robust mine plan and outstanding financial returns, providing a strong foundation for development.

This achievement reflects the dedication of our team, the expertise of our consultants and the ongoing support of our joint venture partner, Epangelo, together with the Namibian Government. Namibia remains one of Africa's premier mining jurisdictions, and we believe Kokoseb is well positioned to become the country's next major gold mine.

The indicative terms agreed with Sprott for a proposed project financing facility, comprising US$360 million debt, and a US$15 million subscription, represent an extremely strong endorsement of both the quality of the Project and the work completed to date.

With the DFS complete, our focus now turns to executing definitive Facility documentation, securing final approvals and progressing early execution activities. We are excited about the next phase of development and the opportunity to create significant value for all stakeholders."

Corey Posesorski, Partner, Sprott, commented:

"*As one of the largest investors dedicated to the natural resources sector, Sprott is excited to partner with Wia on the development of the Kokoseb Gold Project. Kokoseb is a well-defined, high-quality gold development project located in one of Africa's leading mining jurisdictions, and the outcomes of the Definitive Feasibility Study reinforce its compelling economic fundamentals.*

Our financing of Wia is consistent with our strategy to provide innovative and flexible capital to maximise the value of exceptional projects and support strong management teams. Our proposed US$360 million senior debt facility and equity subscription of US$15 million reflect that support, and we look forward to working closely with Wia as the Project advances through construction and into production."

CONFERENCE CALL

CEO, Henk Diederichs, will host an investor conference call for institutional investors and analysts to discuss the DFS results and project financing on Wednesday 12 August at 9.30am (AWST). To register, please use the following link: https://us06web.zoom.us/webinar/register/WN_CmsmcrV-QDOYi61X8VGtew. A recording will be made available following the call.

PROJECT OVERVIEW AND KEY OUTCOMES

Kokoseb is located in Namibia, approximately 320 km by road north-west of the capital city, Windhoek. Located in the Erongo Region, the Project is well supported by existing local infrastructure including power supply, road access and nearby towns. The Project is owned by Wia in a joint venture with Epangelo Mining Company (Pty) Ltd (**Epangelo**), the state-owned mining company of Namibia.

Kokoseb lies within the Okombahe exploration licence, which forms part of Wia's larger Damaran asset portfolio, which consists of 9 tenements covering more than 2,700 km².

Figure 1 – Project Location

The DFS is underpinned by the Company's updated Mineral Resource Estimate of 3.78 Moz, announced to ASX on 10 August 2026 ('Kokoseb Mineral Resource Estimate Increases 29% to 3.78 Moz Au'), which also incorporated a maiden underground Mineral Resource Estimate.

The DFS considered open pit mining only, a conventional leach and carbon-in-pulp processing facility with a nameplate capacity of 5.25 Mtpa, water supply from borefields in the area, power from the Namibian grid, a filtered stack tailings storage facility and associated infrastructure.

Further extensive detail on the process flowsheet, mine plan, capital and operating cost estimates and the maiden Ore Reserve is provided in the DFS Report that is appended to this announcement.

The DFS has defined a maiden Probable Ore Reserve (derived from the Indicated component of the Mineral Resource Estimate) of 69.8 Mt at 0.87 g/t Au for 1.95 Moz of gold, estimated at a conservative gold price assumption of US$2,600/oz.

An open pit mining inventory was developed for the Project based on the declared Ore Reserves. The mining inventory includes only 4.3% of material and 3.8% of contained gold from Inferred Mineral Resources. The mining inventory schedule, which excludes Inferred Mineral Resources from mill feed during the first five years of production, supports average gold production of approximately 150,000 oz per annum over the first 10 years of the 14-year mine life.[1]

Based on capital and operating cost estimates, prepared in line with the Association for the Advancement of Cost Engineering Class 3 estimate to an accuracy level of ±15%, the Study demonstrates that the Project has the capacity to deliver robust returns at the conservative base case gold price of US$3,600/oz.

Key Project outcomes are outlined in Tables 1 to 3 below.

Table 1 – Production and Operating Outcomes

	Unit	Value
Life of Mine (LOM)	Years	14
Total mining inventory	Mt	72.9
Strip ratio (LOM average)	waste:ore	5.5:1
Processing rate	Mtpa	5.25
Average grade	g/t Au	0.87
Average recovery	%	90.4
Average annual gold production, first 5 years	koz pa	161
Average annual gold production, first 10 years	koz pa	150
Average annual gold production, LOM	koz pa	131
Total gold production	koz	1,836
C1 cash operating cost (LOM)	US$/oz	1,519

Table 2 – Financial Outcomes

	Unit	Base Case (US$3,600/oz)	Spot[2] (US$4,075/oz)
Gold price	US$/oz	3,600	4,075
AISC (LOM)	US$/oz	1,696	1,715
Average annual EBITDA (first 10 years)	US$m	281	349
Pre-tax free cash flow (LOM)	US$m	3,020	3,856
Post-tax free cash flow (LOM)	US$m	1,914	2,442
Post-tax $NPV_{5\%}$	US$m	1,214	1,577
Post-tax IRR	%	41	49
Post-tax payback period	Months	21	17

Table 3 – Capital Cost Estimate

	US$m
Plant and infrastructure	362
Owners' costs	46
Contingency	40
Pre-production mining	27
Total pre-production capital	**475**

The DFS capital cost estimate includes development of the Omaruru Alluvial Plains (**OmAP**) borefield (US$34m), which hosts a substantial groundwater resource in excess of 80 Mm^3, significantly

[1] Refer cautionary statement regarding production target on page 1.
[2] The average daily spot price for July 2026

exceeding the Project's long-term water demands. The estimate also incorporates an upgrade to the Project's power supply infrastructure (US$16m), based on a 132 kV connection rather than 66 kV, providing additional capacity for future power demand. Collectively, it is anticipated that these infrastructure investments enable optionality for potential future expansion.

Project permitting is well advanced with the Mining Licence application submitted in October 2025 and the Environmental and Social Impact Assessment (**ESIA**) submitted in March 2026. Approval of both applications, and the granting of the Mining Licence and Environmental Clearance Certificate (**ECC**) is anticipated in the second half of 2026. Preparation of ESIAs for off-site infrastructure is currently underway.

Project development will be preceded by a six-month execution readiness phase that has commenced following completion of the DFS. During this period, project execution plans will be finalised, engineering and consulting groups will be engaged, front-end engineering and design advanced, long lead and critical path procurement activities initiated, and final site investigations completed, with first gold production currently targeted for the fourth quarter of 2028.

PROJECT FINANCING

The Company has completed a highly competitive process to secure funding for the development of Kokoseb in parallel with the completion of the DFS. Following completion of an initial phase of independent technical review and lender due diligence, the Company has agreed an indicative, non-binding term sheet for the Facility, which comprises a US$360m senior secured debt facility and a subscription of US$15m in any future equity raise.

Key aspects of the Facility include a tenor of approximately six years, a highly competitive interest rate linked to SOFR plus a margin, with interest capitalised for the first two years following the closing date of the Facility and no mandatory hedging requirements.

Sprott's investment committee has approved the lender proceeding to detailed due diligence, with the provision of the Facility remaining subject to conditions customary for a facility of this nature, including completion of such due diligence, negotiation and execution of definitive documentation, final investment committee approval from Sprott and receipt of the Mining Licence and ECC. Following financial close and satisfaction of all conditions, including receipt of the permits referred to above, US$50m will be available to be drawn, with the balance available to fund construction.

Wia is targeting completion of definitive documentation and financial close under the Facility in the fourth quarter of 2026. In accordance with the proposed Facility, on financial close Wia will issue Sprott with 67,500,000 warrants over unissued Wia shares.

NEXT STEPS

Following completion of the DFS, the Wia Board has approved the commencement of execution readiness activities ahead of finalising definitive Facility documentation and commencing construction. These activities include:

- Establishing the owner's team, with the Project Director already appointed.
- Finalising the Project execution and contracting strategy.
- Advancing detailed engineering and design in parallel with Project approvals.
- Ordering long-lead items on the Project critical path.
- Mining Licence and environmental approvals are anticipated in H2 2026.
- Initiate plant expansion and underground development studies.

Subject to timely receipt of key approvals and finalising definitive Facility documentation, construction is targeted to commence during H2 2026, with first gold pour targeted for Q4 2028.

MINERAL RESOURCE ESTIMATE

The Kokoseb Mineral Resource Estimate is summarised in Table 4 below at a 0.5 g/t Au cut-off grade for the open pit and 0.85 g/t for the underground. The open pit Indicated Mineral Resource provides the basis for the maiden Ore Reserve set out in Table 5. The Mineral Resource is reported inclusive of the Ore Reserve.

Table 4 – Kokoseb Mineral Resource Estimate

		Indicated			Inferred			TOTAL		
	Cut-off Au g/t	Tonnes (Mt)	Au g/t	Au Moz	Tonnes (Mt)	Au g/t	Au Moz	Tonnes (Mt)	Au g/t	Au Moz
Open Pit	**0.14**	156	0.57	**2.86**	120	0.51	**2.0**	**276**	**0.54**	**4.83**
	0.30	100	0.77	**2.48**	71	0.70	**1.6**	171	0.74	**4.07**
	0.40	80.7	0.87	**2.26**	54	0.81	**1.4**	135	0.84	**3.65**
	0.50	63.9	0.98	**2.01**	41	0.92	**1.2**	**105**	**0.96**	**3.23**
	0.80	32.4	1.32	**1.38**	18	1.3	**0.75**	50	1.3	**2.13**
Underground	**0.85**	**-**	**-**	**-**	**7.9**	**2.2**	**0.56**	**7.9**	**2.2**	**0.56**
Total	**0.50/ 0.85**	**63.9**	**0.98**	**2.01**	**49**	**1.1**	**1.8**	**113**	**1.0**	**3.78**

Kokoseb Mineral Resource estimates for selected cut-off grades. The estimates in this table are rounded to reflect their precision; rounding errors are apparent.

The Mineral Resource Estimate referred to in this announcement was first disclosed in accordance with the requirements of ASX Listing Rule 5.8 in the Company's ASX announcement dated 10 August 2026, titled 'Kokoseb Mineral Resource Estimate Increases 29% to 3.78 Moz Au'. The Company confirms that it is not aware of any new information or data that materially affects the Mineral Resource Estimate contained in this announcement and all material assumptions and technical parameters underpinning the estimate in the previous announcement continue to apply and have not materially changed. The announcement is available to view on www.wiagold.com.au

MAIDEN ORE RESERVE

The Ore Reserve is reported in accordance with the JORC Code (2012 Edition). All Ore Reserves are in the Probable category and are derived entirely from Indicated Mineral Resources, optimised at a gold price of US$2,600/oz. Tonnages are dry metric tonnes and minor discrepancies may occur due to rounding.

Table 5 – Kokoseb Maiden Ore Reserve

Ore Reserve category	Tonnes (Mt)	Grade (g/t Au)	Contained gold (Moz)
Proved	–	–	–
Probable	69.8	0.87	1.95
Total	**69.8**	**0.87**	**1.95**

MATERIAL ASSUMPTIONS AND OUTCOMES OF THE DFS

Key DFS assumptions and outputs are summarised in Table 6 below. Further details are available in the DFS Report, which is appended to this announcement.

Table 6 – Material assumptions and outcomes of the DFS

	Unit	Value
Mining		
Ore Mined	Mt	69.8
Inferred Inventory Mined	Mt	3.1
Total Mining Inventory	**Mt**	**72.9**
Strip ratio (LOM average)	waste:ore	5.5:1
Average Grade	g/t	0.87
Contained Gold	koz	2,031
Contained Gold from Inferred Mineral Resources	%	3.8
Production		
Mine Life	years	14
Processing Rate	Mtpa	5.25
Total Material Processed	Mt	72.9
Average Recovery	%	90.4
Total Gold Production	koz	1,836
Average Gold Production		
Years 1 to 5	koz pa	161
Years 1 to 10	koz pa	150
Life of Mine	koz pa	131
Capital Costs		
Mining Mobilisation & Pre-production	US$m	26.8
Direct Construction Cost	US$m	302.6
Indirect Construction Cost	US$m	59.1
Owners Cost	US$m	46.1
Contingency	US$m	40.2
Total Pre-Production Capital Cost	US$m	474.8
Sustaining Capital Costs	US$m	48.1
Mine Closure Costs (excluding salvage)	US$m	37.9
Operating Costs		
C1 cash operating cost (LOM)	US$/oz	1,519

Table 7 – Financial outcomes of the DFS

	Unit	Base Case (US$3,600/oz)	Spot (US$4,075/oz)
AISC (LOM)	US$/oz	1,696	1,715
Pre-tax $NPV_{5\%}$	US$m	1,923	2,493
Pre-tax IRR	%	51	61
Post-tax $NPV_{5\%}$	US$m	1,214	1,577
Post-tax IRR	%	41	49
Post-tax Payback Period	Months	21	17

Ore Reserve classification criteria

The Ore Reserve Estimate is derived from the Indicated component of the updated Mineral Resource Estimate.[3]

A mining study at a DFS level was carried out on the open pit Indicated portion of the Mineral Resource, including open pit optimisation, open pit mine design, production scheduling 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 Ore Reserve economic cut-off grade has been selected at 0.4 g/t, which is lower than the reported 0.5 g/t cut-off grade used to report the open pit Mineral Resource. However, the open pit Ore Reserve designs are within the “reasonable prospects for eventual economic extraction” optimisation shell used to define the open pit Mineral Resource.

Mr Jake Fitzsimons, the Competent Person for the open pit Ore Reserve estimate has reviewed the work undertaken to date and considers it sufficiently detailed and relevant to the deposit to allow these Ore Reserves to be classified as Probable.

Mining method and assumptions

Open pit mining will be performed as a conventional drill, blast, truck and shovel operation, which is considered appropriate for the style of the deposit. The mining method is based on eight pit stages across two separate, but adjacent, open pits. 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 4% and ore loss of 2%. This is reflective of the Mineral Resource modelling methodology and block size along with the broader and more continuous nature of the orebody. The minimum mining width applied in the designs is 30m and is appropriate for the selected mining equipment fleet.

Processing method and assumptions

Metallurgical testwork has been completed on representative samples from the deposit across 2025 and 2026 and has confirmed the ore is free milling and suitable for conventional leach/CIP processing. The metallurgical testwork defined the key process design criteria and investigated the variability across the deposit as well as including bulk sample 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 primary crushing in a single stage gyratory 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**), grinding ore to 63µm (P_{80}) based on average ore hardness, ahead of treatment in a conventional gravity/leach/CIP circuit with an overall leach residence time of 48 hours. The process plant has been designed based on a throughput of 5.25 Mtpa. Throughput has been assumed to ramp up from 60% of nameplate capacity in month 1 to 100% of nameplate capacity by month 5 of operations.

Power demand for the comminution circuit will vary according to lithology between 25 and 32 kWh/t.

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[3] Refer ASX Announcement dated 10 August 2026 ‘Kokoseb Mineral Resource Estimate Increases 29% to 3.78 Moz Au’

Gold recovery by gravity is expected to be 45% across all material.

Review of leaching performance and kinetics across the full range of variability samples demonstrated an improvement in leach extraction from increasing leaching residence time from the industry standard 24-hours to 48-hours with this increase in extraction justifying the additional cost of the leaching circuit.

Gold recovery has been demonstrated to follow relationships between residual leach residue grade and leach residence time as defined by the formulas set out below (and averages 90.4% over the LOM).

At 63 µm (P_{80}): Residue grade (g/t) = 0.2449 – 0.0431 x ln(residence time (hrs)), minimum 0.083 g/t

At 75 µm (P_{80}): Residue grade (g/t) = 0.3283 – 0.0655 x ln(residence time (hrs)), minimum 0.088 g/t

Cut-off grades

A cut-off grade of 0.4 g/t has been applied for the Ore Reserve estimation with material above the economic cut-off grade and below the Ore Reserve cut-off grade to be stockpiled as mineralised waste for potential later processing.

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 Wia and external parties, such as geotechnical reports and metallurgical results, and a gold price of US$2,600/oz. Designs based on optimised pit shells and subsequent schedules were completed using the Hexagon MinePlan Schedule Optimizer (**MPSO**) scheduling tool.

The Indicated Mineral Resources within the open pit and above the relevant cut-off grade has been defined as Ore Reserves.

Infrastructure

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 supply, borefield 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 within 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 sourced from NamPower with a 132 kV connection developed at the existing Omburu substation and transmitted to the Project via a new 132 kV overhead transmission line. Studies undertaken by NamPower have demonstrated sufficient power supply capacity exists at the Omburu substation to support the Project.

Water supply to the Project operations will be sourced from a borefield developed to access the groundwater discovered in the Omaruru Alluvial Plains paleochannel aquifer. The borefield will consist of 13 bores approximately 200 to 300 m in depth, with pumping infrastructure to deliver the water to the Project site. Power for the borefield will be from a new 33 kV overhead power line run from the Project.

Water for construction will be sourced from the combination of a small borefield across the exploration lease and from the NamWater operated Okombahe water supply scheme, approximately 35 km from the Project site. Minor upgrades will be required to the Okombahe water supply scheme to support the construction water supply.

Accommodation for the Project will be at a site accommodation village with the majority of the Project personnel accommodated during their time at work. Local labour will be sourced from the nearby towns of Omaruru, Uis and Okombahe with these personnel covering day-shift only roles so they can reside in these locations.

Temporary accommodation camps will be established by the construction contractors for the construction period.

Labour and Contracting

Professional, skilled and unskilled labour is readily available in Namibia for the Project.

Mining will be undertaken by a contract mining services company to de-risk the supply of mining operations labour, equipment and systems.

Social, Environmental & Permitting

In March 2026, Wia submitted an ESIA to the Ministry of Environment, Forestry and Tourism (**MEFT**) for consideration with review currently underway and an anticipated approval in the second half of 2026.

An application for the Mining License was submitted in October 2025 to the Ministry of Industries, Mines and Energy (**MIME**) with review currently underway and approval pending the submission of the DFS. Approval of the Mining License is anticipated in the second half of 2026.

An accessory works permit application was submitted to the MIME in May 2026 to cover the development of supporting infrastructure for the Project and is currently under review with approval anticipated in the second half of 2026.

Design and control measures for environmental and social risks have been factored into the mine plan and designs to avoid impact on biodiversity, heritage and cultural sites, groundwater, surface water, air quality, noise and traffic. Socio-economic impacts, positive and negative, have also been assessed with management plans developed and put in place.

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 an arid environment that experiences water scarcity. 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 only the nearby towns of Okombahe, Omaruru and Uis close to the Project. Agriculture and tourism are the main occupations for communities close to the Project with the only nearby industrial employer being the tin mine at Uis. 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 and land used for grazing. Wia 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.

The Project is located on communal land within the jurisdiction of the !Oe≠Gân Traditional Authority and is therefore subject to both customary and statutory land tenure arrangements in Namibia. Access to communal land for mining activities is regulated by the Communal Land Reform Act, 2002 and the Minerals (Prospecting and Mining) Act, 1992. While mineral rights are vested in the State and administered by the MIME the occupation and use of communal land for mining purposes requires the consent of the relevant traditional authority and approval from the Communal Land Board. These legislative frameworks also establish obligations to compensate affected land users for loss of, or interference with, their rights and assets and, where necessary, to implement appropriate relocation measures. Baseline studies have identified eighteen homesteads, comprising forty-three Project affected persons that may require either physical displacement, economic displacement, or both.

Stakeholder engagement, compensation and relocation activities will be implemented through a relocation action plan, which will be developed based on, and expanded upon, the existing relocation planning framework completed in 2026.

Capital Cost

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 open pit mining. All significant and measurable items are itemised with smaller items factored in as per industry practice. A 3% royalty plus a 1% gross revenue export tax has been included. Treatment and refining charges are based on pricing from a global refining company.

PRODUCTION TARGET

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 Report, which is appended to this announcement.[4]

This includes assumptions about the availability of funding (see Project Financing above). While the Company 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 the Company 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 the Company's existing shares. Given the uncertainties involved, investors should not make any investment decisions based solely on the results of the DFS.

This announcement is authorised for release by the Board of Directors.

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Competent Persons Statements

The information in this announcement that relates to Ore Reserves is based on and fairly represents information compiled by Mr Jake Fitzsimons, BEng (Geological), a Competent Person who is a Fellow of the Australasian Institute of Mining and Metallurgy. Mr Fitzsimons is a full-time employee of Orelogy Consultants Pty Ltd. Mr Fitzsimons 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 JORC Code (2012 Edition). Mr Fitzsimons consents to the inclusion in this announcement of the matters based on his information in the form and context in which it appears.

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[4] Refer also cautionary statement regarding production target on page 1.

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 JORC Code (2012 Edition). 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

This announcement contains certain forward-looking statements and projections regarding: estimated, resources and reserves; planned production and operating costs profiles; planned capital requirements; and planned strategies and corporate objectives.

Forward-looking statements are subject to a variety of known and unknown risks, uncertainties and other factors that could cause actual events or results to materially differ from those reflected in the forward- looking statements, including, without limitation: inherent uncertainties and risks associated with mineral exploration; uncertainties related to the availability of future financing necessary to undertake activities on Wia's properties; uncertainties related to the possible recalculation of, or reduction in Wia's mineral resources; uncertainties related to the outcome of studies; uncertainties relating to fluctuations in gold prices; the risk that Wia's title to its properties could be challenged; risks related to Wia's ability to attract and retain qualified personnel, uncertainties related to general economic and global financial conditions; uncertainties related to the competitiveness of the industry; risk associated with Wia being subject to government regulation, including changes in regulation; risks associated with Wia being subject to environmental laws and regulations, including a change in regulation; risks associated with Wia's need for governmental licenses, permits and approvals; uninsured risks and hazards; risk related to the integration of businesses and assets acquired by Wia; risk associated with Wia having no history of earnings or production revenue; risks associated with fluctuation in foreign exchange rates; risks related to default by joint venture parties (if any), contractors and agents, inherent risks associated with litigation; risk associated with potential conflicts of interest; risk related to effecting service or process on directors resident in foreign countries; uncertainties related to Wia's limited operating history; risks related to Wia's lack of a dividend history; risks relating to short term investments; and uncertainties related to fluctuations in Wia's share price.

The forward-looking statements contained in this announcement are based on the assumptions, beliefs, expectations and opinions of management as of the date hereof and which Wia believes are reasonable in the circumstances, but no assurance can be given that these expectations will prove to be correct. These assumptions include but are not limited to that Wia's exploration of its properties and other activities will be in accordance with Wia's public statements and stated goals, that there will be no material adverse change affecting Wia or its properties, anticipated costs and timing for Wia's activities and such other assumptions as set out herein. Such forward looking statements/projections are estimates for discussion purposes only and should not be relied upon. They are not guarantees of future performance and involve known and unknown risks, uncertainties and other factors many of which are beyond the control of the Company. The forward-looking statements/projections are inherently uncertain and may therefore differ materially from results ultimately achieved.

Wia does not make any representations, provides no warranties concerning the accuracy of the forward-looking statements and disclaims any obligation to update or revise any forward-looking statements based on new information, future events or otherwise except to the extent required by applicable laws. Except for statutory liability which cannot be excluded, each of Wia and its related bodies corporate and their respective officers, employees and advisers disclaim any responsibility for the accuracy or completeness of the material contained in this announcement and exclude all liability whatsoever (including in negligence) for any loss or damage (direct or indirect) which may be suffered by any person as a consequence of any information in this announcement or any error in it or omission from it.

JORC Table 1

Section 1 Sampling Techniques and Data

Criteria	JORC Code explanation	Commentary
Sampling techniques	• *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 down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.* • *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.*	• Reverse circulation (RC) drilling was completed using a dedicated RC rig. • RC samples were collected from the drill rig • cyclone over 1 m down-hole intervals and subsampled by cone-splitting; full length of the drill holes was sampled. • Samples are typically circa 2-4 kg weight. A duplicate sample was retained on site for future reference. • Diamond drilling was completed using a dedicated diamond rig. Drillholes were generally angled at -60° from surface. • Diamond core was cut in half using a core saw for HQ diameters; NQ diameters were sampled full core. Sampling intervals are decided by a Company Geologist, based on the lithological contacts and on any change in alteration or mineralisation style. • Core sample length vary between 0.5 m and 2 m. The half core sampling is done by a Company Geologist. • Deep diamond drill holes (over 300 m downhole depth) are not sampled along most of the hanging wall side, which is known as barren from initial wide spaced drilling, as well as not sampled along large granitic bodies intersected downhole.
Drilling techniques	• *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).*	• RC drilling utilised 140 mm (5.5 inch) face sampling bits. • Diamond drilling was undertaken at HQ (drill holes from surface) or NQ (from surface or as diamond tails) diameters, and oriented using the Reflex Act III digital core orientation equipment.
Drill sample recovery	• *Method of recording and assessing core and chip sample recoveries and results assessed.* • *Measures taken to maximise sample recovery and ensure representative nature of the samples.* • *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.*	• RC samples were routinely weighed, with sample weights available for all of mineralised domain samples, indicating an average recovery for mineralised samples of around 76%, representative of good quality RC drilling. • Recovered core lengths for generally 3 m core runs show an average recovery of around 99.9% for mineralised samples, which is representative of excellent quality diamond drilling. • RC sampling was closely supervised by Wia field geologists and employed face sampling bits and drilling equipment with sufficient capacity to provide dry, high recovery samples for the majority of mineralised drilling, with field geologist's sample condition logging categorising around 97.8% of mineralised domain RC samples as dry, and 1.9% as moist and 0.3% as wet respectively.

Criteria	JORC Code explanation	Commentary
		• There is no notable association between sample recovery and gold assay grade for RC or diamond samples, and available information indicates that the sampling is free of any biases associated with preferential loss or gain.
Logging	• *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.* • *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.* • *The total length and percentage of the relevant intersections logged.*	• The entire length of all RC and diamond holes were logged by Company geologists using industry standard methods, including recording of lithology, alteration, mineralisation and weathering. Sieved RC sample collected for logging were stored in chip trays for future geological reference and all core was routinely photographed. All core was geotechnically logged, including recording of RQD and fracture frequency. • The logging is qualitative and quantitative in nature and is of appropriate detail for support the current Mineral Resource estimates.
Sub-sampling techniques and sample preparation	• *If core, whether cut or sawn and whether quarter, half or all core taken.* • *If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.* • *For all sample types, the nature, quality and appropriateness of the sample preparation technique.* • *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.* • *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.* • *Whether sample sizes are appropriate to the grain size of the material being sampled.*	• Diamond core was either halved with a diamond saw to provide assay sub-samples over generally 1 m in length – for HQ diameters – or sampled full core – for NQ diameters. RC samples were collected from the rig cyclone and sub-sampled by cone-splitting. Where required, to produce sample weights of around 2.5 kg, larger samples were passed through a riffle splitter. Channel samples were collected 1 m intervals from trench walls, with samples collected in a halved PVC pipe used to ensure consistent coverage of each interval. • RC samples were generally dry, with field geologist's sample condition logging categorising around 97.8% of mineralised domain RC samples as dry, and 1.9% as moist and 0.3% as wet respectively. The rare wet samples were not shipped to the laboratory. • Field sampling was closely monitored by Company Geologists. Quality control monitoring included routine collection of RC field duplicates, and submission of coarse blanks and certified reference standards. • The sampling technique is considered industry standard and effective for this style of drilling. The sample sizes are appropriate for the material being sampled. • Samples collected until December 2025 were submitted to ALS in Okahandja, Namibia for preparation comprising oven drying, crushing to greater than 70% passing 2 mm with 1 kg riffle split sub-samples pulverized to 85% passing 75 microns in a disc pulveriser. Sample pulps were shipped to ALS Johannesburg for analysis. • Samples collected post December 2025 were submitted to MsAlabs in Omaruru, Namibia for preparation and assaying. Preparation comprised oven drying, crushing to greater than 70% passing 2 mm with 1 kg riffle split sub-samples pulverized to 85% passing 75 microns in a disc pulveriser. Sample pulps where then assayed at the same laboratory.
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.* • *For geophysical tools, spectrometers, handheld XRF instruments, etc, the*	• RC samples and core samples were assayed by 50 g lead collection fire assay in new pots and analysed by Atomic Absorption Spectroscopy (AAS) for gold, a technique that is considered total. • Industry best practice procedures were followed and included submitting blanks, field duplicates and Certified

Criteria	JORC Code explanation	Commentary
	parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc. • *Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.*	Reference Material. Acceptable levels of accuracy and precision have been confirmed.
Verification of sampling and assaying	• *The verification of significant intersections by either independent or alternative company personnel.* • *The use of twinned holes.* • *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.* • *Discuss any adjustment to assay data.*	• At this stage, the intersections have been verified by the Company Geologists and umpire samples were submitted for checks to a second laboratory (Interteck Genalysis until December 2025 and then ALS from January 2026 onward). • No dedicated twin holes have been drilled at Kokoseb; however close spaced drilling (less than 25 m spacing) was locally completed • All field data is manually collected, entered into excel spreadsheets, validated and loaded into Wia's master database. Assay results are directly merged into the database from laboratory source files. • Electronic data is stored on a cloud server and routinely backed up. • Data is exported from the database for processing in a number of software packages with verification undertaken by company personnel including checking for consistency within, and between database tables. • Assay data was not adjusted for use in resource modelling.
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.* • *Specification of the grid system used.* • *Quality and adequacy of topographic control.*	• Collars for all drill holes included in the current estimates were accurately surveyed in WGS84 Zone 33S coordinates by contract surveyors using differential GPS equipment. • RC holes were generally down-hole surveyed at 20 m down-hole intervals with a magnetic Trushot tool (410 holes), or less commonly a BDVG42 tool at intervals of around 2 m to 15 m (37 holes), with 10 holes assumed to run straight at the design orientation. Diamond holes were down-hole surveyed at intervals of around 20 m with a Reflex mutlishot EZ-TRAC tool. • Mineral resources are reported below a DTM generated from WorldView2 stereo-imagery with an estimated vertical accuracy of approximately 1 m.
Data spacing and distribution	• *Data spacing for reporting of Exploration Results.* • *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.* • *Whether sample compositing has been applied.*	• The data spacing and distribution of sampling is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource estimation procedures and classifications applied. • Sample assay grades were composited to 2 m down hole intervals for open pit resource modelling and 1 m down hole intervals for underground resource modellling. • Drill holes were planned on a set grid with spacing varying between 25 m and 100 m, depending on the sections.

Criteria	JORC Code explanation	Commentary
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.* • *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.*	• Drill holes were planned using geological information collected from the trenches and from the detailed mapping completed over the prospect. They are positioned perpendicular to the main schistosity and so to the inferred mineralisation main controls. • Drill holes are inclined at around 55 to 60 degrees. Trenches are sub-horizontal. The orientation of sampling achieves un-biased sampling.
Sample security	• *The measures taken to ensure sample security.*	• Sampling is supervised by Wia geologists and all samples are bagged and sealed on site prior to delivery to the laboratory in Okahandja by company staff. No other personnel are permitted un-supervised access to the samples prior to delivery to ALS.
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	• Mr Abbott reviewed the sampling quality information and drill data in April 2023 for the maiden MRE; it showed no inconsistencies, or issues of concern.

Section 2 Reporting of Exploration Results

(Criteria listed in the preceding section also apply to this section.)

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 security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.*	• The Damaran Project comprises 9 exclusive prospecting licenses (EPLs 4833, 7246, 4818, 6534, 6535, 8249, 7980, 8021, 8709) and located in central Namibia.EPL4833, EPL4818, EPL7246, and EPL8249 are held under an 80% earn-in and join venture agreement between Damaran Exploration Namibia (Pty) Ltd and Epangelo Mining (Pty) Limited, a private mining investment company with the Government of the Republic of Namibia as the sole shareholder. • EPL6534, and EPL6535 are held under a company called Gazina Investments which is owned 90% by Wia and 10% by the vendor. • EPL7980, EPL8021 and EPL8709 are 100% held by Wia in the name of Damaran Exploration Namibia (Pty) Ltd
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	• Work completed prior to WiaGold includes stream sediment sampling, mapping, soil and rock chip sampling by Teck Cominco Namibia but data is unavailable. • This work did not cover the Okombahe permit (EPL4818), host of the Kokoseb deposit.
Geology	• *Deposit type, geological setting and style of mineralisation.*	• The Kokoseb Gold Project lies withing the Northern Central Zone of the Pan-African Damaran Orogenic Belt. The Project area is underlain by neo-Proterozoic metasediments, including the Kuiseb schist formation, host of Kokoseb, Twin Hills and Ondundu gold deposits in Namibia. Known gold deposits, including Kokoseb, are orogenic type deposits by nature. • Kokoseb gold mineralisation is hosted by the Kuiseb schist formation, biotite-schists (metasediments) which have been intruded by several granitic phases. The gold mineralised zone appears as a contact like aureole around a central granitic pluton, with a diameter of approximately 3km in each direction. • Gold mineralisation is present as native gold grains and lesser silver bearing gold grains that are spatially associated with sulphides dominated by pyrrhotite, löllingite and arsenopyrite. Gold grains have developed at the contact between löllingite and arsenopyrite following a retrograde reaction.
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.*	• No new results are reported in this announcement. • All drill hole collars informing this MRE are presented at the various plan views and perspectives

Criteria	JORC Code explanation	Commentary
	• *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.*	
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.* • *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.* • *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	• No drill hole results are reported in this announcement.
Relationship between mineralisation widths and intercept lengths	• *These relationships are particularly important in the reporting of Exploration Results.* • *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be 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').*	• No new results are reported in this announcement.
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.*	• Appropriate maps are included in this announcement.
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.*	• No new results are reported in this announcement.
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.*	• No other exploration data is being reported at this time.

Criteria	JORC Code explanation	Commentary
Further work	• *The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling).* • *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	• Refer to the text in the announcement on follow-up and/or next work programs.

Section 3 Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

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 validation procedures used.*	• Database entries are routinely validated by Wia personnel using a variety of software packages. • Data verification checks are conducted on a regular basis by Mr Couderc to ensure internal consistency within, and between database tables.
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 Abbott visited the Kokoseb Gold Project from the 22nd to the 24th of February 2023. Mr Abbott inspected surficial exposures, drill samples, and drilling and sampling activities and had detailed discussions with Company geologists gaining an improved understanding of the geological setting and mineralisation controls, and sampling activities. • Mr Couderc, in his capacity of Group Exploration Manager, has been present on site at least once per quarter since 2021. Mr Couderc conducted the initial surface geological mapping at Kokoseb, has logged key representative drill holes in detail and is directly involved in the day-to-day management of drilling and drill logging alongside the Company's geologists.
Geological interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.* • *The effect, if any, of alternative interpretations on Mineral Resource estimation.* • *The use of geology in guiding and controlling Mineral Resource estimation.* • *The factors affecting continuity both of grade and geology.*	• Interpretation of the deposit's geological setting is based on detailed surface mapping, on geological logging of drill samples, and on observations from trenches. Kokoseb mineralisation is hosted by biotite-schists representing metamorphosed sedimentary rocks which have been intruded by several granitic phases, with local mineralisation trends consistent with schistosity. • The mineralised domains used for resource modelling are consistent with geological interpretations. A surface representing the base of weathering interpreted from drill hole logging, which within the pit shell constraining the MRE ranges from around 1 to 100 m depth and averages around 33 m depth was used for density assignment. • Confidence in the geological interpretation is sufficient for the current resource estimates. Alternative interpretations are considered unnecessary.
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.*	• The Open Pit MIK modelling utilised a set of mineralised domains interpreted by Matrix which capture composites with gold grades of generally greater than 0.1 g/t and delineate zones within which the tenor of mineralisation is similar. Within the resource pit shell, the mineralised domains have a combined strike length of around 7 km, with average horizontal widths within of around 70 m, 100 m, 70 m, 50 m and 60 m for the south, northwest, north, east and splay domains respectively. • Open Pit Mineral Resources are reported within an optimal pit shell generated at a gold price of $US3,250/oz. The pit shell extends over around 6.4 km of strike and reaches a maximum depth of around 580 m. Open Pit Mineral Resources extend from surface to 580 m depth with around 80% from depths of less than 280 m and 95% from shallower than 450 m.

Criteria	JORC Code explanation	Commentary
		• The Underground resource estimates are derived from block models generated by Matrix within wire-framed mineralised domains interpreted by Wia, designated as the South, Central, Northwest, Central-north and North zones respectively. These domains, which are interpreted by Wia to represent continuous geological features capture 1 m down-hole composited gold grades for drill hole intercepts of generally greater than 0.85 g/t, with lower grade intervals included for continuity. The mineralised domains were interpreted with a minimum down-hole intercept length of generally around 3 m. Underground Mineral Resources comprise estimates from the south and central zones constrained below the pit shell. Estimate for the central-north and north zones, which have only sparse drill coverage below the resource pit do not inform Mineral Resources. • The southern domain is sub-vertical, and trends north-south, plunging toward the south at around 65 degrees. The portion of the domain included in mineral resources extends over around 200 m of strike to a maximum of around 560 m, around 270 m below the resource pit in this area, with an average horizontal width of around 17 m. The central domain dips towards the north east at around 75 degrees with a moderate southerly plunge. The portion of the domain included in mineral resources extends over around 600 m of strike to around 720 m depth, around 400 m below the resource pit shell in this area, with a horizontal width averaging around 10 m.
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.* • *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 assumptions made regarding recovery of by-products.* • *Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).* • *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.* • *Any assumptions behind modelling of selective mining units.* • *Any assumptions about correlation between variables.*	**OPEN PIT MODELLING** • Open Pit Mineral Resources were estimated by Multiple Indicator Kriging with block support correction to reflect open pit mining selectivity, a method that has been demonstrated to provide reliable estimates of resources recoverable by open pit mining for a wide range of mineralisation styles. The modelling technique is appropriate for the mineralisation style, and potential mining method. The estimates are based on 2m down-hole composited gold assay grades from RC and diamond drilling. • Micromine software was used for data compilation, domain wire framing and coding of composite values and GS3M was used for resource estimation. The resulting estimates were imported into Micromine for pit optimisation resource reporting. • Grade continuity was characterised by indicator variograms modelled at 19 indicator thresholds. Class grades were derived from class mean grades with the exception of the upper bin grades which were selected as follows: - South: 4 outliers of > 15 g/t were excluded, giving a bin mean of 6.19g/t - Northwest: 12 outliers of > 30 g/t were excluded, giving a bin mean of 10.56 g/t - North: 2 outliers of > 8 g/t were excluded, giving a bin mean of 4.33 g/t - East: 1 outlier composite of > 9 g/t was exclude giving a bin mean of 2.96 g/t

Criteria	JORC Code explanation	Commentary
	• *Description of how the geological interpretation was used to control the resource estimates.* • *Discussion of basis for using or not using grade cutting or capping.* • *The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.*	- Spay: Upper bin mean (3.18 g/t). • No composites were excluded from the dataset used for grade modelling. This approach reduces the impact of small numbers of extreme gold grades on estimated resources and in the Competent Person's experience is appropriate for MIK modelling of highly variable mineralisation such as Kokoseb. • The model estimates include a variance adjustment to give estimates of recoverable resources above gold cut-off grades for open pit mining selectivity of around 5 by 5 by 5 m with ore definition based on grade control sampling of around 7.5 by 10 m (cross strike, strike), with 1.25 m vertical samples notionally representing 1.5 down-hole samples from 55 degree incline holes. The variance adjustments were applied using the direct lognormal method and variance adjustment factors derived from the northwest domain variogram model of gold grades. • The available sampling tests mineralisation at along strike spacings of generally around 50 to rarely 200 m. Modelling utilised 25 by 25 by 10 m panels (east, north, vertical). The estimates are generally extrapolated to around 50 m from drilling, with comparatively rare extrapolation in areas of greater continuity to a maximum of around 100 m from drilling. Around 90% of the combined estimates are within 50 m of drilling and 99% are within around 75 m. • Estimation included a five-pass octant search strategy with ellipsoids aligned with local mineralisation orientation, with radii and minimum data requirements as follows: - Search 1 Radii: 60,60,15 m (dip, strike, cross strike), minimum data/octants:16/4, maximum data:48 - Search 2 Radii: 90,90,22.5 m (dip, strike, cross strike), minimum data/octants:16/4, maximum data:48 - Search 3 Radii: 120,120,30 m (dip, strike, cross strike), minimum data/octants:16/4, maximum data:48 - Search 4 Radii: 120,120,30 m (dip, strike, cross strike), minimum data/octants:8/2, maximum data:48 - Search 5 Radii: 240,240,60 m (dip, strike, cross strike), minimum data/octants:8/2, maximum data:48 • Indicated Mineral resources are primarily informed by search pass 1 and 2 which inform around 86.3% and 13.1% respectively, with searches 3 to 5 contributing only a small proportion. Inferred Resources are dominated by search pass 1 to 4 panels (99.9%), with search pass 5 contributing around 0.1%. **UNDERGROUND MODELLING** • The Underground resource modelling utilises two block models generated by Matrix within wire-framed mineralised domains interpreted by Wia, designated as the south and central zones respectively. These domains, interpreted by Wia to represent continuous geological features, capture 1 m down-hole composited gold grades for drill hole intercepts of generally greater than 0.85 g/t, with lower grade intervals included for continuity.

Criteria	JORC Code explanation	Commentary
		• The mineralised domains were interpreted with a minimum down-hole intercept length of generally around 3 m. • The southern domain is sub-vertical, and trends north-south, plunging toward the south at around 65 degrees. The portion of the domain included in mineral resources extends over around 140 m of strike to a maximum of around 560 m, around 270 m below the DFS pit in this area, with an average horizontal width of around 16 m. • The central domain dips towards the north east at around 75 degrees with a moderate southerly plunge. The portion of the domain included in mineral resources extends over around 600 m of strike to around 720 m depth, around 400 m below the DFS pit shell in this area, with a horizontal width averaging around 10 m. • The Kriging utilised 4 by 25 by 20 m blocks (cross strike, strike, vertical) with sub-blocking to minimum dimensions of 0.25 by 1.25 by 1.0 m for precise representation of domain boundary. • Estimation included a five-pass octant search strategy with ellipsoids aligned with local mineralisation orientation, with radii and minimum data requirements as follows: - Search 1 Radii: 50,50,10 m (dip, strike, cross strike), minimum data/octants:8/2, maximum data:16 - Search 2 Radii: 75,75,15 m (dip, strike, cross strike), minimum data/octants:8/2, maximum data:16 - Search 3 Radii: 75,75,15 m (dip, strike, cross strike), minimum data/octants:4/1, maximum data:16 - Search 4 Radii: 100,100,20 m (dip, strike, cross strike), minimum data/octants:4/2, maximum data:16 - Search 5 Radii: 200,200,40 m (dip, strike, cross strike), minimum data/octants:4/2, maximum data:16 - Search 6 Radii: 200,200,40 m (dip, strike, cross strike), minimum data/octants:2/1, maximum data:16 • Underground Mineral resources are primarily informed by search pass 1 to 4 which inform around 91% and, with searches 5 and 6 contributing around 9% respectively. • The estimates are based on variably spaced drilling, with intercepts with the MRE volume approximating 100 m spacing and are extrapolated to generally a maximum of around 50 m from drilling. **GENERAL** • The modelling did not include estimation of any deleterious elements or other non-grade variables. No assumptions about correlation between variables were made. • Block model reviews included visual comparisons of the model with the informing data.
Moisture	• *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	• Tonnages were estimated on a dry basis with densities derived from immersion measurements of oven dried diamond core samples.

Criteria	JORC Code explanation	Commentary
Cut-off parameters	• *The basis of the adopted cut-off grade(s) or quality parameters applied.*	• The cut-off grades selected for reporting reflect Wia's view of potential Project economics.
Mining factors or assumptions	• *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is 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.*	• Open Pit Mineral Resource estimates include a variance adjustment to give estimates of recoverable resources above gold cut-off grades for open pit mining selectivity of around 5 by 5 by 5 m with ore definition based on grade control sampling of around 7.5 by 10 m by 1.25 m (cross strike, strike, vertical). These parameters reflect Wia's conceptual development options for the Project. • The underground MRE reflects long hole sublevel open stoping, with a minimum mining width of around 2.5 m.
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.*	• Metallurgical test work conducted on two bulk samples composited from RC rejects fresh sulphide material suggest: - Standard process by gravity recovery and direct cyanidation leaching achieved 92% gold recoveries - Fast leaching kinetics been with the majority of gold leaching in 2-4 hours - Latest metallurgical assessment completed as part of the DFS work has validated the recoveries achieved on these samples.
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.*	• Study work completed as part of the DFS indicates that there are unlikely to be any specific environmental issues that would preclude potential eventual economic extraction.
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 bulk density for bulk material must have been measured by methods that*	• Bulk densities of 2.63 and 2.71 t/bcm were assigned to weathered and fresh mineralisation respectively on the basis of 3,349 wax-coated immersion measurements performed by Wia on oven dried diamond core samples.

Criteria	JORC Code explanation	Commentary
	adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit. • *Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.*	
Classification	• *The basis for the classification of the Mineral Resources into varying confidence categories.* • *Whether appropriate account has been taken of all relevant factors (ie 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).* • *Whether the result appropriately reflects the Competent Person's view of the deposit.*	• Classifications assigned Mineral Resources reflect confidence in the reliability of the informing data, interpreted mineralisation continuity and sample spacing. • Open pit model panels were primarily classified as Indicated and Inferred by estimation search pass and cross-sectional polygons outlining areas of relatively consistently spaced drilling. Comparatively few panels were subsequently re-classified to give a consistent distribution of categories. The classification approach assigns estimates for mineralised domain panels tested by drilling spaced at generally around 50 m, generally extrapolated to around 25 m from drilling as Indicated. Inferred estimates are generally based on 100 m spaced drilling, generally extrapolated to around 50 m from drilling, with comparatively rare extrapolation in areas of greater continuity to a maximum of around 100 m from drilling. Around 90% of the combined estimates are within 50 m of drilling and 99% are within around 70 m. • The underground estimates are classified as Inferred reflecting the confidence in the interpretation of mineralisation continuity relative to drill spacing. The estimates are based on variably spaced drilling, with intercepts with the MRE volume approximating 100 m spacing and are extrapolated to generally a maximum of around 50 m from drilling. • The classifications take appropriate account all relevant factors and reflect each Competent Person's view of the deposit and informing data.
Audits or reviews	• *The results of any audits or reviews of Mineral Resource estimates.*	• The resource estimates have been reviewed by Wia geologists and are considered to appropriately reflect the mineralisation and drilling data and their understanding of the mineralisation.
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 statement should specify whether it relates to global or local estimates, and,*	• Confidence in the relative accuracy of the estimates is reflected by the classification of estimates as Indicated and inferred.

Criteria	JORC Code explanation	Commentary
	if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. • *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	

Section 4 Estimation and Reporting of ore reserves

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

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 Mineral Resource estimate, for the Kokoseb Gold Project (Project) as detailed in ASX release dated 10 August 2026 titled "Kokoseb Mineral Resource Estimate Increases to 3.78Moz Au" and available at www.wiagold.com.au, has been used for Ore Reserve estimation for the Project. The Mineral Resource has been reported in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC 2012). The Open Pit Mineral Resource estimate at a cut-off grade of 0.5 g/t reported 105 Mt at 0.96 g/t (and at a cut-off grade of 0.4 g/t reported 135 Mt at 0.84 g/t). The Mineral Resource Estimate for the Project is reported inclusive of the Ore Reserves.
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.*	The Ore Reserve Estimate was completed by Mr Jake Fitzsimons, FAusIMM. Mr Fitzsimons is employed by Orelogy Consulting and has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the mining activity being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the JORC Code. Mr Christopher Jones, a Principal Consult employed by Orelogy Consulting conducted a site visit to the Project area, on behalf of Mr Fitzsimons. Observations from the site visit held have been factored into the estimation of Ore Reserves.
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 Pre- Feasibility Study 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 Mineral Resource has been converted to an Ore Reserve through the completion of a Feasibility Level Mining Study (FS). The mine plan is considered technically achievable and involves the application of conventional technology and open pit mining methods widely utilised in Namibia. Financial modelling shows the project to be economically viable using current assumptions on gold price and quoted costing. Material Modifying Factors that relate to mining and processing of ore and recovery of gold have been considered for the Ore Reserve Estimate. The Study was compiled by WIA Gold with input from: • WIA Gold (Geology). • Dhamana Consulting (Project Execution, Strategy & Operations Management). • Matrix Resource Consultants (Mineral Resources). • Peter O'Bryan & Associates (Mine Geotechnical). • Orelogy Consulting (Mine Planning and Ore Reserve). • Maelgwyn Mineral Services Africa (Metallurgical Test Work). • Dhamana Consulting (Metallurgical Test Work Management and Interpretation). • Senet (Process Plant Design and Costing). • Knight Piesold Namibia (Non-Process Infrastructure Design and Costing). • Knight Piesold Namibia (Tailings Storage Facility). • SLR Namibia (Hydrogeology). • GSFA Consulting Engineers (Power Supply and Costing).

Criteria	JORC Code explanation	Commentary
		• Environmental Compliance Consultancy Namibia (Environment, Social, Approvals).
Cut-off parameters	• *The basis of the cut-off grade(s) or quality parameters applied*	Break-even cut-off grades were determined by considering: • Gold price, net of refining charge and royalties, of US$2,589.25/oz. • Achievable gold recovery from ore processing averaging 90.5% (marginally higher than the life of mine mining inventory gold recovery of 90.4%, which includes 3.8% of gold in Inferred Mineral Resources). Variable recoveries based on the metallurgical test work and proposed process plant design were derived using the formula: • Recovery = (head_grade – 0.083)/(head_grade)%. • Feasibility Study ore processing costs of $14.58/t for a 5.25 Mtpa processing plant. A minimum diluted cut-off grade of 0.4 g/t was applied.
Mining factors or assumptions	• *The method and assumptions used as reported in the Pre-Feasibility or Feasibility Study 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.*	The Open Pit Ore Reserve Estimate is underpinned by mine plans that deliver ore for processing on site to produce gold for sale. The mine planning activities included to derive the Ore Reserve were: • Additional dilution modelling applied to the LMIK grade to account for blasting movement and selective mining operation. • Open pit optimisation and selection of a viable economic shell using $2,600/oz gold price as the basis for design. Pit shells were selected based on cashflow, geotechnical constraints and operational considerations. • Development of ultimate pit designs split into practical internal stages suitable for the size of the mining equipment and batter-berm parameters based on recommendations provided by an external geotechnical consultant. • Mine scheduling included balancing value objectives with practical considerations. • Mining cost estimation based on submissions from experienced contract mining service providers. • Conventional open pit mining method using excavators and rigid dump trucks was selected as the most appropriate mining method. • The mining method and grade control practises to be employed at the Project are aimed at mining the ore zones selectively using backhoe configured excavators on a 2.5 m flitch to minimise dilution and ore loss. Blasting of all rock was assessed on 10 m benches. • Final pits were split into stages with each stage designed with access using dual lane ramps except for the final two benches where single lanes were adopted. The mine design used minimum mining width of 30 m for the base of pits. The stage designs targeted a minimum mining width of 50 m as a practical mining limit without compromising operability. • A geotechnical assessment was completed by Peter O'Bryan and Associates supported by 25 diamond core holes drilled between 2022 and 2025 and a suite of geotechnical test work on samples selected by the geotechnical engineer. Recommendations have been used during detailed mine design. The Mineral Resource Model was used during the pit optimisation process using Measured and Indicated resources only with physical,

Criteria	JORC Code explanation	Commentary
		technical and economic parameters applied to the Mineral Resource Model generating an "ideal" open pit excavation geometry which was carried through to detailed mine design. Only diluted blocks with a positive value and grade above 0.4 g/t were identified as Ore during pit optimisation. Ore loss (mining recovery) and dilution was modelled during the conversion of the Resource Model to a Mining Model taking into account inherent dilution from the LMIK resource estimation method, ore width, orebody dip, the selective mining unit and the grade of the diluent material by applying a 0.5 m mixing zone at the boundaries between or and waste. The ore losses and dilution applied to the LMIK model reported at zero grade were: • Indicated Ore– 1% extra ore loss and 3% extra dilution. The LMIK ore loss and dilution are not quantifiable. No Inferred Mineral Resources have been included in the Ore Reserve Estimate. Inferred Mineral Resources were treated as waste and assigned no economic value.
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 test work 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 test work 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 proposed process flowsheet includes a single stage crushing, Semi Autogenous Grinding, Ball Milling and Pebble Crushing (SABC) comminution circuit with conventional gravity recovery followed by thickening, pre-oxidation with oxygen, cyanide leaching and carbon-in-pulp (CIP) process. Tailings are treated using the SO_2/Air process for cyanide destruction prior to thickening and pressure filtration for water recovery. The process flowsheet uses a nominal grind size of 80% passing 63 microns and a total leaching time of 48 hours to maximise gold extraction. The metallurgical process proposed is commonly used in gold process plants in Namibia and internationally. A variable metallurgical gold recovery, averaging 90.5% over the proposed LOM for the Ore Reserves, is based on a constant tail from leaching and adsorption of 0.083 g/t (using a grind size of 80% passing 63 microns and 48-hours leach residence time). This recovery model was developed based on test work completed for the FS in 2025 and 2026. Testing was completed in duplicate on 15 variability samples covering a range of head grades, lithologies, degree of weathering and spatially within the proposed pits with no statistically significant relationship between the various factors identified. The results of the variability test work were replicated on tests carried out on composite samples. All samples tested were from material considered in the FS mill feed and provided a basis for engineering parameters to design the proposed processing plant, and economic evaluation. No minerals are defined by a specification.
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.*	The following environmental assessments have been undertaken across the Project exploration lease: • Terrestrial Ecology – Assess biodiversity, habitat and ecosystem services, identifying any species of concern. • Hydrology – Assess potential flooding and determine drainage and sediment management requirements. • Hydrogeology – Asses potential groundwater contamination from aspects such as the pit, TSF and WRDs. • Air Quality – Assess potential emissions and dust impacts. • Noise – Assess the potential impact of noise on the environment. • Visual and Sense of Place – Assess the impact on the environment from a visual perspective. • Soil and Land Use – Assess existing land use and the quality and quantity of soils for rehabilitation.

Criteria	JORC Code explanation	Commentary
		• Traffic – Assess impact from the Project on traffic in the area influence by the Project. • Heritage and Culture – Identify and assess any areas of cultural significant that may be impacted by the Project. • Visual and Tourism – Assess any impact on potential tourism in the Project area. • Social and Economic – Assess the impact on communities in the Project area or that may be impacted by the Project activities. • Geochemical – Assess waste rock, ore and tailings for mineralogical composition and potential acid mine drainage or neutral metals leaching. • Climate Change – Assess the potential impact from climate change on the Project along with the Project's greenhouse gas emissions footprint. • Vibration – Assess the impact of blasting on receptors in the Project area. These studies, the associated impacts and high-level management plans were incorporated into an Environmental and Social Impact Assessment which was submitted to regulators in March 2026. The Company is of the reasonable opinion that impacts associated with mine development of the Project can be mitigated and minimised through the implementation of management measures identified and these are likely to be acceptable to regulators with respect to obtaining requisite project approvals. There is no evidence in any of the test work to suggest any acid mine drainage potential from the materials associated with the Project.
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.*	The Project site is located approximately 320 km by road north-west from national capital of Windhoek and is accessed via sealed and unsealed public roads (with the majority sealed). The Project site is also approximately 280 km by road north-east of Walvis Bay and accessible via sealed and unsealed public roads (with the majority sealed). The port of Walvis Bay is available for importation of bulk reagents and other goods required for operation of the Project. The FS assumes that the workforce will be accommodated in an accommodation village at site, other than employees drawn from the local towns of Okombahe, Omaruru and Uis. The FS has explored in detail the connection of the Project to the Namibian grid with a power supply application to NamPower submitted in 2025 and negotiations underway for the connection via a new 132 kV overhead power line from the Omburu substation, approximately 100 km from the Project site. Water supply investigations have been extensive and have included collaboration with NamWater. Hydrogeological investigations have identified sufficient groundwater for the Project at the Omararu Alluvial Plains (OmAP) to the south-west of the Project area. Detailed geophysics and water bore drilling and pump testing has been completed with additional drilling and pump testing underway. Other infrastructure will include a ROM pad, pit dewatering infrastructure, water supply bore field, mine services area (including offices, workshop, and stores), magazine, process plant offices and stores, wastewater treatment facility, tailings storage facility, power line, village and site roads. A mining license application was submitted in October 2025 which provides sufficient area to accommodate all infrastructure required for the Project.

Criteria	JORC Code explanation	Commentary
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.* • *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.*	The FS level capital cost estimate in 2026 USD prices has been developed by a range of consultants based on preliminary designs, mechanical equipment lists and material take-offs with vendor pricing for large mechanical items and in-house Engineering estimates for process and non-process infrastructure in accordance with AACE Class 3 estimate and are considered to be estimated at a +/-15% accuracy consistent with a FS. The capital cost in the various areas were estimated by: • Knight Piesold - Bulk earthworks (including construction pads, roads and surface water management), non-process infrastructure, TSF, borefields and water supply pipelines and village. • Senet – Process plant including crushing, grinding, gravity, leach, CIP circuits, tailings filtration and associated process service infrastructure. • SLR – Water bore drilling. • GSFA Consulting Engineers – Power supply. • Dhamana Consulting – Owner's project costs and compilation of the overall capital cost estimate. • Orelogy – Contractor mobilisation, site establishment and pre-production mining costs. Processing and General and Administration operating cost estimates have been developed by Dhamana Consulting based on metallurgical test work results, process plant design, quotations for reagents and consumables, HR Study to determine personnel labour rates, published power pricing from NamPower, quoted rates for various G&A aspects and typical allowances. The FS mining cost estimate prepared by Orelogy was supported by budget pricing obtained from the open pit mining contractors which and included diesel fuel consumption estimates at a net price of $1.25/L. Mine owner overhead operating costs have been estimated based on 2026 labour market estimates plus site based allowances, flights, accommodation, and oncosts. All operating costs are considered to be estimated at a +/-15% accuracy consistent with a FS. All revenue and cost calculations have been done using the native currency and converted to United States Dollars using consensus forward forecasts for 2027/28 when the Project is anticipated to be constructed. Transportation and refining charges of US$4.31/oz are based on budget quotations received from service providers. An allowance has been made for the 3% Royalty and 1% export levy. There are no private royalties payable.
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.*	Ore production and gold recovery estimates for revenue calculations were based on detailed mine designs, mine schedules, mining factors and cost estimates for mining and processing. A gold price of US$2,600 per ounce has been used for economic analysis.

Criteria	JORC Code explanation	Commentary
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.*	There is a transparent quoted market for the sale of gold. No industrial minerals have been considered.
Economic	• *The inputs to the economic analysis to produce the net present value (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 Ore Reserve Estimate has been evaluated through a standard financial model. All operating and capital costs as well as revenue factors were included in the financial model. This process has demonstrated the Ore Reserve Estimate has a positive economic return. Sensitivity analysis has been carried out with significant assumptions and inputs varied by +/- 25%, which is consistent with the order of accuracy of FS level assumptions and inputs. The Ore Reserve Estimate is most sensitive to mined grade, gold recovery and gold price.
Social	• *The status of agreements with key stakeholders and matters leading to social license to operate.*	The Project is a greenfield mine site and has good working relationships with neighbouring stakeholders. There are no existing or pending claims over the Project site. Appropriate stand-off distances have been applied to environmentally or culturally sensitive exclusion zones.
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 Pre-Feasibility or Feasibility study. Highlight and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.*	The Company has long-standing granted exploration licences extending over the Kokoseb deposit where Ore Reserves have been defined. Application has been made in October 2025 for the granting of a Mining Lease over the Project and the Company has no reason to believe this will not be granted in the second half of 2026. There are no likely identified naturally occurring risks that may affect the Ore Reserve Estimate area. There are reasonable grounds to expect that all necessary Government approvals will be received within standard timeframes after lodgement of requisite applications.
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 classification of the Ore Reserve Estimate has been carried out and reported in accordance with the 2012 Edition of the JORC Code. The Ore Reserve Estimate reflects the Competent Person's view of the deposit. The Probable Ore Reserve is based on that portion of Indicated Mineral Resource within the mine designs that may be economically extracted and includes allowance for dilution and ore loss.

Criteria	JORC Code explanation	Commentary
Audits or reviews	• *The results of any audits or reviews of Ore Reserve estimates.*	Peer review on the Ore Reserve Estimate has been completed internally by Orelogy Consulting. SRK Consulting (UK) have also undertaken an Independent Technical review on behalf of potential lenders to the Project.
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 production data, where available.*	The Mineral Resource Estimate and hence the Ore Reserve Estimate relate to global estimates. No production or reconciliation data is yet available for comparison. It is noted that Ore Reserve Estimates are an estimation only and subject to numerous variables common to mining projects and/or operations. It is however, in the opinion of the Competent Person that at the time of reporting, economic extraction of the Ore Reserve estimate can be reasonably justified. The mine design, mine schedule and financial model on which the Ore Reserve Estimate is based have been completed to a Feasibility Study standard with a corresponding level of confidence. Assumed ore treatment recoveries are supported by metallurgical testwork. It is in the opinion of the Competent Person that cost assumptions and modifying factors applied in the estimation of the Ore Reserve are reasonable. Relevant contractor costs are based on budget level pricing supplied by suitably qualified mining contractors. There are reasonable grounds to expect that all primary and secondary mining approvals will be received within the timeframes required for project development.

KOKOSEB GOLD PROJECT

Definitive Feasibility Study

Rev 0

7 August 2026

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ASX:WIA

Forward Looking Statements & Important Notice

This Definitive Feasibility Study (DFS) Report ("Report") contains certain forward-looking statements and projections regarding: estimated, resources and reserves; planned production and operating costs profiles; planned capital requirements; and planned strategies and corporate objectives.

Forward-looking statements are subject to a variety of known unknown risks, uncertainties and other factors that could cause actual events or results to materially differ from those reflected in the forward- looking statements, including, without limitation: inherent uncertainties and risks associated with mineral exploration; uncertainties related to the availability of future financing necessary to undertake activities on Wia Gold Limited's (Wia) properties; uncertainties related to the possible recalculation of, or reduction in Wia's mineral resources; uncertainties related to the outcome of studies; uncertainties relating to fluctuations in gold prices; the risk that Wia's title to its properties could be challenged; risks related to Wia's ability to attract and retain qualified personnel, uncertainties related to general economic and global financial conditions; uncertainties related to the competitiveness of the industry; risk associated with Wia being subject to government regulation, including changes in regulation; risks associated with Wia being subject to environmental laws and regulations, including a change in regulation; risks associated with Wia's need for governmental licenses, permits and approvals; uninsured risks and hazards; risk related to the integration of businesses and assets acquired by Wia; risk associated with Wia having no history of earnings or production revenue; risks associated with fluctuation in foreign exchange rates; risks related to default by joint venture parties (if any), contractors and agents, inherent risks associated with litigation; risk associated with potential conflicts of interest; risk related to effecting service or process on directors resident in foreign countries; uncertainties related to Wia's limited operating history; risks related to Wia's lack of a dividend history; risks relating to short term investments; and uncertainties related to fluctuations in Wia's share price.

The forward-looking statements contained in this Report are based on the assumptions, beliefs, expectations and opinions of management as of the date hereof and which Wia believes are reasonable in the circumstances, but no assurance can be given that these expectations will prove to be correct. These assumptions include but are not limited to that Wia's exploration of its properties and other activities will be in accordance with Wia's public statements and stated goals, that there will be no material adverse change affecting Wia or its properties, anticipated costs and timing for Wia's activities and such other assumptions as set out herein. Such forward looking statements/projections are estimates for discussion purposes only and should not be relied upon. They are not guarantees of future performance and involve known and unknown risks, uncertainties and other factors many of which are beyond the control of the Company. The forward-looking statements/projections are inherently uncertain and may therefore differ materially from results ultimately achieved.

Wia does not make any representations, provides no warranties concerning the accuracy of the forward-looking statements and disclaims any obligation to update or revise any forward-looking statements based on new information, future events or otherwise except to the extent required by applicable laws. Except for statutory liability which cannot be excluded, each of

Wia and its related bodies corporate and their respective officers, employees and advisers disclaim any responsibility for the accuracy or completeness of the material contained in this Report and exclude all liability whatsoever (including in negligence) for any loss or damage (direct or indirect) which may be suffered by any person as a consequence of any information in this announcement or any error in it or omission from it.

Mineral Resource Estimate

The Mineral Resource Estimate referred to in this Report was first disclosed in accordance with the requirements of ASX Listing Rule 5.8 in the Company's ASX announcement dated 10 August 2026, titled "Kokoseb Mineral Resource Estimate Increases to 3.78Moz Au". The Company confirms that it is not aware of any new information or data that materially affects the Mineral Resource Estimate contained in this Report and all material assumptions and technical parameters underpinning the estimate in the previous announcement continue to apply and have not materially changed. The announcement is available to view on www.wiagold.com.au.

Competent Persons Statement

The information in this Report that relates to Ore Reserves is based on and fairly represents information compiled by Mr Jake Fitzsimons, BEng (Geological), a Competent Person who is a Fellow of the Australasian Institute of Mining and Metallurgy. Mr Fitzsimons is a full-time employee of Orelogy Consultants Pty Ltd. Mr Fitzsimons 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 JORC Code (2012 Edition). Mr Fitzsimons consents to the inclusion in this Report of the matters based on his information in the form and context in which it appears.

The information in this Report 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 JORC Code (2012 Edition). Mr Watkins consents to the inclusion in this Report of the matters based on his information in the form and context in which it appears.

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ACRONYMS AND ABBREVIATIONS

AACE	Association for the Advancement of Cost Engineering
AASHTO	American Association of State Highway and Transportation Officials
AC	Aircore drill hole
AEP	Average exceedance probability
Ai	Abrasion index
AISC	All in sustaining cost
ALS	ALS Limited
AMD	Acid mine drainage
AMS	Annual maximum series
AMSL	Above mean sea level
ANC	Acid neutralising capacity
ANCOLD	Australian National Committee on Large Dams
AoI	Area of influence
ARI	Annual recurrence interval
ASM	Artisanal scale mining
ASTM	American Society for Testing and Materials
ASX	Australian Securities Exchange
BBWi	Bond ball mill work index
BGL	Below ground level
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
Company	Wia Gold Limited
COTO 2020	Committee of Transport Officials, Standard Specifications for Road and Bridge Works for South African Road Authorities (2020 Edition)
CRM	Certified reference material
CWi	Crushing work index
Damaran	Damaran Exploration Namibia (Pty) Ltd

DCP	Dynamic cone penetration test
DDH	Diamond core drill hole
DFS	Definitive feasibility study
Dhamana Consulting	Dhamana Consulting Pty Ltd
ECC	Certificate of environmental compliance
ECC Ltd	Environmental Compliance Consultancy (Pty) Ltd
EHS	Environmental, health and safety
EIA	Environmental Impact Assessment
EMA	Environmental Management Act
Epangelo	Epangelo Mining Company (Pty) Ltd
EPCM	Engineering, procurement and construction management
EPL	Exclusive prospecting licence
ESIA	Environmental and social impact assessment
ESMP	Environmental and social management plan
ESMS	Environmental and social management system
FEL	Front end loader
G&A	General and administration
GEV	Generalized extreme value
GHG	Greenhouse gas
GISTM	Global Industry Standard for Tailings Management
GSFA	GS Fainsinger and Associates Consulting Engineers
GWMMP	Groundwater monitoring and management plan
HDPE	High density polyethylene
HW	Hanging wall
ICMM	International Council on Mining and Metals
IFC	International Finance Corporation
IPMT	Integrated project management team
IRR	Internal rate of return
JORC	Australasian Joint Ore Reserves Committee
Knight Piesold	Knight Piésold Consulting (Pty) Ltd
LOM	Life of mine
LRP	Livelihood restoration plan
Maelgwyn	Maelgwyn South Africa (Pty) Ltd

Mandarin	Mandarin Investments (Pty) Ltd
Matrix	Matrix Resource Consultants Pty Ltd
mbgl	Metres below (natural) ground level
mbs	Metres below surface
MEFT	Ministry of Environment, Forestry and Tourism
MIA	Mining infrastructure area
MIME	Ministry of Industries, Mines and Energy
MRE	Mineral Resource estimate
MTO	Material take off
NAF	Non acid forming
NAG	Net acid generation
NAPP	Net acid producing potential
NMD	Neutral mine drainage
NPI	Non process infrastructure
NPV	Net present value, with discount rate as a subscript after
OHPL	Overhead power line
OmAP	Omaruru alluvial plain
OP	Open pit
Orelogy	Orelogy Consulting Pty Ltd
PAF	Potential acid forming
PCD	Pollution control dam
POBA	Peter O'Bryan and Associates
PMC	Project management consultant
QAQC	Quality assurance and quality control
RC	Reverse circulation
RCGC	Reverse circulation grade control drill hole
RI	Return interval
RMR	Mean rock mass rating
ROM	Run of mine
SAG	Semi autogenous grinding
Senet	Senet (Pty) Ltd
SG	Specific gravity
SLR	SLR Environmental Consulting (Namibia) (Pty) Ltd
SMC	SAG mill comminution test

SPT	Standard penetration tests
SSP	Shared socio-economic pathways
SWMP	Stormwater management plan
TARP	Trigger, action, response plan
TDS	Total dissolved solids
TOFR	Top of fresh rock
TSF	Tailings storage facility
UCS	Unconfined compressive strength
VIP	Very important person
Wia	Wia Gold Limited
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^2h	Dry solids per metre squared per hour
g	grams
g/t	Grams per tonne
GWh	Gigawatt hours
h/y	Hours per year
ha	Hectares
kt	Thousand tonnes
kg	kilograms
kg/d	Kilograms per day
kg/t	Kilograms per tonne
km	Kilometre
km^2	Square kilometres
koz	Thousand troy ounces
koz/a	Kilo troy 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	Metres per day
m/month	Metres per month
m/s	Metres per second
m^2	Square metres
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
mbs	Metres below surface
mg	Milligrams
mg/L	Milligrams per litre
mm	Millimetres
mRL	Metres reduced level (to assumed vertical datum)
Moz	Million troy ounces
Moz/a	Million troy ounces per annum
MPa	Megapascals
mRL	Metres relative level
Mt	Million tonnes
Mtpa	Million tonnes per annum
MVA	Mega volt ampere
MW	Megawatt
^{o}	Degrees
^{o}C	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 metres squared per hour
t/m^3	tonnes per cubic metres
US$	United States dollar
US$m	Million United States Dollars

1 INTRODUCTION

The Kokoseb Gold Project (the Project) is located in Namibia, approximately 320 km by road north-west of the capital city, Windhoek. Located in the Erongo Region the Project is well supported by existing local infrastructure including power supply, road access and nearby towns. The Project is owned by Wia Gold Limited (Wia or the Company) in a joint venture with Epangelo Mining Company (Pty) Ltd (Epangelo), the state-owned mining company of Namibia.

The Project is underpinned by a Mineral Resource reported in accordance with the JORC Code (2012). The open pit Mineral Resource comprises 105 Mt at 0.96 g/t containing 3.23 Moz of gold, of which 2.01 Moz, or 62% is classified in the Indicated category at a 0.5 g/t cut-off grade, although the open pit Mineral Resource Estimate was developed at a range of cut-off grades. The underground Mineral Resource contains 0.56 Moz of gold, classified as Inferred Mineral Resources at a 0.85 g/t cut-off grade. The underground Mineral Resource is not considered for mining in the definitive feasibility study (DFS).

The DFS considered open pit mining only, a conventional leach and carbon-in-pulp (CIP) processing facility with a nameplate capacity of 5.25 Mtpa, water supply from demonstrated borefields in the area, low-cost power from the Namibian grid, a filtered stack tailings storage facility (TSF) and associated infrastructure.

The DFS has defined an open pit Probable Ore Reserve, reported in accordance with the JORC Code (2012), of 69.8 Mt at 0.87 g/t containing 1.95 Moz of gold at a 0.4 g/t cut-off grade.

A mining inventory was developed for the Project based on the declared Ore Reserves. The mining inventory includes only 4.3% of material and 3.8% of contained gold from Inferred Mineral Resources. 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 target associated with the mining inventory will be realised.

The mining inventory schedule, which excludes Inferred Mineral Resources from mill feed during the first five years of production, supports average gold production of approximately 161,000 oz per annum over the first five years of operation and 150,000 oz per annum over the first 10 years of the 14-year life of mine.

A capital cost estimate for the Project was prepared as an Association for the Advancement of Cost Engineering (AACE) Class 3 estimate, with an expected accuracy level of ±15%. Capital costs were estimated at US$475m. This cost estimate includes execution readiness, early works, pre-production operating costs, construction owners team and project management costs, operational readiness costs and a contingency which was estimated at 9.9%. The estimate excludes sunk costs, corporate costs, Project funding costs and escalation.

At a consensus gold price assumption of US$3,600 per oz, the Project delivers a post-tax net present value at 5% discount ($NPV_{5\%}$) of US$1,214 million and an internal rate of return (IRR) of 41%, achieving payback in 21 months. Financial outcomes improve significantly at the spot gold price of US$4,075. Key project metrics and financial outcomes are included in Table 1.1.

Table 1.1: Key Project Outcomes

	Units	Base Case US$3,600/oz	Spot Price US$4,075/oz
Production			
Mine Life	years	14	
Total Gold Production	koz	1,836	
Average Gold Production			
Years 1 to 5	koz/a	161	
Years 1 to 10	koz/a	150	
Life of Mine	koz/a	131	
Proportion Inferred (contained gold)			
Years 1 to 5	%	Nil	
Life of Mine	%	3.8	
Costs			
Capital Costs	US$m	407.8	
Mining Mobilisation & Pre-production	US$m	26.8	
Contingency	US$m	40.2	
Sustaining Capital Costs	US$m	48.1	
Mine Closure Costs (excluding salvage)	US$m	37.9	
C1 Cash Costs	US$/oz	1,519	
All-in Sustaining Costs (AISC)	US$/oz	1,696	1,715
Financial			
Pre-Tax $NPV_{5\%}$	US$m	1,923	2,493
Pre-Tax IRR	%	51	61
Post-Tax $NPV_{5\%}$	US$m	1,214	1,577
Post-Tax IRR	%	41	49
Post-Tax Payback Period	Months	21	17

Project permitting is well advanced with the Mining Licence application submitted in October 2025 and environmental and social impact assessment (ESIA) submitted in March 2026. Approval of both applications, and the granting of the environmental clearance certificate (ECC) and Mining Licence is expected in the second half of 2026. Preparation of the final ESIAs for off-site infrastructure is currently underway.

Project development will be preceded by a six-month execution readiness phase, commencing upon completion of the DFS. During this period, project execution plans will be finalised, engineering and consulting groups will be engaged, front-end engineering and design (FEED) advanced, long lead and critical path procurement activities initiated, and final site investigations completed. From commencement of main construction, project delivery is anticipated to take approximately 22 months, with first gold production targeted for the fourth quarter of 2028.

2 DEFINITIVE FEASIBILITY STUDY CONTRIBUTORS

A number of independent consultants contributed to the DFS with key consultants and their responsibilities outlined in Table 2.1. These consultants were managed by the Wia executive team, supported 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	MATRIX Resource Consultants Pty Ltd; WIAGOLD
Mining Reserve Estimation, Mine Planning & Design	Orelogy
Mine Geotechnical	PETER O'BRYAN & Associates, Consultants in Mining Geomechanics
Metallurgical Testwork	MAELGWYN Mineral Services Africa; Dhamana consulting
Process Plant Design	SENET, A DRA Global Group Company
Non-Process Infrastructure Design	Knight Piésold Consulting
Tailings Management & Site Geotechnical Engineering	Knight Piésold Consulting
Hydrogeology & Hydrology	Knight Piésold Consulting
Power Supply & Distribution	G.S. Fainsinger & Associates
Capital Cost Estimation	SENET, A DRA Global Group Company; Orelogy; Knight Piésold Consulting; GSFA; Dhamana consulting
Operating Cost Estimation	SENET, A DRA Global Group Company; Orelogy; GSFA; FUTURE/C, Clarity | Sustainability | Ethics; Dhamana consulting; WIAGOLD
Financial Analysis	Infinity Corporate Finance, Partners for Growth
Environmental	ECC Environmental Compliance Consultancy
Mine Closure Estimation	Orelogy; Australian Mining Advisors; Dhamana consulting
Overall Study Management	Dhamana consulting

3 PROJECT DESCRIPTION

3.1 Project Location

Wia is an ASX listed resource company exploring for gold in the Damaran Belt in Namibia, focussing on the planned development of its Kokoseb discovery.

The Kokoseb Gold Project (Kokoseb or the Project) is in north-west Namibia approximately 320 km by road from the capital city, Windhoek, located as shown in Figure 3.1. Kokoseb lies within the Okombahe exploration licence, shown in Figure 3.2.

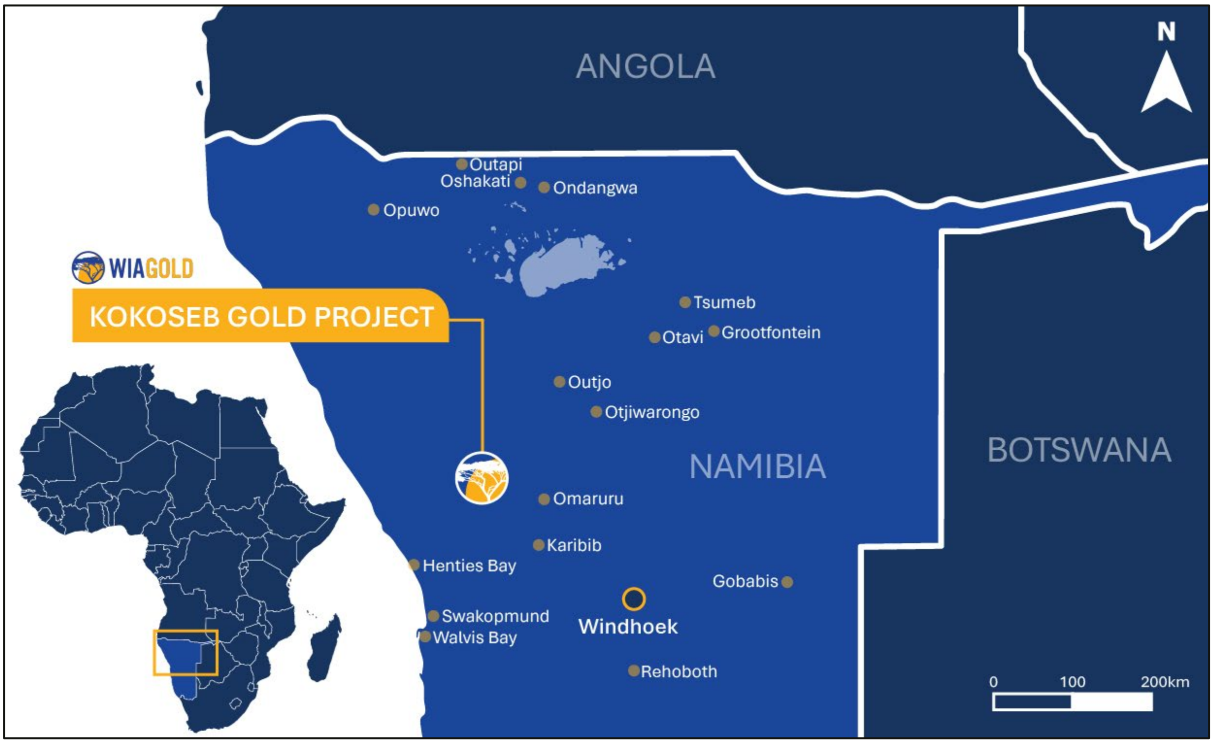


Figure 3.1: Kokoseb Gold Project Location in Namibia

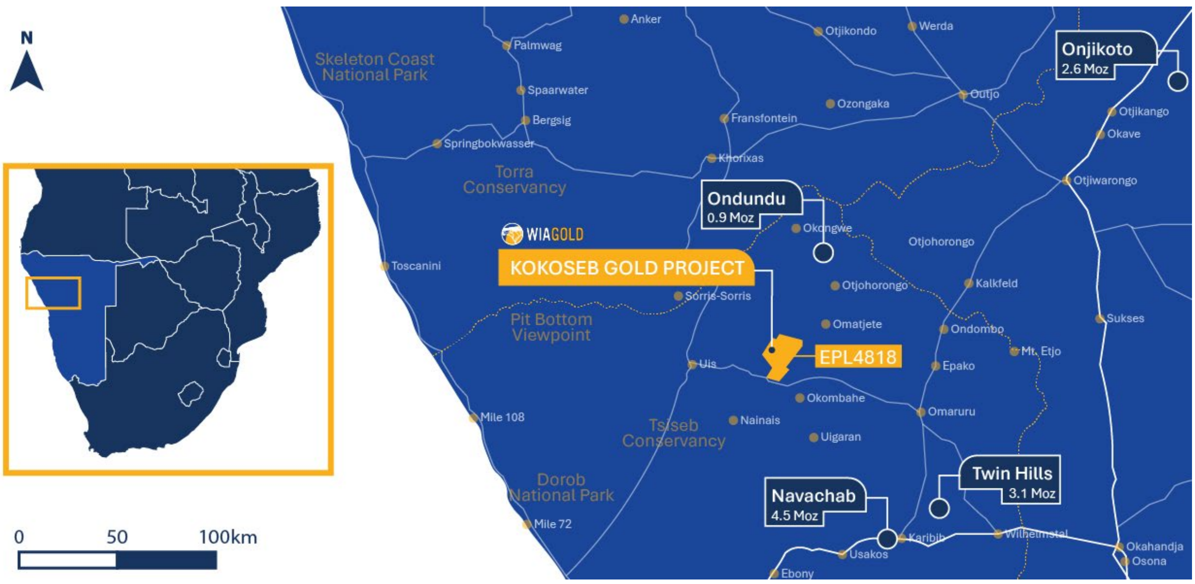


Figure 3.2: Project and Exploration Permit Locations

The Okombahe exploration licence is part of Wia’s broader Damaran Belt tenure in Namibia, which consists of 12 tenements across an area of over 2,700 km^2.

3.2 Ownership

Wia’s landholding in Namibia comprises three separate entities, representing two joint ventures and one wholly owned subsidiary. The ownership of the Okombahe Exclusive Prospecting Licence 4818 (EPL 4818) is held through Mandarin Investments (Pty) Ltd, an 80:20 joint venture between Wia and Epangelo. Epangelo is a Namibian incorporated mining company wholly owned by the Republic of Namibia.

Wia’s 80% beneficial interest in Mandarin Investments is held through its wholly owned Namibian subsidiary, Damaran Exploration Namibia (Pty) Ltd (Damaran). The ownership structure is illustrated in Figure 3.3.

Figure 3.3: Project Ownership Structure

EPL 4818, which hosts the Kokoseb Project, is located between 21°5’35” S and 21°17’43 S and between 15°11’50” E and 15°23’20” E covering an area of 253 km^2. The licence was initially granted to Epangelo on 4 August 2011. On 4 February 2019, Epangelo entered into a joint venture agreement with Damaran. Damaran earned an initial 51% interest through exploration expenditure of A$500,000, a cash payment of A$100,000, and the issue of 5,000,000 shares in Tanga Resources Limited (now Wia Gold Limited). On 22 August 2024, Damaran earned a further 29% interest, increasing its ownership to 80%, by completing the required expenditure commitment of A$1.5m over a two-year period.

Upon completion of the DFS, future Project funding requirements will be contributed by the joint venture partners in proportion to their respective ownership interests. Any partner electing not to contribute its share of funding will be diluted in accordance with the terms of the joint venture agreement.

EPL 4818 is in good standing and is currently valid until 26 August 2026. In line with the three-month notice period required by the Namibian Ministry of Industries, Mines and Energy (MIME) the application for renewal of the exploration licence was submitted on 22 May 2026. Given the substantial investment made in exploration and project development, together with the submission of the Mining Licence application for the Kokoseb Project, the Company expects the renewal application to be approved. The Company remains in regular communication with MIME and has not received any information that would indicate otherwise.

The remaining exploration licences held through the Mandarin Investments (Pty) Ltd joint venture are also located within Namibia's Erongo Region. The Company is following a systematic approach on regional exploration activities across these licences.

3.3 Project Layout and Access

3.3.1 Access

Both Namibia's capital city, Windhoek, and the regional port city of Walvis Bay are serviced by direct international flights from a range of locations, including Johannesburg and Cape Town in South Africa, Addis Ababa in Ethiopia, Frankfurt and Munich in Germany, Zurich in Switzerland and various other locations across Southern Africa including Luanda, Victoria Falls, Gaborone and Maun.

Road access to the Project from Windhoek is via the A1 dual carriageway sealed highway to Okahandja, followed by the B2 sealed highway to Karibib and the C33 sealed road to Omaruru, a total distance of approximately 245 km. From Omaruru, access to site is via the all-weather unsealed C36 road towards Uis to the Okombahe turn-off and then the D3712 road towards Omatjete to Nu-Uis, and the D3714 road towards Uis, which traverse the Project area, with this section from Omaruru covering distance of approximately 90 km.

Alternatively, the Project can be accessed from Walvis Bay via the B2 sealed highway to Swakopmund, the C34 sealed highway to Henties Bay, and the C35 sealed road to Uis, a total distance of approximately 230 km. At the time of reporting, the final 18 km of road upgrades on this route were nearing completion. From Uis access to the Project is via the all-weather unsealed C36 road and the D3714 road towards Nu-Uis, which traverse the Project area, over a distance of approximately 50km.

Dedicated access to the Project will be provided by a purpose-built 13.5 km all-weather unsealed road connecting site to the C36 road. The proposed turn-off is located approximately 80 km from Omaruru and 40 km from Uis.

Current access within the Project area is via existing farm tracks. These tracks are unsealed, and Wia has completed minor upgrade work where required for exploration activities.

3.3.2 Project Layout

The project layout includes:

- Open pit mining area with two separate pits and associated waste rock dump (WRD), high grade stockpile, low-grade stockpile, mineralised waste stockpile and associated haul roads.

- Processing facility co-located with the mining infrastructure area, administration and maintenance facilities.
- Incoming 132 kV power line, high voltage switchyard and NamPower metering switchyard.
- TSF.
- Explosives storage magazine.
- Accommodation village.
- Access roads and security infrastructure.

In addition, various aspects of supporting infrastructure are located remotely from site, including:

- 132 kV grid connection to the NamPower Omburu substation.
- Borefield and water pipeline from the OmAP borefield, including 33 kV overhead power supply.

The facilities are described further in this report, and the site layout is provided in Figure 3.4, while the regional infrastructure layout is provided in Figure 3.5.

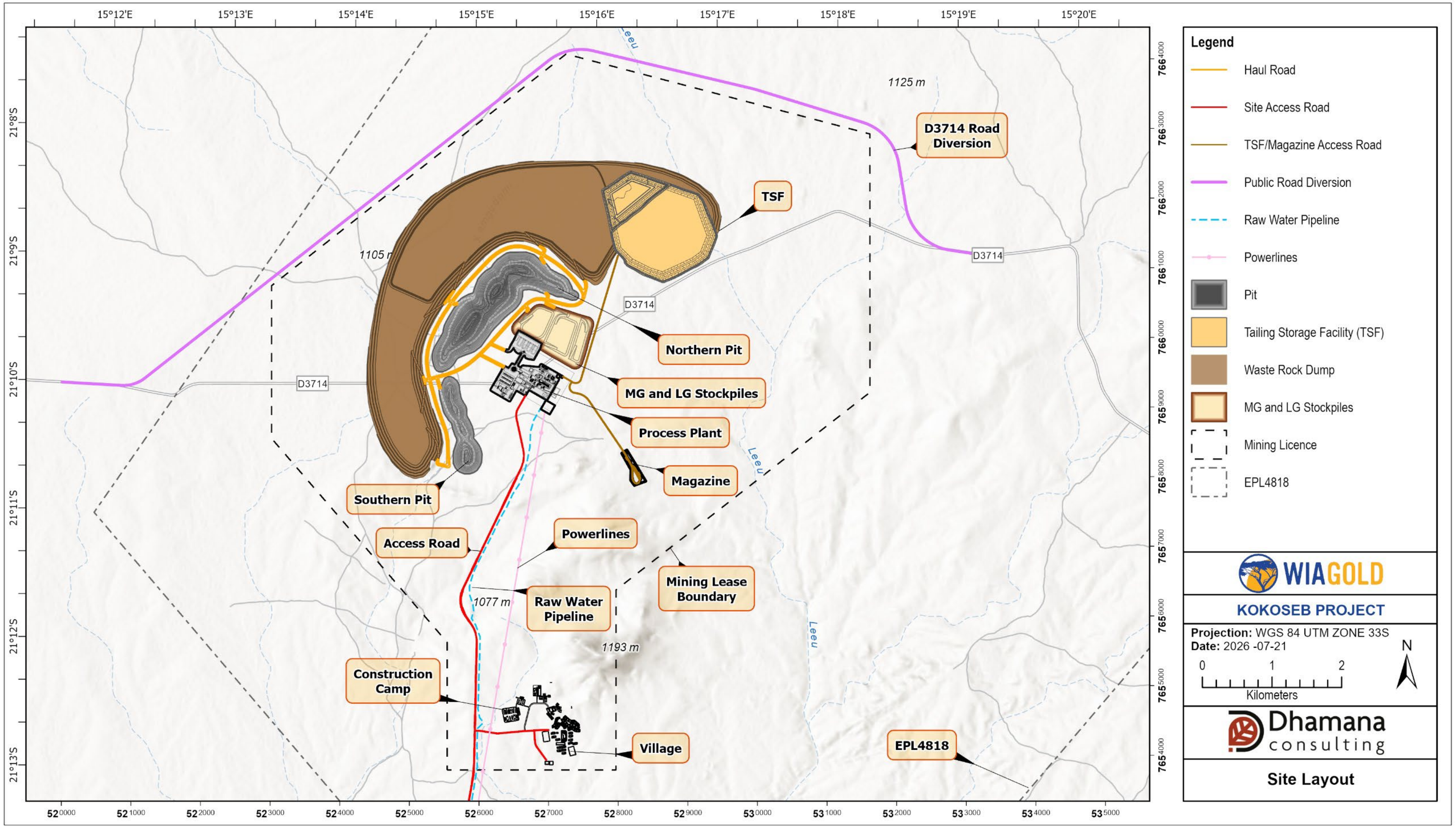


Figure 3.4: Site Layout

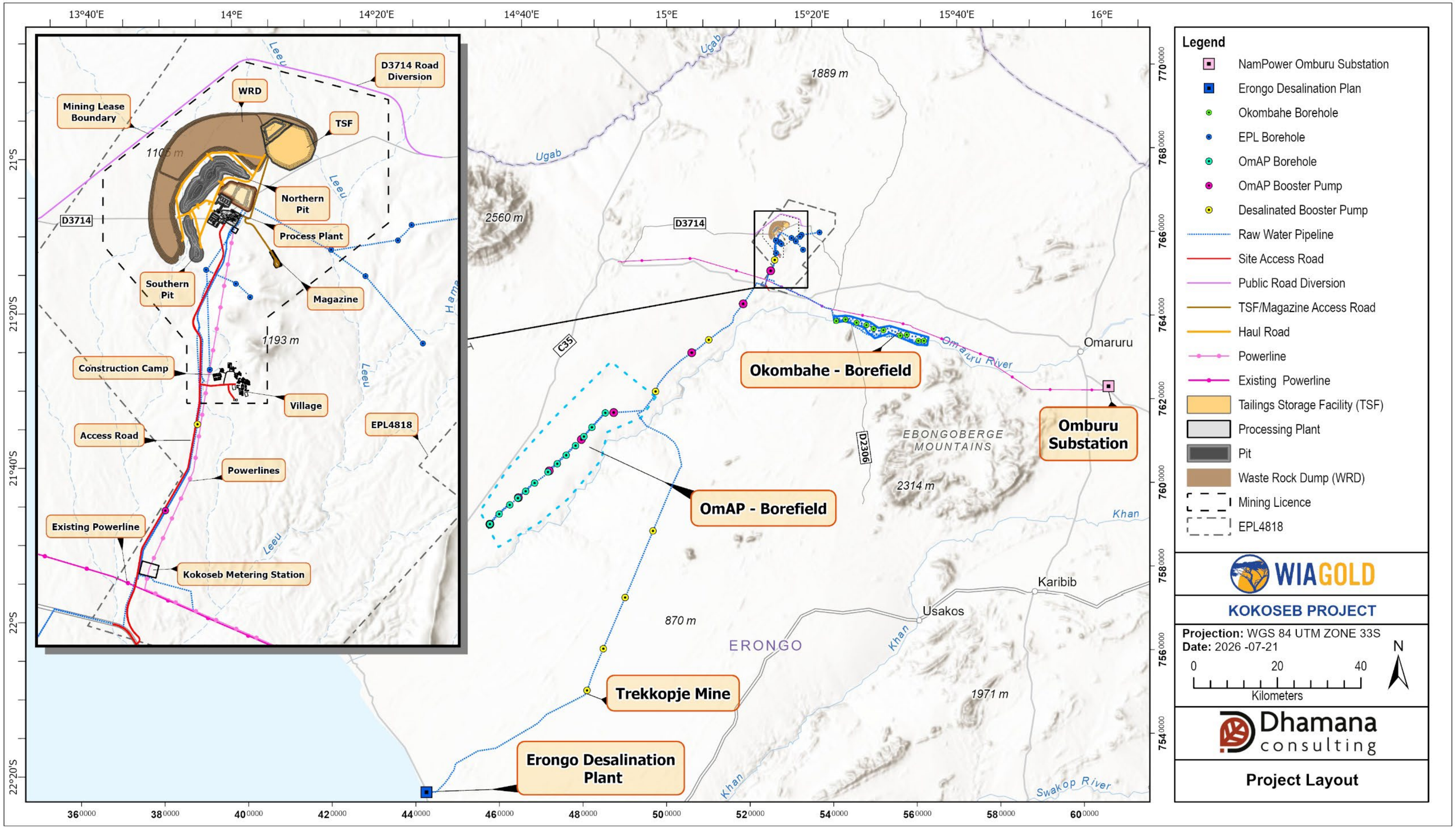


Figure 3.5: Regional Infrastructure Layout

3.4 Climate

The regional climate is considered arid with desert-like conditions. Due to the proximity to the Atlantic coast, the climate is dominated by the South Atlantic Anticyclone, which produces strong coastal winds from the south and blocks the inflow of warm, moist air from the north. Precipitation typically increases with distance from the ocean. In addition to the influence of the South Atlantic Anticyclone, the climate is significantly influenced by the cold Benguela Current, which flows northward along the coast of Namibia and suppresses convective activity, contributing to low and highly variable rainfall.

Rainfall in the broader region is predominantly associated with convective summer systems, and the principal wet season typically occurs between November and March. However, at the Project area and along the adjacent coastal belt, total precipitation is low, and the seasonal signal is relatively weak, with low rainfall amounts and fog events contributing moisture throughout the year.

Average rainfall at the Project is at approximately 140 mm per annum across an average of only 19 days of rain (above 1 mm). Evaporation rates are very high, averaging around 2,500 mm per annum.

The highest average maximum day temperatures, approximately 37°C, occur between March and May, while the lowest average minimum temperatures, around 10°C, occur between June and November. Seasonal variation in temperature is relatively small, with average temperatures ranging from approximately 14°C to 22°C throughout the year.

Prevailing winds are generally moderate, with south-westerly winds dominating during the spring and summer months (September to February), and north-easterly to easterly winds prevailing during the autumn and winter (March to August).

3.5 Land Use

Current land use within and surrounding the Project area is primarily extensive livestock grazing, undertaken under a combination of communal and freehold farming systems. Cattle and small stock production represents the dominant economic activity in the area and is generally conducted at low stocking densities in response to the arid climatic conditions. Subsistence cropping activities occur on a limited scale in areas where groundwater or alluvial water resources are accessible, particularly along ephemeral river systems such as the Omaruru River.

The region also supports low-intensity nature-based tourism associated with nearby features such as the Brandberg Mountain, the Ugab River and surrounding desert landscapes. Tourism infrastructure is limited and generally comprises small lodges, campsites and guided activities.

Overall, land use in the Project area is characterised by low-density, extensive rural activities, supported by a sparse population and dispersed settlements.

3.6 Local Resources and Infrastructure

The Project is located in a remote part of the Erongo Region; however, it is well supported by established infrastructure and services at local, national and regional levels that are relevant to the development and operation of a large-scale mining project.

Omaruru, located approximately 90 km from the Project area, serves as the primary local service centre. The town provides a range of supporting infrastructure and services, including smaller construction contractors, basic fabrication workshops, vehicle maintenance facilities, accommodation and general services. Telecommunications (mobile and data) are generally available in settlement areas but may be intermittent in more remote locations. The surrounding region is predominantly agricultural, and local infrastructure is largely geared towards farming activities, with only limited industrial capability. Grid power is available in Omaruru, Uis and Okombahe distributed from the Omburu substation, located near Omaruru, which forms an important node within Namibia's national electricity network.

At a national level, the Project benefits from access to Namibia's well-developed mining, industrial and logistics hubs. Swakopmund and Walvis Bay, located approximately 250 km south-west of the Project, are connected via established road networks and form the country's principal coastal industrial corridor. Walvis Bay hosts Namibia's only deep-water port, which supports import of heavy equipment, bulk materials and consumables, as well as export logistics. The port is supported by warehousing, freight forwarding, and integrated supply chain infrastructure.

Swakopmund and nearby mining town of Arandis provide access to a broad range of mining-related services, including specialised construction contractors, equipment suppliers, fabrication workshops and maintenance providers linked to the region's established uranium and mining industries.

Windhoek, located approximately 300 km from the Project, provides the highest level of national infrastructure and services. These includes advanced consulting services (engineering, environmental, legal and financial), procurement and supply chain management, project management, and access to national distributors. More specialised fabrication, large-scale construction capability and technical maintenance services are typically sourced from Windhoek or the coastal centres.

Namibia's national road network, electrical grid and telecommunications infrastructure provide reliable connectivity between these national centres and the Project area.

For specialised and large-scale requirements, Namibia is integrated into the broader southern African industrial and mining supply chain. South Africa, in particular, represents a key source of heavy engineering fabrication, process plant construction, mining equipment supply and specialist technical services. Regional hubs such as Johannesburg and Cape Town provide access to advanced consulting expertise, original equipment manufacturers and mining-focused contractors.

Cross-border transport and logistics infrastructure within the Southern African Development Community (SADC) region is well established, with efficient road freight corridors linking Namibia to South Africa and neighbouring countries. Walvis Bay further enhances this connectivity as a regional logistics gateway, facilitating the import of capital equipment and consumables while providing efficient access to international shipping routes and global markets.

3.7 Regulatory Environment

Namibia has a well-established legislative framework governing the mining industry, supported by a stable political system and a long history of mineral development. The sector is principally regulated by the *Minerals (Prospecting and Mining) Act, 1992*, which provides a clear and transparent licensing regime for exploration and mining activities administered by the Ministry of Industry, Mines and Energy. All mineral resources are vested in the State, with rights to prospect and mine granted through a structured and well-defined permitting process.

Environmental management is regulated through the *Environmental Management Act, 2007* and associated environmental impact assessment regulations, which require a formal impact assessment process to be undertaken and an environmental clearance certificate to be issued prior to project commencement.

Labour relations are governed by the *Labour Act, 2007,* providing a formal framework for employment conditions, worker protections and dispute resolution. Collectively, these legislative frameworks provide a comprehensive and comparatively stable regulatory environment compared to many other African jurisdictions.

Namibia's fiscal regime for mining is generally regarded as transparent, predictable and supportive of long-term investment. The regime includes a corporate income tax of 37.5%, applicable to mining companies, royalties typically in the range of approximately 3% to 5% depending on the commodity (3% for gold), and other levies such as the 1% export levies currently applicable to gold. Historically, the fiscal framework has been stable and competitive, contributing to sustained foreign investment in the mining sector, with limited restrictions on foreign ownership under the broader investment.

Overall, the regulatory and fiscal setting can be characterised as mature, transparent and relatively stable, with policy developments generally focused on balancing investor confidence with the delivery of broader national benefits. This has contributed to Namibia's reputation as one of the more attractive mining jurisdictions in Africa, supported by strong institutions and rule of law.

4 GEOLOGY AND MINERAL RESOURCES

4.1 Geology and Geological Interpretation

Kokoseb lies within the Northern Central Zone of the Pan-African Damaran Orogenic Belt around 15 km south of the Otjihorongo Thrust, which separates the Northern Zone from the Northern Central Zone, and about 30 km west of the NNE trending Welwitschia Lineament. The project area is underlain by metasediments of the Arandis, Karibib and Kuiseb Formations of the Swakop Group. Gold mineralisation is found in the Kuiseb Formation metasediments which are extensively intruded by both late syn-tectonic and post tectonic granites, and minor N-S to NNE-SSW trending mafic dykes (Figure 4.1). There is generally moderate to good rock exposure throughout the licence area though the Kuiseb Formation tends to only sub-outcrop and is commonly covered by thin soil, colluvium or pisolitic calcrete up to 2 m thick.

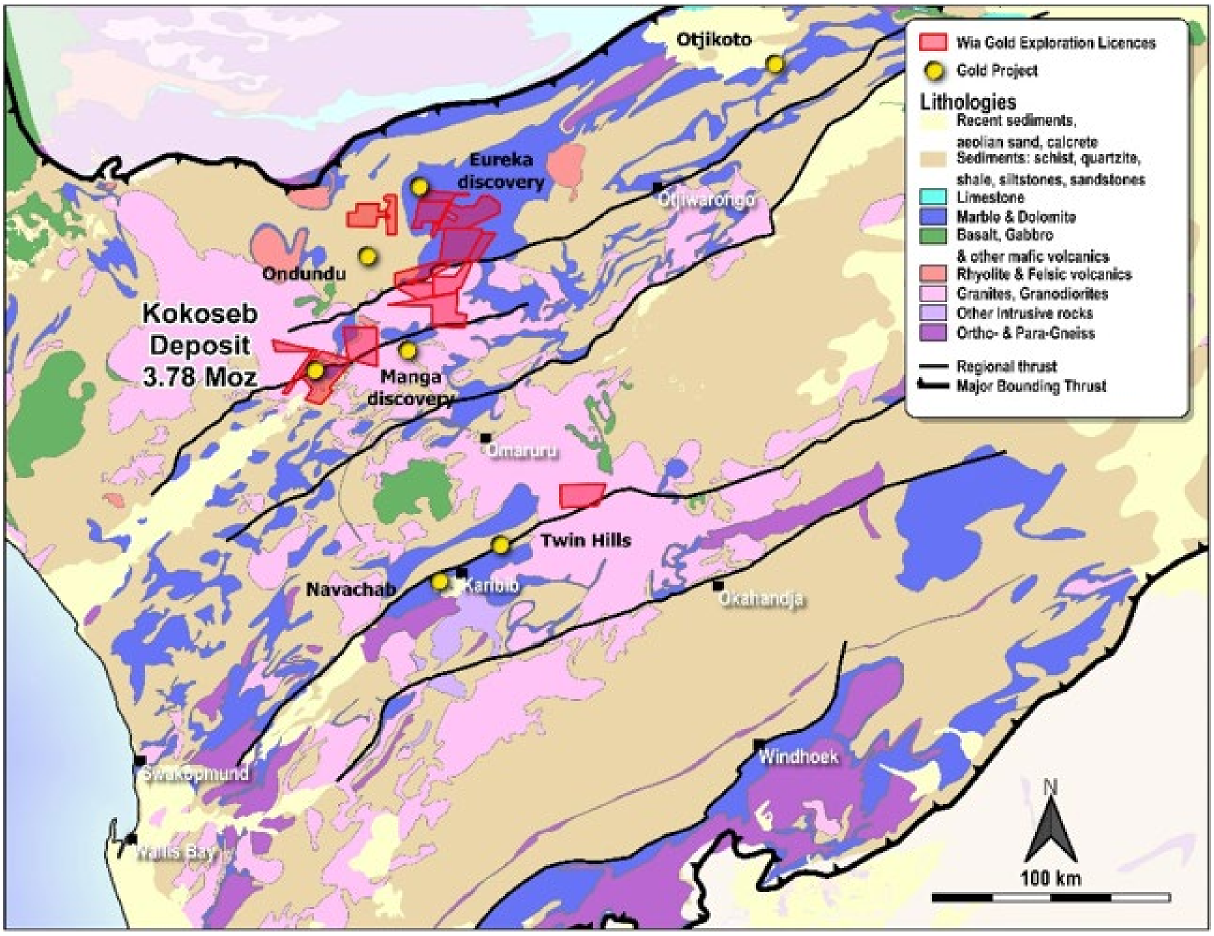


Figure 4.1: Location of Kokoseb and WIA Namibian Projects with Regional Geology

The Arandis Formation consists of alternating schists, calc-silicates (commonly scapolitic) and marble units which core two prominent domal features in the central portions of the Okombahe exploration licence with the easternmost of these domes named the Otjongeama Dome (Figure 4.2). The Arandis Formation is overlain by the Karibib Formation which is dominated by impure marbles and lesser calc-silicates and is capped by the calcitic, graphite bearing marbles of the Arises River Member. The metasediments of the overlying Kuiseb Formation consist mainly of quartz/plagioclase/K-feldspar/biotite schist and biotite schist with minor quartzites and calc-silicates. The schists appear to have undergone local weak partial melting.

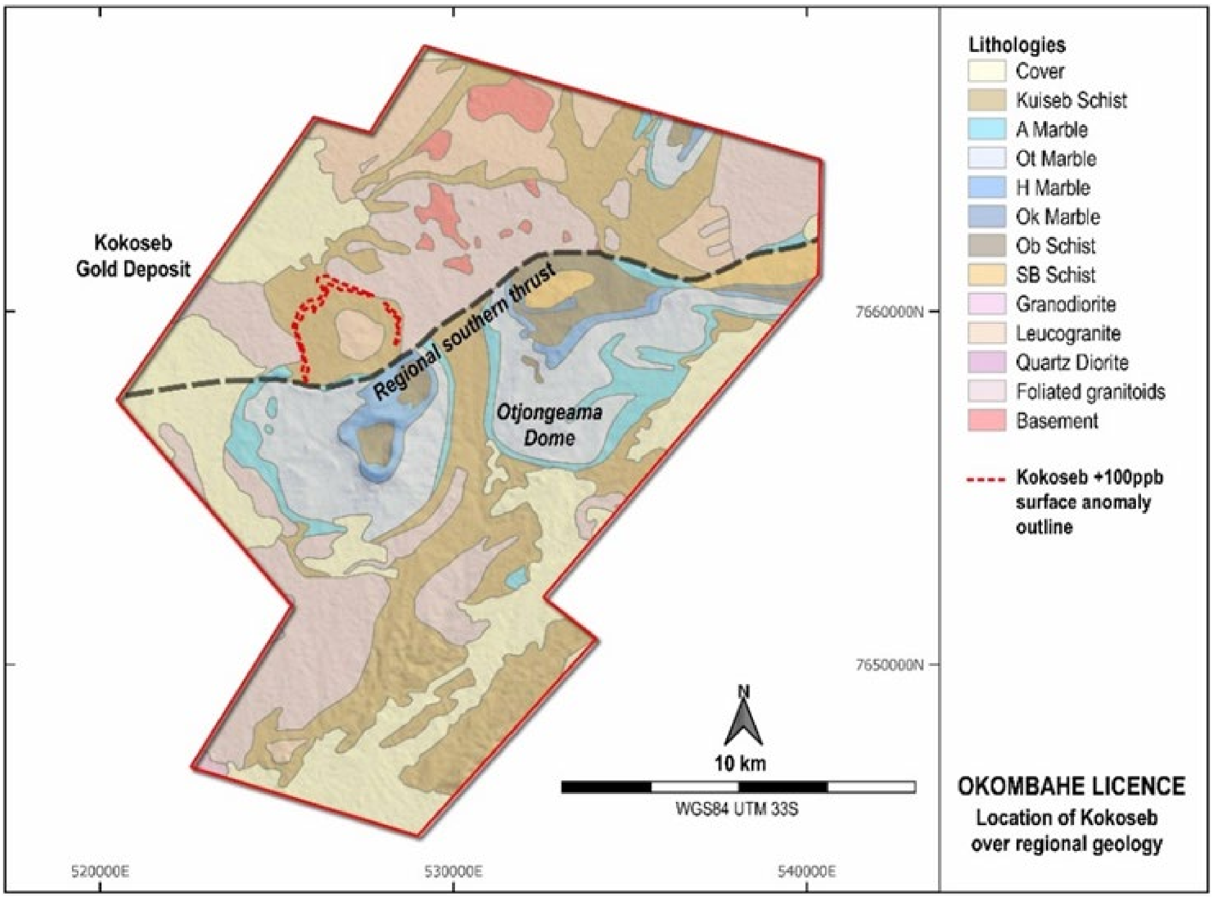


Figure 4.2: Location of the Project in Geological Context of the Okombahe Exploration Licence

Along the southern edge of the Kokoseb Gold Prospect, the domal features cored by Arandis and Karibib Formation rocks are thrusted over the Kuiseb Formation, to the north, along the regional southern thrust, resulting in a prominent marble ridge that marks the southern boundary of currently known mineralisation. This thrusting dissects the domal features in the area and is post D3 in age.

Within the Kokoseb area, the Kuiseb schist forms a domal feature cored by a post tectonic leucogranite, the "Central Granite Pluton", which consists predominantly of medium grained quartz, K-feldspar and plagioclase, with accessory biotite, muscovite, magnetite, garnet and tourmaline. Granite dykes, granitic veinlets and pegmatites cross cutting the Kuiseb schist represent the same granite phase or later granitic phases. Gold mineralisation wraps around this pluton in a roughly arcuate form but seems best developed along the western and northern margins of the Central Pluton. A coarse grained, pre-syn tectonic, porphyritic-feldspar granite encloses the mineralised schists in the west, east and northeast. The Kuiseb schists formation consists mainly of quartz–plagioclase–K-feldspar–biotite schists and biotite-rich schists, with minor quartzite and calc-silicate interbeds. These schists display a clear segregation into mesosome, melanosome and leucosome domains, reflecting partial melting processes, and can be interpreted as migmatitic paragneiss

4.2 Mineralisation

Gold mineralisation, present as native gold grains and lesser silver bearing gold grains, is spatially associated with sulphides dominated by pyrrhotite, löllingite and arsenopyrite in order of abundance. Sulphides manifest as foliation-controlled blebs, stringers and disseminations and löllingite is always spatially associated with arsenopyrite and pyrrhotite where a retrograde reaction rim of arsenopyrite is always developed at the contact between pyrrhotite and löllingite. This contact zone between löllingite and arsenopyrite is typically where gold grains are developed, though they can also occur as partial inclusions within löllingite and rarely within pyrrhotite. Gold is commonly associated with bismuthinite and native Bi mineralisation.

Pyrite is the most common sulphide but does not show any direct association with gold mineralisation.

Mineralisation generally outcrops, with locally a maximum of 1 m to 2 m of barren superficial material. Weathering extends to an average of around 33 m depth.

4.3 Drilling

Drilling at Kokoseb commenced with a diamond campaign undertaken between March and May 2022, followed by reverse circulation (RC) drilling commencing late June 2022. Drilling activities have continued uninterrupted since that time and currently total 191,840 m of drilling, comprising 114,553 m of RC drilling and 77,287 m of diamond drilling. This includes 162 diamond drill holes, 565 RC drill holes and 24 RC pre-collar/diamond tail holes.

Drilling completed since the release of the 2025 Mineral Resource estimate (MRE) has primarily focussed on infill drilling in the open pit area to improve resource confidence and classification, as well as testing deeper mineralisation. These programmes have supported the estimation of a maiden underground Mineral Resource for the Project. Relative to the dataset used for the 2025 MRE, the current database incorporates assay results from an additional 88 diamond drill holes, 143 RC drill holes and eight RC pre-collar/diamond tail holes.

The diamond drilling was completed using dedicated diamond drill rigs. Holes were either cored from surface at HQ or NQ diameter or completed as NQ tails following RC pre-collar. All core was oriented using Reflex digital system.

RC drilling was carried out using dedicated RC drill rigs with face sampling bits of 140mm diameter.

The principal drill spacing supporting the Inferred Mineral Resource category within the Kokoseb deposit is generally 100 m between sections and 50 m between drill holes. The Indicated Mineral Resource category has largely been defined by drilling on an approximate 50 m by 50 m pattern. The maiden underground Inferred Mineral Resource is supported by drilling spaced at approximately 50 m intervals.

4.4 Sampling and Sub-Sampling Techniques

HQ diameter core from diamond drill holes completed from surface was halved using a core saw along the entirety of the metasediment lithologies of the drill holes, leaving unsampled some of the long sections of barren granite. NQ diameter core was sampled full core. Sampling intervals were assigned based on lithological contacts and on any change in alteration or mineralisation style. Core sample length varies between 0.5 m and 2 m.

RC sampling was also undertaken along the entire length of the drill holes, with the exception of long granitic intervals which are known as barren, in a few drill holes. One metre samples were collected from the rig cyclone which directly provided a bagged sample, to avoid any further manipulation. Bulk samples and the split sub-samples were routinely weighed, with sub-samples typically around 2 kg to 4 kg. Duplicate sub-samples were retained for future reference.

Blanks and standards were regularly inserted in the sampling stream to monitor quality control and representativeness of the sampling. Field duplicates were collected at regular intervals for RC drilling.

4.5 Sample Analysis

Drill samples were submitted to ALS laboratories until December 2025 and then to the MSALABS laboratory in Omaruru, Namibia.

4.5.1 ALS Laboratories

Samples were prepared, crushed and pulverised at the Okahandja laboratory in Namibia before shipping to ALS Johannesburg for assay testing.

Samples preparation consists of logging the sample to the laboratory tracking system, fine crushing it to 70% passing 2 mm, splitting 1 kg sample using riffle splitter and pulverizing the 1 kg to 85% passing 75 µm. All phases of preparation include ALS's standard quality control tests.

All drill samples were assayed for gold using the Au-AA24 method. The initial set of core samples, from drill hole KDD001 to drill hole KDD021 were also assayed for multi element using the ME-MS61 method.

Au-AA24 is a fire assay method with atomic absorption (AA) finish, specifically designed for gold analysis. A 50-gram sample pulp is fused with a lead flux to separate precious metals from the gangue material. The resulting lead button is cupelled to remove lead, leaving behind a doré bead containing gold (and silver, if present). This bead is then dissolved in acid and analysed using atomic absorption spectroscopy to determine gold concentration. The method is highly reliable for detecting gold concentrations in the range of 0.005 ppm to 10 ppm.

Higher grade samples have been analysed using the complementary fire assay method Au-AA26, which offers an upper detection limit at 100 ppm, or by the Au-GRA22 fire assay method with gravimetric finish, which allows for an upper detection limit of 10,000 ppm.

4.5.2 MSALABS Laboratory

Sample preparation and assaying were carried out at the MSALABS facility in Omaruru, Namibia, approximately 90 km from the Project area.

Samples preparation consists of logging the sample to the laboratory tracking system, fine crushing it to 70% passing 2 mm, splitting 1 kg sample using riffle splitter and pulverizing the

1 kg to 85% passing 75 µm. All phases of preparation include MSALABS standard quality control tests.

All drill samples were assayed for gold using the FAS-221 method.

FAS-221 is a fire assay method with atomic absorption (AA) finish, specifically designed for gold analysis. A 50 g sample is fused with a lead flux to separate precious metals from the gangue material. The resulting lead button is cupelled to remove lead, leaving behind a doré bead containing gold (and silver, if present). This bead is then dissolved in acid and analysed using atomic absorption spectroscopy to determine gold concentration. The method is highly reliable for detecting gold concentrations in the range of 0.01 to 100 ppm.

Higher grade samples have been analysed using a complementary fire assay method with gravimetric finish, FAS-P004.

4.6 Open Pit Mineral Resources

4.6.1 Mineral Resource Estimation

Open pit Mineral Resources for the Kokoseb deposit were estimated using Multiple Indicator Kriging (MIK) with block support correction to reflect open pit mining selectivity, a method that has been demonstrated to provide reliable estimates of resources recoverable by open pit mining for a wide range of mineralisation styles.

The estimates were based on 2 m down-hole composited gold assay grades from RC and diamond drilling completed up to 31 May 2026. The open pit Mineral Resource is primarily informed by information from RC drilling with composites from this sampling type providing around 73% of the mineralised domain estimation dataset within the pit shell constraining the MRE. Assay results obtained since the 2025 MRE provide around 26% of the mineralised domain dataset within the resource pit shell.

Micromine software was used for data compilation, domain wire framing, coding of composite values and pit optimisation. GS3M was used for resource modelling. The estimation methodology is appropriate for the mineralisation style.

The MIK modelling utilised a set of mineralised domains interpreted which capture composites with gold grades of generally greater than 0.1 g/t. These domains comprise three groups designated as the western, north-east and splay zones. The western zone consists of a southern domain and northwest domain. The continuous north-east zone is subdivided along strike into a comparatively higher-grade north domain, and a lower gold grade east domain. The splay zone represents a subsidiary, east-west trending zone adjacent the northwest domain. These domains are show in Figure 4.3

The mineralised domains have a combined strike length of around 7 km, with average horizontal widths of around 70 m, 100 m, 70 m, 50 m and 60 m for the south, north-west, north, east and splay domains respectively.

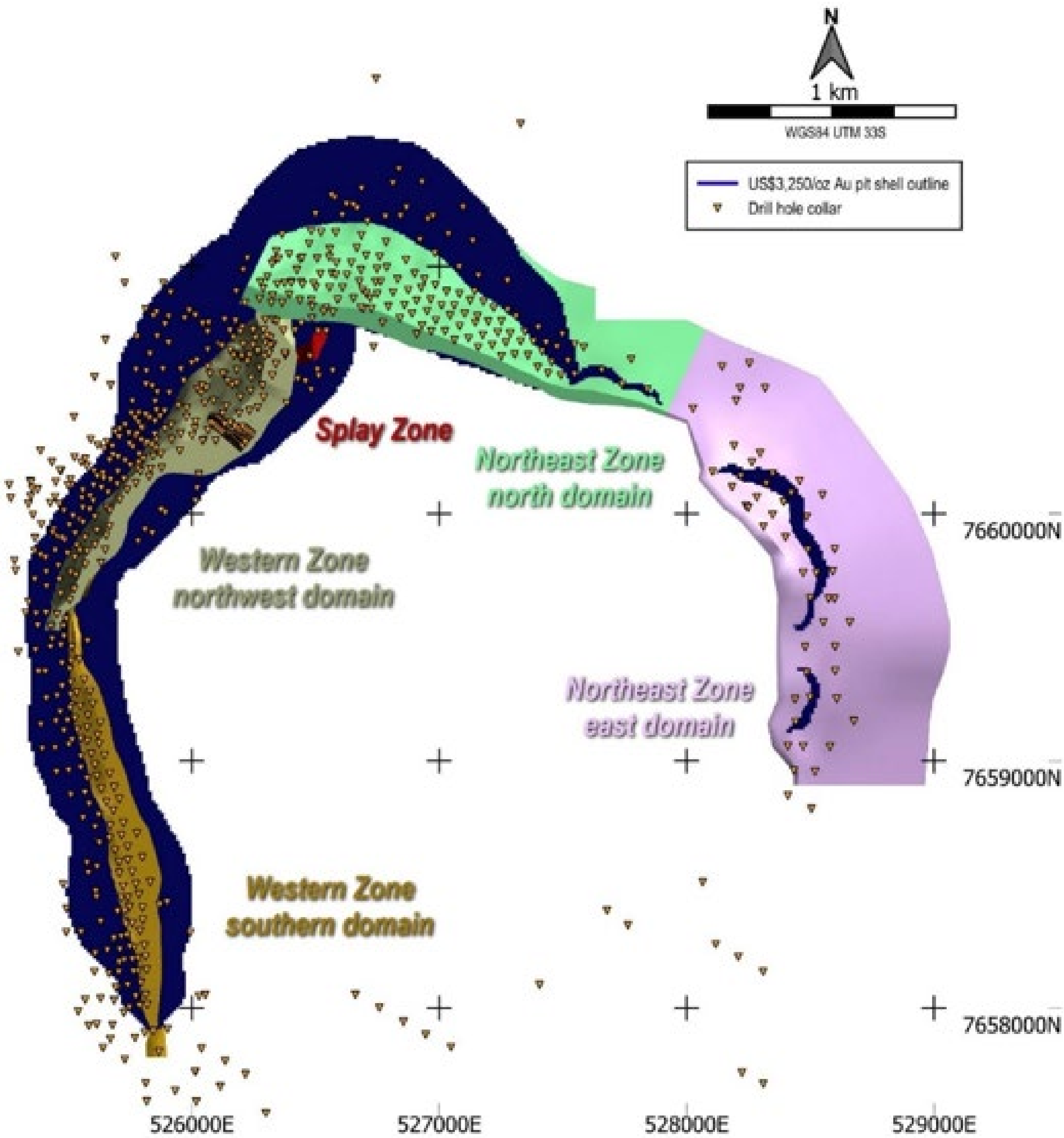


Figure 4.3: Plan View of the Kokoseb Interpreted Mineralised Domains

A surface representing the base of weathering interpreted from drill hole logging, which, within the pit shell constraining the MRE ranges from around 1 m to 100 m depth and averages around 33 m depth was used for density assignment.

Grade continuity was characterised by indicator variograms modelled at 19 indicator thresholds. For determination of variance adjustment factors a variogram was modelled from composite gold grades. The modelled variograms are consistent with geological interpretations and trends shown by composite gold grades. Indicator thresholds and class grades used for MIK modelling were derived from class mean grades with the exception of the upper bin grades of the south, north-west, north and east domains, which were derived from the class means excluding between 1 and 12 high-grade outlier composites with gold grades of greater than

15 g/t, 30 g/t, 8 g/t and 9 g/t respectively. No composites were excluded from the dataset used for grade modelling.

The MIK modelling utilised five progressively relaxed search passes which were selected on the basis of the drill hole spacing and mineralisation trends to inform a reasonably large proportion the mineralised domains while allowing blocks to be estimated by reasonably close data where possible. For grade estimation, the mineralised domains were subdivided into 20 zones of consistent orientation and the search ellipsoids and variograms used for estimation were aligned with local mineralisation trends.

The estimates include a variance adjustment to give estimates of recoverable resources above gold cut-off grades for open pit mining selectivity of around 5 m by 5 m by 5 m with ore definition based on grade control sampling of around 7.5 m by 10 m spacing (cross strike by strike direction). The estimates can be reasonably expected to provide appropriately reliable estimates of potential mining outcomes at the assumed selectivity without application of additional mining dilution or mining recovery factors.

Bulk densities of 2.63 t/bcm and 2.71 t/bcm were assigned to weathered and fresh mineralisation respectively on the basis of 3,439 immersion measurements performed on oven dried wax-coated diamond core samples.

Figure 4.4 shows the drill collars over the block model and grades along with the pit shell developed for assessment of the reasonable prospect of eventual economic extraction (RPEEE) and Figure 4.6 shows the block model in perspective, looking north-west.

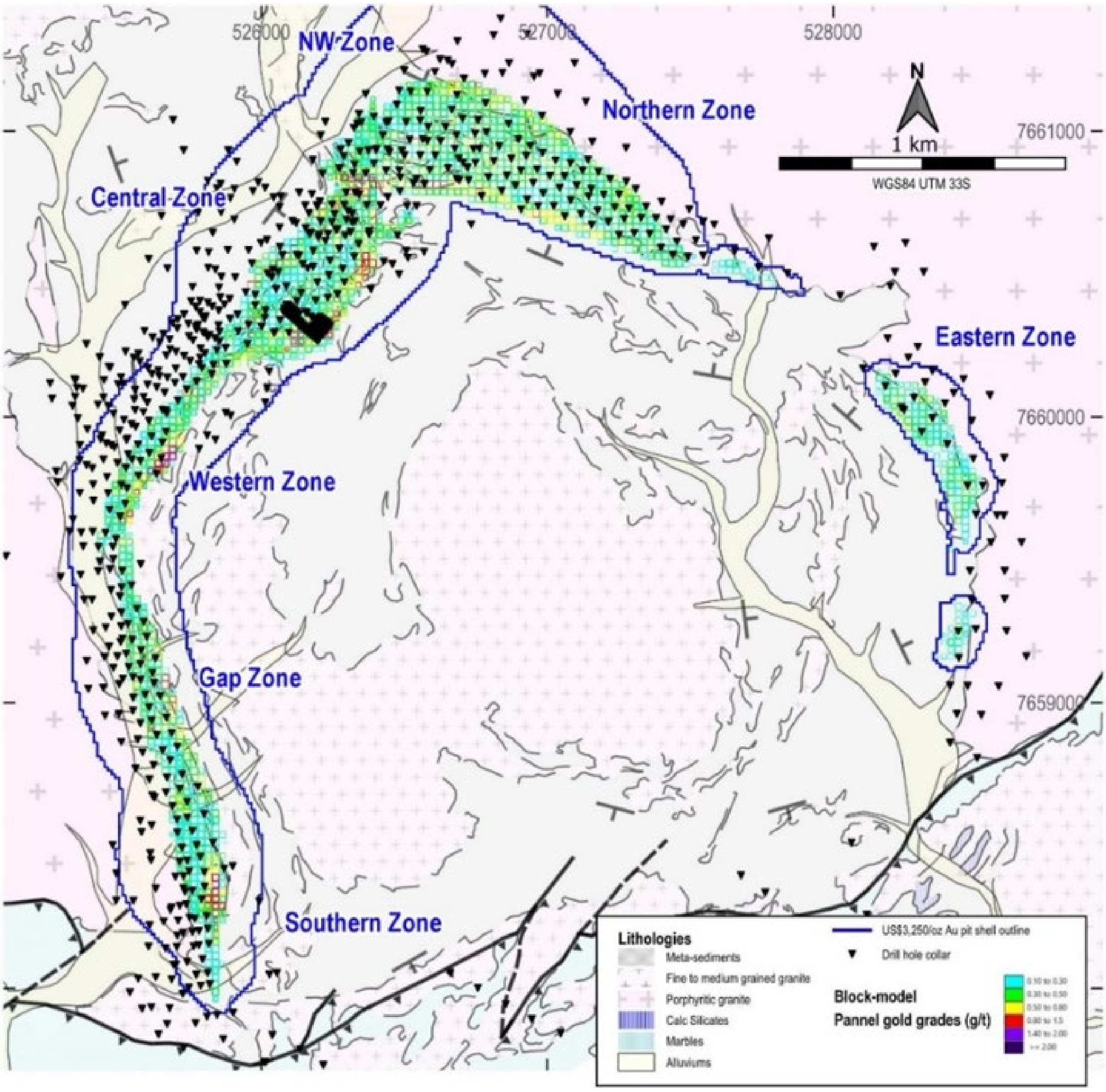


Figure 4.4: Plan View of Kokoseb showing the Drill Hole Collars over the Block Model

4.6.2 Resource Classification

Classifications assigned to the open pit MRE reflect confidence in the reliability of the informing data, interpreted mineralisation continuity and sample spacing. Model panels were primarily classified as Indicated and Inferred by estimation search pass and cross-sectional polygons outlining areas of relatively consistently spaced drilling. Comparatively few panels were subsequently re-classified to give a consistent distribution of categories. Figure 4.5 shows the block model by resource classification.

The classification approach assigns estimates for mineralised domain panels tested by drilling spaced at generally around 50 m, generally extrapolated to around 25 m from drilling as Indicated. Inferred estimates are generally based on 100 m spaced drilling, generally extrapolated to around 50 m from drilling, with comparatively rare extrapolation in areas of greater continuity to a maximum of around 100 m from drilling. Around 90% of the combined estimates within the open pit mineral resource are within 50 m of drilling and 99% are within around 75 m.

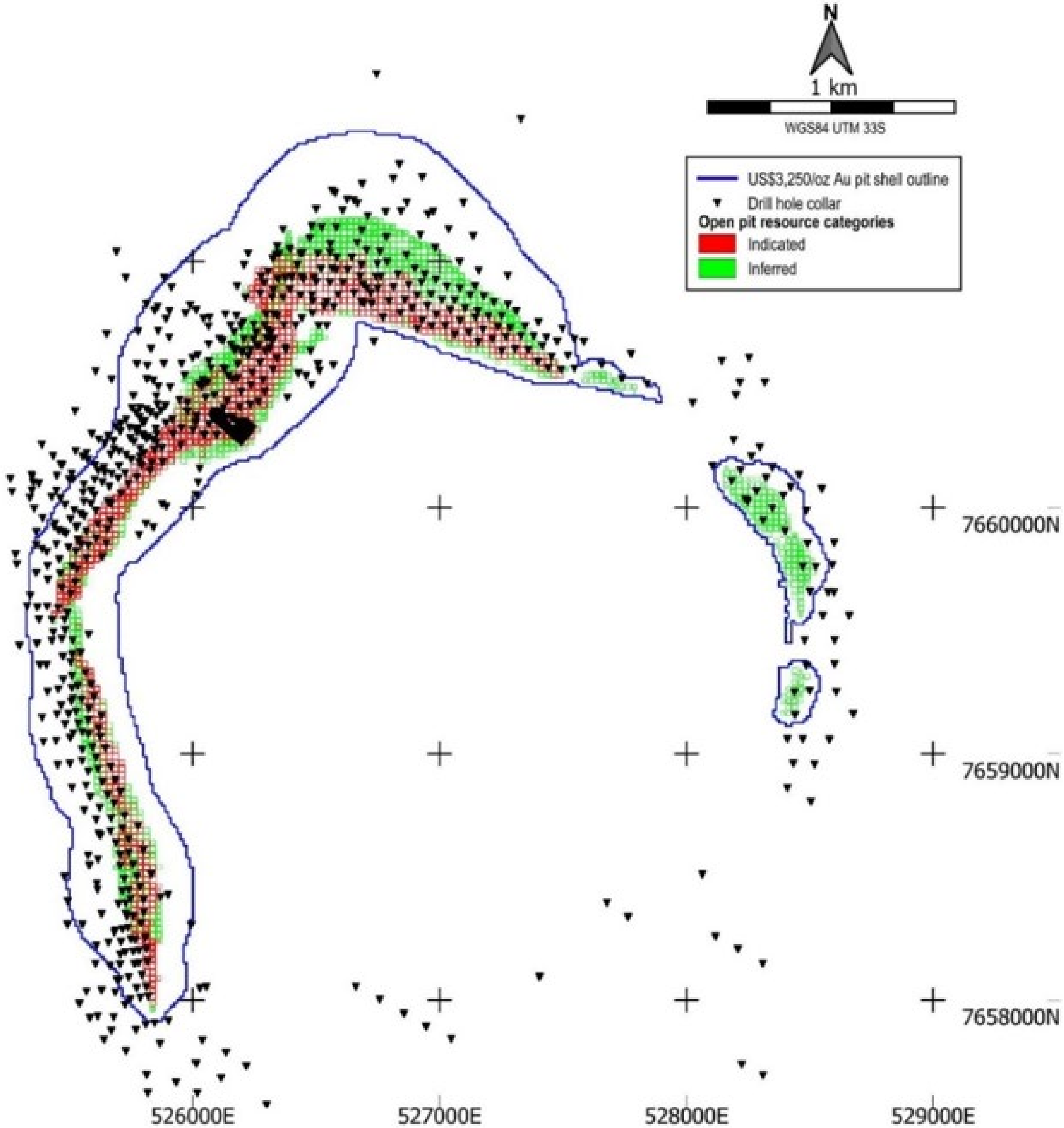


Figure 4.5: Plan View of Kokoseb Open Pit Block Model Displayed by Resource Classification

4.6.3 Reasonable Prospect of Eventual Economic Extraction

Prior to pit optimisation for the assessment of RPEEE, the open pit block model was depleted of all material assigned to the underground Mineral Resource model that occurs below the Ore Reserve pit shell. The Ore Reserve pit shell was derived from the mining studies and forms the basis of the Ore Reserve estimate, as described in Section 8.

To demonstrate RPEEE, the open pit MRE is reported within an optimal pit shell generated using appropriate cost and revenue assumptions, including a gold price of US$3,250/oz, selling costs of US$2.75/oz, processing costs of $12.15/t and metallurgical recovery of 92%. Mining costs were assigned to weathered and fresh material using a surface cost of $1.85 and $2.00/t respectively, increasing with depth at $0.003/t per vertical metre.

The resulting optimised pit shell comprises a main pit accessing the south, northwest and north domains over around 4.9 km of strike, and two satellite pits accessing the east domain totalling around 1 km of strike. The combined shell covers around 5.9 km of mineralised domain strike, with widths of up to 800 m, and a maximum depth around 580 m.

The open pit MRE extends from surface to a maximum depth of approximately 580 m, with around 80% of the contained ounces occurring above 280 m depth and approximately 95% occurring above 450 m depth.

4.6.4 Cut-Off Grades

The pit optimisation used in the open pit RPEEE analysis above give an optimal cut-off of around 0.14 g/t Au. The open pit Mineral Resource is reported over a range of cut-off grades from 0.14 g/t to 0.8 g/t Au to show its sensitivity to cut-off and also to allow consistent comparison with previous Mineral Resource Estimates and comparable regional deposits.

4.7 Underground Mineral Resources

4.7.1 Mineral Resource Estimation

The underground MRE, which is constrained below the Ore Reserve pit shell, is derived from block models generated within wire-framed mineralised domains interpreted designated as the South, Central, Northwest, Central-north and North zones respectively. These domains, which have been interpreted to represent continuous geological features, capture one-metre down-hole composited gold grades for drill hole intercepts of generally greater than 0.85 g/t, with lower grade intervals included for continuity. The mineralised domains were interpreted with a minimum down-hole intercept length of generally around 3 m. The domains, open pit block model, underground block model, exploration targets and the Ore Reserve pit shell are show in Figure 4.6 and long sections of the underground block model are show in Figure 4.7 and Figure 4.8.

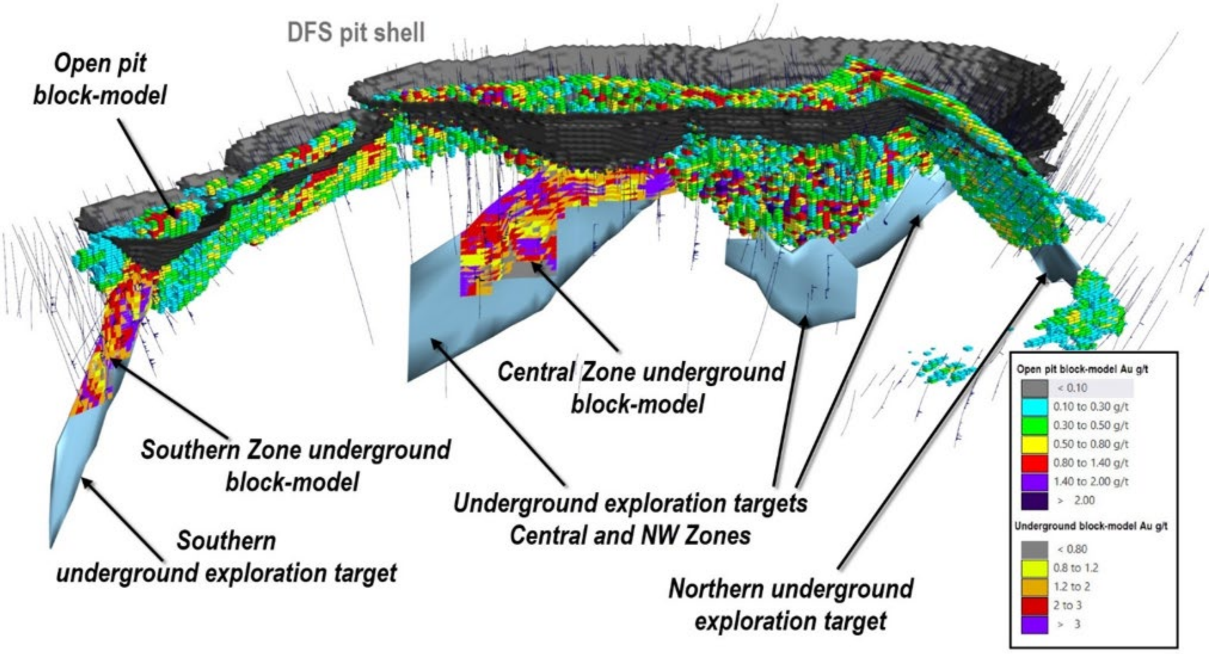


Figure 4.6: Perspective View of Kokoseb looking North-West showing the Open Pit and Underground Block Models and the Ore Reserve Pit Shell

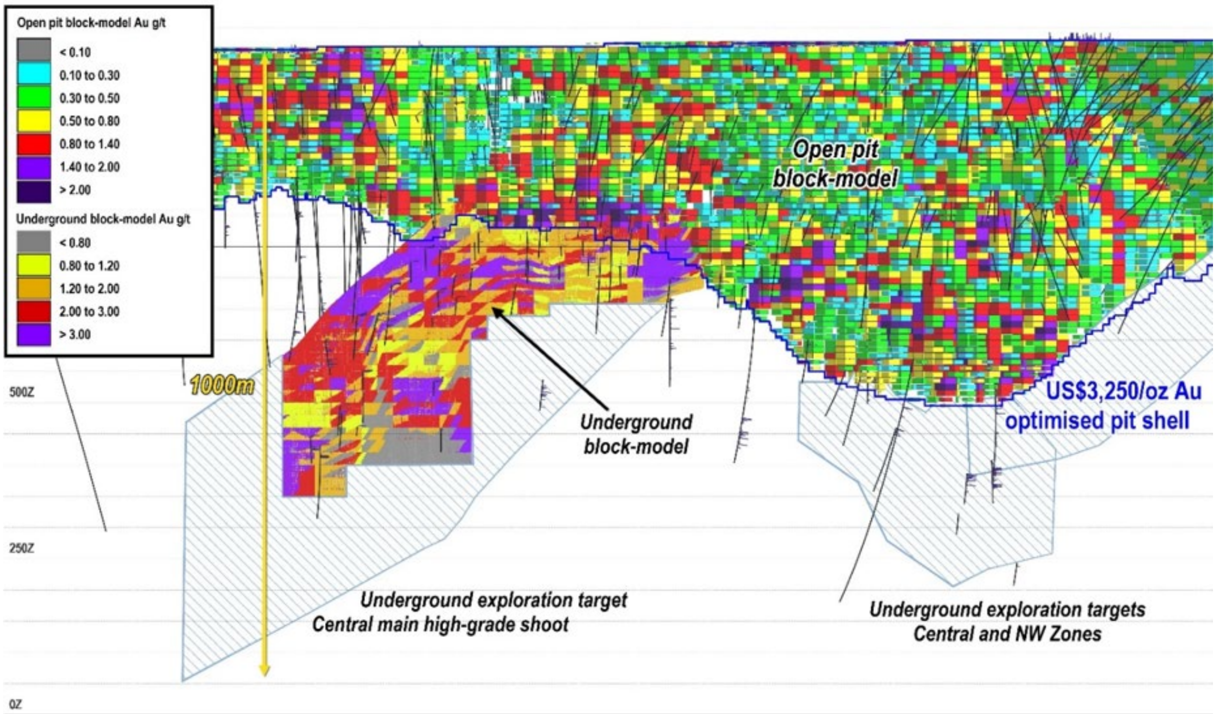


Figure 4.7: Long Section of Underground Block Model – West and Central Zones

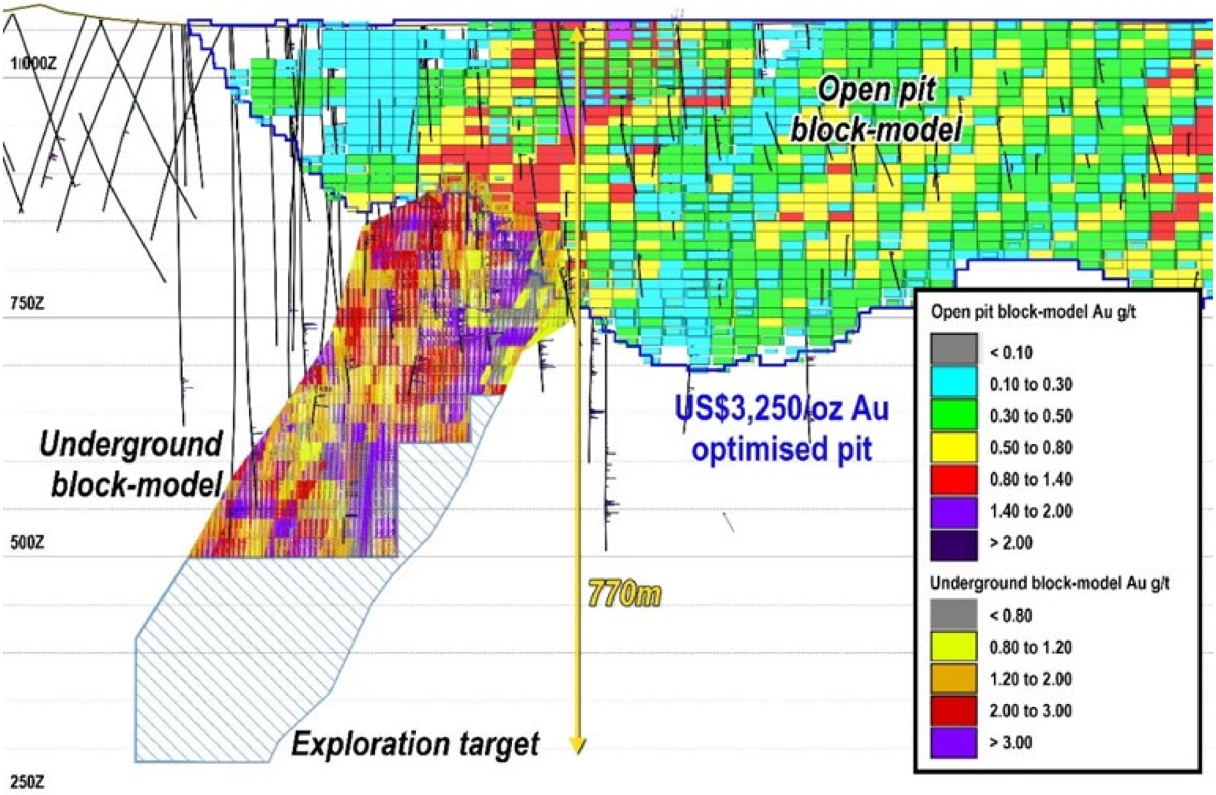


Figure 4.8: Long Section of Underground Block – Southern Zone

Gold grades were estimated by Ordinary Kriging of 1 m down-hole composited gold grades from RC and diamond drilling within the mineralised domains inclusive of upper cuts approximating the 99^{th} percentile of each dataset. The Kriging utilised 4 m by 25 m by 20 m blocks (cross strike by strike by vertical directions) with sub-blocking to minimum dimensions of 0.25 m by 1.25 m by 1.0 m for precise representation of domain boundaries. Bulk densities were assigned consistent with the open pit model, using a value of 2.71 t/m^3 for fresh mineralisation.

Underground Mineral Resources comprise estimates from the south and central zones constrained below the Ore Reserve pit shell. Estimates for the northwest, central-north and north zones, which have only sparse drill coverage below the Ore Reserve pit shell, do not inform Mineral Resources.

The South domain is sub-vertical, and trends north-south, plunging toward the south at around 65 degrees. The portion of the domain included in Mineral Resources extends over around 200 m of strike to a maximum of around 560 m depth, around 270 m below the Ore Reserve pit in this area, with an average horizontal width of around 17 m.

The central domain dips towards the north-east at around 75 degrees with a moderate southerly plunge. The portion of the domain included in Mineral Resources extends over around 600 m of strike to around 720 m depth, around 400 m below the Ore Reserve pit in this area, with a horizontal width averaging around 10 m.

4.7.2 Resource Classification

The classifications assigned to the underground Mineral Resource reflect confidence in the reliability of the informing data, interpreted mineralisation continuity and sample spacing. All estimates are classified as Inferred reflecting the confidence in the interpretation of mineralisation continuity relative to drill spacing. The estimates are based on variably spaced drilling, with intercepts with the MRE volume approximating 100 m spacing and are extrapolated to generally a maximum of around 50 m from drilling.

4.7.3 Reasonable Prospect of Eventual Economic Extraction

Potential underground mining scenarios for the underground Mineral Resource include long hole sublevel open stoping, with a minimum mining width of around 2.5 m. For this scenario, potential mining costs are estimated at around US$70/t of mineralisation, with processing costs, recoveries and royalties consistent with those assigned to the open pit MRE RPEEE analysis. At US$3,250/oz, these parameters give a a cut-off grade of around 0.85 g/t Au and demonstrate that the Underground Mineral Resources have reasonable prospects of eventual economic extraction.

4.7.4 Cut-off Grades

The mining, processing and revenue parameters considered for potential underground mining, and outlined above, give an economic cut-off grade of around 0.85 g/t Au, consistent with the nominal cut-off grade represented by the modelling domains on which the Underground Resources are based.

4.8 Mineral Resource Estimates

Table 4.1presents the MRE, reported in accordance with the JORC Code (2012) delineated by open pit and underground Mineral Resources. The open pit Mineral Resource is presented at a range of selected cut-off grades to provide comparison to previous MREs announced for the Project. The figures in the table are rounded to reflect the precision of the estimates and include apparent rounding errors.

The MRE referred to in this Report was first disclosed in accordance with the requirements of ASX Listing Rule 5.8 in the Company's ASX announcement dated 10 August 2026, titled "Kokoseb Mineral Resource Estimate Increases to 3.78Moz Au". The Company confirms that it is not aware of any new information or data that materially affects the MRE contained in this Report and all material assumptions and technical parameters underpinning the estimate in the previous announcement continue to apply and have not materially changed. The announcement is available to view on www.wiagold.com.au.

Table 4.1: Kokoseb Mineral Resource Estimates

Cut Off Au g/t	Indicated			Inferred			Total		
	Tonnes Million	Au g/t	Au Moz	Tonnes Million	Au g/t	Au Moz	Tonnes Million	Au g/t	Au Moz
Open Pit									
0.14	156	0.57	2.87	120	0.51	2.0	276	0.54	4.83
0.30	100	0.77	2.48	71	0.70	1.6	171	0.74	4.07
0.40	80.7	0.87	2.26	54	0.81	1.4	135	0.84	3.65
0.50	63.9	0.98	2.01	41	0.92	1.2	105	0.96	3.23
0.80	32.4	1.32	1.38	18	1.3	0.75	50	1.3	2.13
Underground									
0.85	-	-	-	7.9	2.2	0.56	7.9	2.2	0.56
Total									
0.50/0.85[3]	63.9	0.98	2.01	49	1.1	1.8	113	1.0	3.78

Notes:
1. The Competent Person responsible for the data informing the estimates is Mr Pierrick Couderc, Wia Group Exploration Manager.
2. The Competent Person responsible for the Mineral Resource modelling is Mr Jonathon Abbott MAIG, Director of Matrix Resource Consultants Pty. Ltd.
3. Total Mineral Resource Estimate is reported at an open pit cut-off grade of 0.50 g/t and an underground cut-off grade of 0.85 g/t to match previous reporting and to provide easy comparison with comparable regional deposits, however the full open pit mineral resource to the cut-off grade of 0.14 g/a Au, as reported in Table 4.1, was used in mining studies.

4.9 Future Exploration and Growth Potential

Exploration activities at the Project have continued to add additional Mineral Resources since discovery, with gold mineralisation remaining open at depth along the greater than five-kilometre strike of the open pit resource and drilling has continuing to return regular high-grade gold intercepts.

Sparse deep drilling completed to date was used to estimate an underground Exploration Target of approximately 10 Mt to 15 Mt at approximately 2 g/t Au to 2.5 g/t. This Exploration Target is additional to, and exclusive of, the Mineral Resources reported in Table 4.1, including the maiden underground Inferred Mineral Resource of 0.56 Moz. The potential quantity and grade of the Exploration Target is conceptual in nature; there has been insufficient exploration to estimate a Mineral Resource and it is uncertain if further exploration will result in the estimation of a Mineral Resource.

The underground Exploration Target is derived from estimates from several Ordinary Kriged block models constrained within geological interpretations around broadly sampled areas, generally more than 50 m from drill holes and not extended beyond 200 m from any drill hole. Additional drilling to approximately 50 m by 50 m spacing would be required to test the validity of the Exploration Target.

In addition, the underground resource of 0.56 Moz represents the initial, closest drilled portions of the potential underground opportunity. High-grade gold mineralisation was identified in several other areas of the deposit as large panels that extend at depth immediately beneath the Ore Reserve pit shell. A portion of the current open pit resource sits below this reserve pit shell and, with further drilling and the underground studies to follow, offers clear scope for reclassification from open pit to underground Mineral Resources.

5 GEOTECHNICAL

5.1 Mining

5.1.1 Investigations

Ground conditions influencing stability in the proposed open pit mine were investigated using:

- Current geological interpretations developed through the course of exploration of the deposit.
- Geotechnical logging of DDH core (HQ and NQ, including half cores) by experienced geotechnical engineers and geotechnical geologists from recent geotechnical investigation and historic exploration boreholes. This included:
  - Drilling of specific DDHs and extension of some exploration DDHs for geotechnical exploration.
  - Logging of intervals from 25 holes spatially dispersed across the proposed open pit (based on the Scoping Study pit optimisation shell and preliminary pit design) with a focus on the central area and southern pit, as these form the focus of early mining.
  - Logging of approximately 5,190 m of diamond drill core, as shown in Figure 5.1.
- Laboratory measurement of physical properties of representative samples of country rocks taken from drilling described above comprising:
  - 65 uniaxial compressive strength (UCS) tests with elastic property determinations (Young’s modulus and Poisson’s ratio as well as UCS determination).
  - 20 multi-stage defect direct shear tests performed on natural defects.

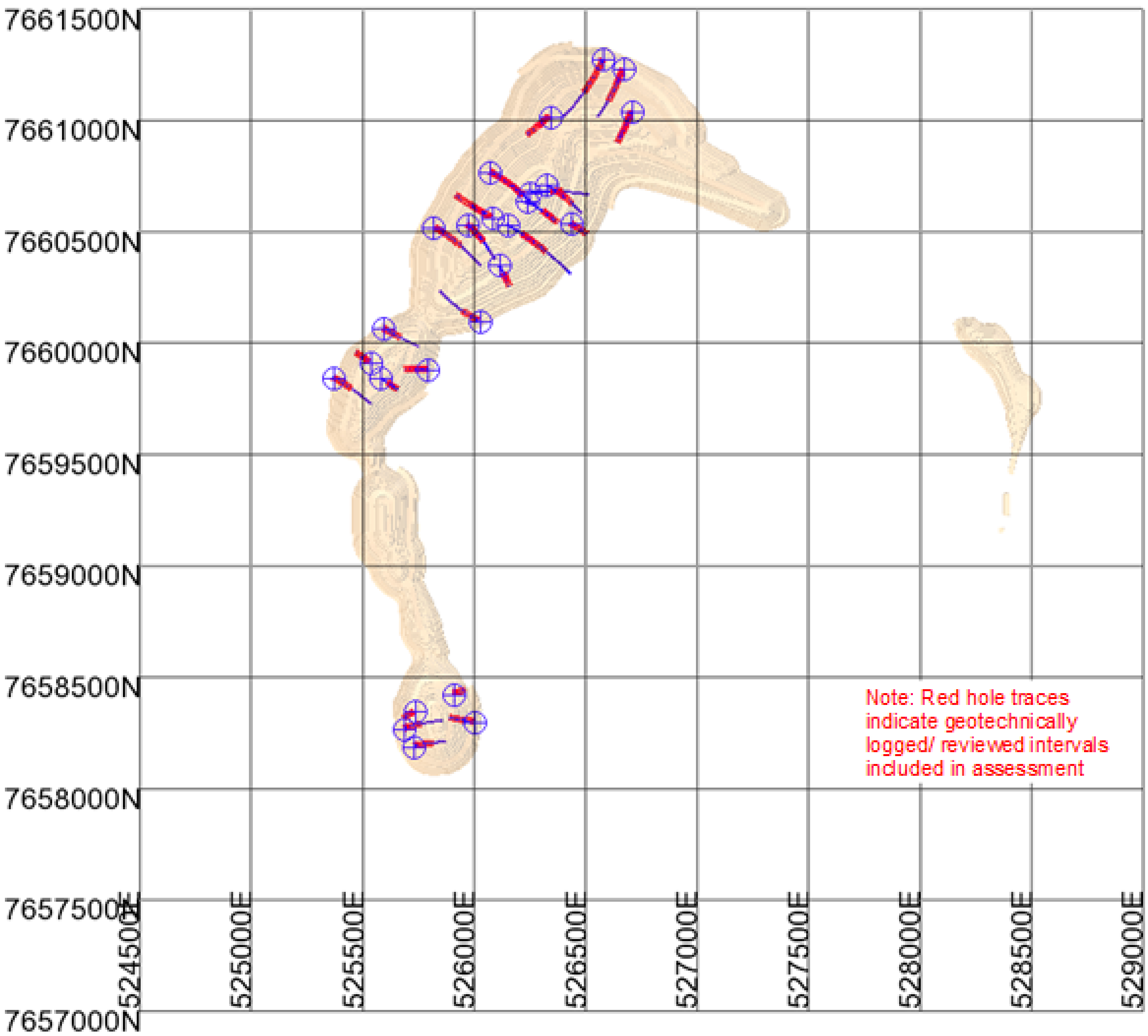


Figure 5.1: Geotechnical Logging (with Preliminary Pit Design)

5.1.2 Major Structures

Currently interpreted major structures within the deposit area comprise:

- An east-west trending, moderately steep south dipping regional thrust fault (the Southern Thrust Fault) at the contact with marble rocks to the immediate south (approximately 200 m) of the southern limit of planned/ potential open pit mining. The fault hanging wall is to the south.
- A north-east trending fault or shear structure adjacent to the south-western limit of the potential mining area, currently interpreted as terminating at the pit crest and not intersecting the pit. This structure is interpreted to bisect and displace the east-west trending thrust fault and marble rock unit.
- A series of steep west dipping mafic dykes located within some of the mineralised zones.

5.1.3 Rock Weathering

The current rock weathering interpretation does not differentiate weathering grade (as per base of complete oxidation (BOCO) and top of fresh rock (TOFR)) and the base of the weathered zone is inferred to represent the TOFR across the project.

The current interpretation indicates that rock weathering extends to moderate though variable depths across the site. The depth of weathering is reported to range from approximately 2 m below surface (mbs) to approximately 110 mbs, with an average of approximately 30 mbs.

Within geotechnically logged boreholes fresh rocks were intersected between 1 mbs and 61 mbs.

As inspected and logged across a number of borehole cores, a typical rock weathering profile, from surface, comprises:

- Less than 2 m soil, residual weathered material/ soil, frequently as unconsolidated gravel.
- Less than 5 m of saprolite/ clay material, however, the unit is locally absent.
- Between 1 m to 61 m highly to moderately weathered rock.

5.1.4 Seismicity

The Project is located within a region of Southern Africa that is considered to have a low risk of significant seismic activity over the anticipated life of the open pit operation.

According to the Global Earthquake Model 6 Foundation, the estimated peak ground acceleration in a 50-year period with a 10 % chance of being exceeded is approximately 0.04g (0.4 m/s^2).

Earthquake-induced ground accelerations of this magnitude would be expected to have minimal influence on pit wall stability. While localised rockfalls from loose material may occur, and marginally stable or metastable zones could be triggered to fail during a seismic event, earthquake-induced effects are not considered a significant risk to the overall stability performance of the proposed open pit.

5.1.5 Rock Mass Classification

Based on the core logging and testwork undertaken on the samples, the rock mass rating was assessed across the proposed mining area.

The mean rock mass rating across all areas ranged from “Fair” to “Very Good” with weathered rocks mean rock mass rating ranging between “Very Poor” to “Good” and fresh rocks mean rock mass rating “Good”.

In addition, there was limited variation in the range of rock mass ratings between individual boreholes and potential wall positions with the mean rock mass quality assessed to be generally “Good”.

5.1.6 Open Pit Geotechnical Design

Recommendations for wall design parameters, given in Table 5.1, were derived from:

- Review of borehole geotechnical logs.
- Results of kinematic stability analyses based on defect data obtained from boreholes.
- Basic rock mass classification using empirical methods.
- Assumed dry/ depressurised wall rock conditions (or that these conditions may be achieved).
- Relevant experience in investigation, assessment, design and operation of open pits of similar scale in similar geotechnical settings.

Table 5.1: Pit Wall Design Parameters

Parameter	Units	All Walls		Eastern Walls[1] (Facing 250° to 330°)	Western Walls[1] (050° to 150°) Northern Wall[1] (180° to 240°) South End Wall[1] (330° to 050°)	Southern Wall (320° to 025°)
		Surface to 5 mbs	5 mbs to TOFR	Below TOFR		
Batter Height	m	5	10	20	20	20
Batter Face Angle	°	40	50	70 ± 10[2]	75	50 ± 10[2]
Berm Width	m	3	5	8	8	8
Inter-Ramp Angle	°	29.2	36.7	52.6 (45.7 – 60.0)	56.3	38.9 (32.1 – 45.7)
Bench Stack Berm (at 80 m with no ramp pass)	m	N/A	N/A	15	15	15
Overall Slope Angle (Crest to Toe)	°	N/A	N/A	50	51	39

Notes:

1. Alternate design parameters using a batter face angle of 90°, berm width of 15 m giving an inter-ramp angle of 53.1° with no bench stack berms required should be adopted if/where initial mining is found to experience excessive loss of wall/ berm profile during blasting and/ or structurally controlled instability.
2. ± 10° batter face angle should aim to match that of rock fabric where dominant.

5.2 Infrastructure

A detailed geotechnical investigation was undertaken for the Project site during November 2025. Based on the layout of facilities from the scoping study, the investigation focused on the proposed plant site, TSF, access roads, diversion drains, diversion bunds, accommodation village and explosives storage area. In addition, a high-level assessment of potential construction material borrow sources was carried out.

The investigation comprised an initial desktop review of available geological and geotechnical information, followed by a fieldwork programme which included 118 test pits, 108 dynamic cone penetration (DCP) testing, 25 double ring infiltrometer (DRI) testing, 9 rotary core boreholes between 9 m and 20 m deep and 9 in-situ falling head permeability tests. Representative soil and rock samples were collected and submitted for laboratory testing.

The laboratory testing carried out included:

- Natural moisture content, pH and conductivity.
- Petrographic analysis.
- Unified soil classification system grading and Atterberg limits tests for determining foundation design parameters.
- Modified AASHTO and California Bearing Ratio for compaction.
- Direct shear testing and consolidated undrained triaxial testing for determination of the shear strength parameters.
- Uniaxial compression tests for rock strength, Youngs modulus and Poisson’s ratio.
- Basson index for assessment of pavement quality.
- Los Angeles abrasion test to assess aggregate durability.
- Emerson crumb, double hydrometer and chemical tests to determine erosion and piping potential.
- Constant head test to determine hydraulic conductivity.

The findings from the geotechnical investigation included:

- **Typical soil profile**: The project area is generally underlain by a thin layer of surficial deposits comprising colluvium and, locally, alluvium. These materials overlie residual soils derived predominantly from the weathering of granitic bedrock, with localised residual profiles developed on schist. The thickness of the residual profile is variable, with schist-derived residual soils typically thicker than those overlying granite. The residual soils exhibit varying degrees of pedogenic cementation, ranging from calcareous soils to hardpan calcrete. Shallow bedrock is commonly encountered within the upper 2 m comprising granite, schist and in places marble or calc silicates with localised deeper soil profiles.
- **Excavatability**: Across the site, excavation conditions are predominantly classified as “soft excavation” to depths ranging from surface (where marble outcrops are

encountered) extending beyond 2.4 m, with a mean depth of approximately 2 m below ground level. It is noted that localised variations in excavation conditions may occur. Deeper than 2 m it is anticipated that “intermediate to hard” excavation conditions will be met.

- **Material re-use suitability for fill**: All the soils encountered across the site are generally granular ranging from silty sand to sandy gravels and cobbles classifying as between G5 and G9 according to COTO 2020. The material encountered on site may be considered potentially suitable for use as construction platform and road-building materials, subject to the applicable structural design requirements. However, the generally thin soil horizons and shallow bedrock depth across the site indicate limited quantities for the use with higher specification granular pavement materials needing to be imported for use in construction.
- **Modified material re-use suitability**: The individual soil horizons that comprise materials of acceptable quality may be selectively blended to increase the available volumes for reuse; however, any such mixed material will require confirmatory laboratory testing to verify compliance with the relevant specification and to assess potential variability. The rock material may be considered suitable for use as structural fill and within lower selected road layerworks if excavated, although processing, including crushing, may be required to control particle size and achieve the specified grading envelope. Additional laboratory testing to assess the suitability of the rock for use as aggregate, including alkali–silica reaction evaluation, to confirm its appropriateness for the intended application, testing to verify compliance with the relevant specification and to assess potential variability. The use of imported aggregate was assumed in the DFS pending this testwork.
- **Groundwater and seepage**: Groundwater was not encountered within the depth of investigation in any of the test pits excavated during the test pit programme. Static water levels recorded in the boreholes were measured at depth greater than 3.8 m below natural ground level (mbgl).
- **Foundation evaluation general**: Considering the stratigraphy of the site, the upper loose transported material is deemed to be compressible and is deemed unsuitable for founding, even under light loads. Structural loads can, however, be placed via pad or strip footings on underlying medium dense residual soils or more competent material such as bedrock. The shallow founding horizon typically occurs at depths from surface to depths of approximately 1.3 mbgl.
- **Foundation evaluation roads**: Site roads are anticipated to be located at or near the current ground level. Therefore, it is recommended that all organic rich horizons are removed prior to the preparation of the subgrade, and in-situ materials ripped (to 300 mm below surface) and recompacted to 93% modified AASHTO compaction density at 0% to +2% of optimum moisture content. The designed layerworks can then be constructed upon this material taking cognisance of the expected California Bearing Ratio values for the in-situ materials that varies between 5 and 35 at 93% modified AASHTO compaction.

The findings from the geotechnical investigation of the Project site and test work on samples taken were detailed in a factual geotechnical report and, based on preliminary loads from the DFS process plant and TSF design an interpretive geotechnical report was prepared outlining the design requirements for foundations and footings.

6 HYDROGEOLOGY AND HYDROLOGY (EPL 4818)

6.1 Hydrogeology

6.1.1 Geographical and Geological Setting

The geology underlying the Project consists of the Damara Orogenic Belt and a complex assemblage of metasedimentary rocks as well as intrusive bodies (Figure 6.1). The lithological framework includes Kuiseb Formation mica schists (NKs), Karibib Formation marbles and impure marbles (NKbAi, NKbHm, NKbOj), Arandis Formation schists and marble/calc-silicate units (NArOb, NArOy, NArSp), intrusive granitoids (including biotite granites and leucogranites (NgSAs/OgSA, Ogl, Egls) and local basement gneisses (MAB). Surficial cover is thin with limited potential for alluvial (porous) aquifer systems due to low order drainage observed on site. Structurally, the area is highly dissected by faults, lineaments, and dyke intrusions, predominantly trending northeast–southwest.

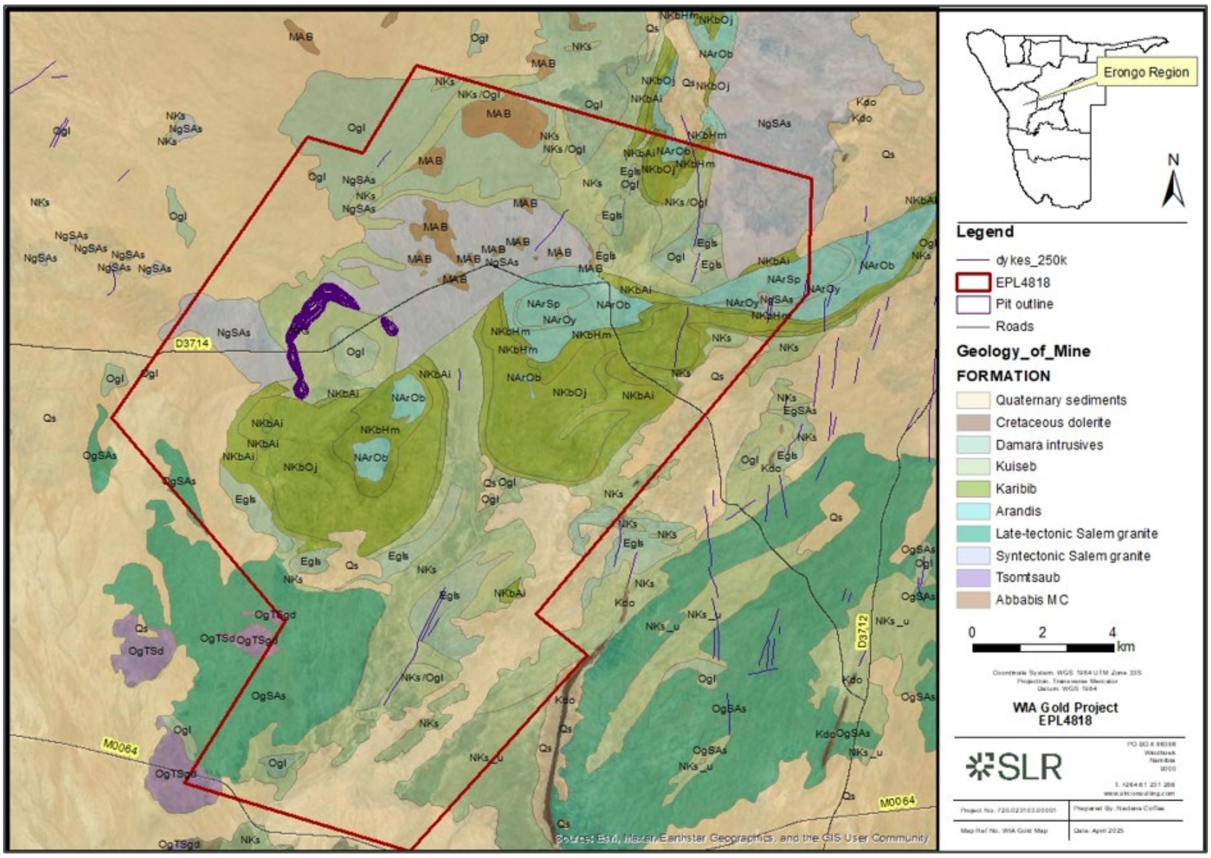


Figure 6.1: Geology of the Project Area

Groundwater occurs predominantly in secondary permeability features such as fractures, lithological contacts or dykes in aquifers hosted in crystalline and metamorphic rocks. Secondary (fractured) aquifers on site have low permeability and are generally low yielding. Groundwater flow is largely controlled by fractures, faults and geological contacts which act as preferential pathways or, in some cases, barriers.

Karibib Formation marble units, near the Kokoseb orebody, are the most favourable hydrogeological targets due to potential occurrence of dissolution features, such as karst cavities, that enhance porosity and aquifer permeability. Granitic intrusions, including Salem-type granites and related bodies, typically exhibit lower permeability even when groundwater is restricted to fractured zones and weathered contacts.

6.1.2 Groundwater Regime

A hydrogeological regime was conceptualised to support numerical groundwater modelling (Figure 6.2). A regional numerical groundwater model was developed as part of a hydrological and hydrogeological impact assessment in support of the development of the EISA for the Project. Following the regional numerical groundwater model, a localised, pit dewatering numerical model was constructed to more accurately simulate possible pit inflow and pore pressures.

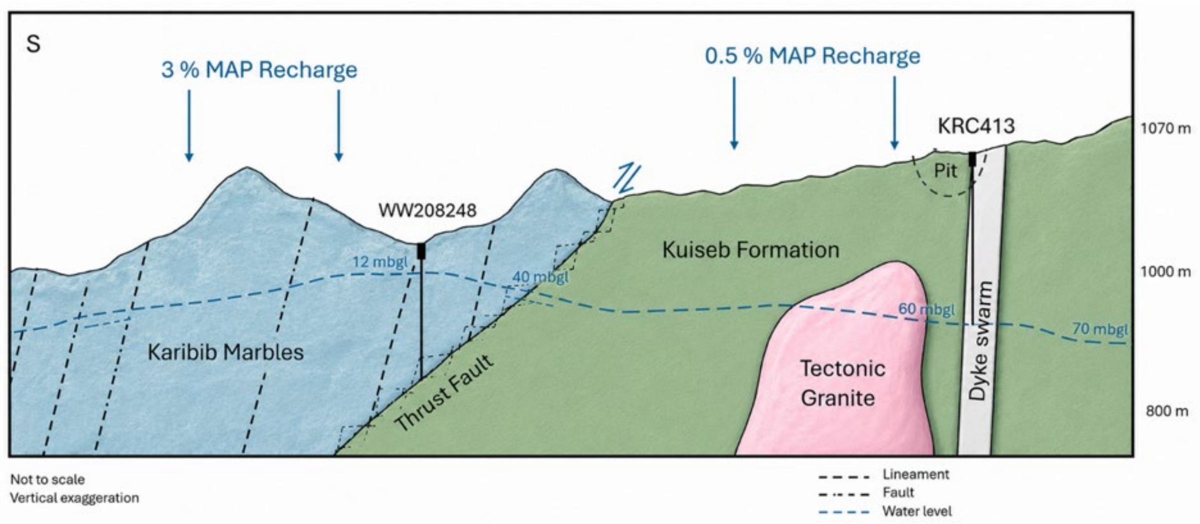


Figure 6.2: Conceptual Model of the Groundwater Regime at Kokoseb

6.1.3 Groundwater Modelling

Given the semi-arid climatic setting and the fractured hard-rock aquifer environment, a comprehensive assessment of both surface water and groundwater systems is key to understanding impacts mining activities will have and inform management measures to minimise risks throughout the life of mine and post-closure as well as informing the requirements for pit dewatering and potential for supplementing the Project water supply.

Numerical modelling considered all surface and groundwater characteristics of the Project area, including the following:

- Ephemeral drainage systems generally drain southward toward the Omaruru River with episodic and event-driven surface water flow, low mean annual rainfall and high evaporation rates.
- Groundwater levels typically vary between 20 mbgl and 45 mbgl, with groundwater elevations ranging between 1,160 m and 970 mamsl from south to north across the EPL. Groundwater flow is generally from north/northeast to south/southwest broadly aligned with the direction of surface flow.

- Groundwater quality is relatively fresh, considering the semi-arid environment, with total dissolved solids (TDS) concentrations between 600 mg/l and 1,500 mg/l.

A three-dimensional numerical groundwater flow and mass transport model was developed using FEFLOW (v10.0) and calibrated using observed groundwater head measurements. The model was designed to evaluate long-term groundwater responses and incorporated the following key assumptions and limitations:

- All predictive scenarios were simulated under steady-state conditions and represented long-term hydraulic equilibrium. As a result, seasonal variability, episodic rainfall events, transient drawdown development, delayed yield effects, groundwater rebound dynamics, and the time required to reach steady-state conditions were also not explicitly assessed.
- Mass transport simulations adopted a conservative tracer approach with a continuous source concentration. Reactive geochemical processes (sorption, precipitation, redox reactions, degradation, density effects and matrix diffusion) were not included.
- Effective porosity and dispersity values were assumed (FEFLOW defaults) and are not site-measured parameters.
- Post-closure simulations assumed pit lake formation at approximately 1,000 mamsl and that the pit will function as a long-term hydraulic sink. Groundwater rebound time and transient filling dynamics were not modelled.
- Due to the arid environment, overflow under extreme rainfall conditions was not considered likely and therefore not assessed within the groundwater model. Transport predictions therefore represent conservative screening-level estimates.

Despite uncertainties, the model was calibrated within acceptable standards as indicated by calibration statistics (RMSE 3.54 m; NRMSE <10%; R^2 = 0.8769). The model was then run across three scenarios to assess the operations phase, mass transport models and post closure.

Scenario 1 simulated full operational pit dewatering under steady-state conditions to quantify predicted inflow rates, drawdown extent and regional hydraulic response, thereby assessing potential impacts on groundwater level, flow and surrounding groundwater users and receptors. Results indicated that the pit acts as a hydraulic sink during operations, capturing groundwater flow and directing potential contaminant migration toward the pit. The pit reaches a maximum depth with the deepest floor at approximately 840 mamsl and, peak groundwater inflow into the pit was calculated in refined modelling of approximately 486 m^3/day (in operating year 5). Groundwater levels surrounding the scoping study pit designs are show in Figure 6.3.

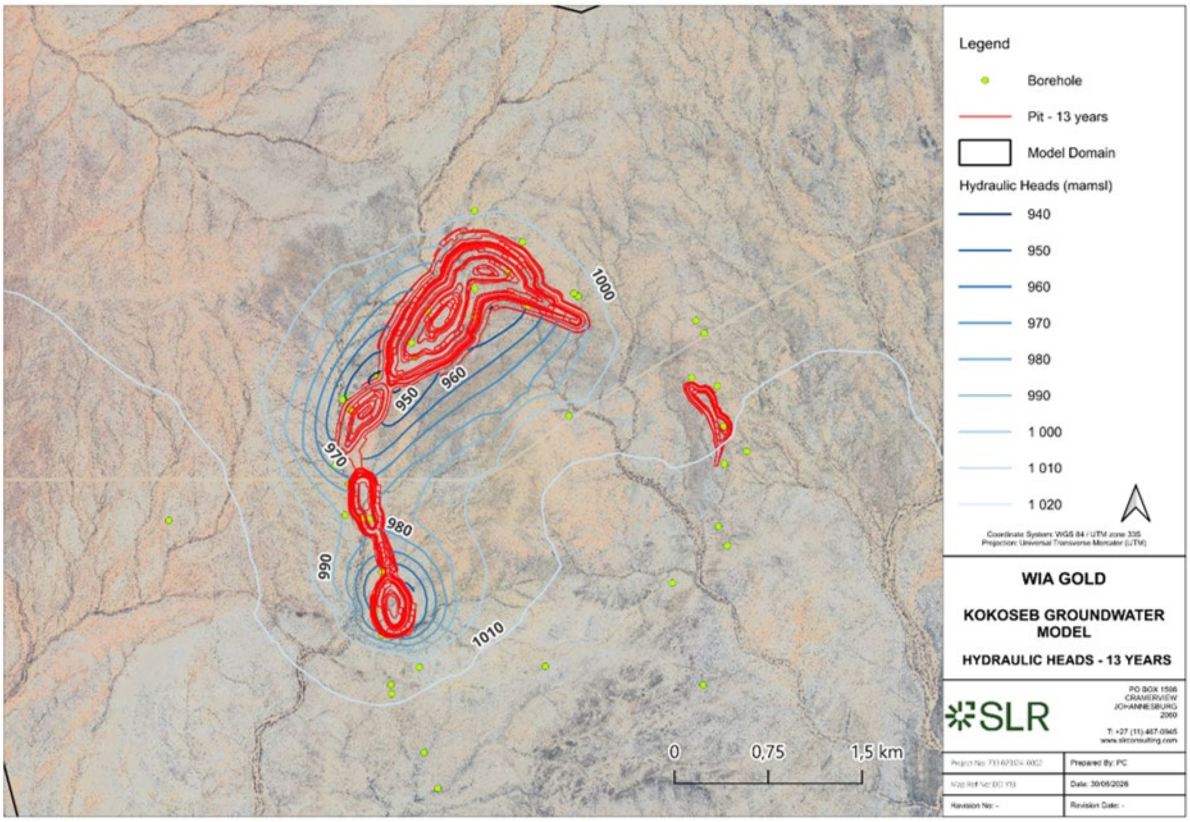


Figure 6.3: Hydraulic Groundwater Levels at LOM

Scenario 2 evaluated operational mass transport for use in assessing potential contaminant plumes determining that any contaminants from the Project would be highly unlikely to impact on nearby receptors. This analysis is fully detailed in the ESIA.

Scenario 3 simulated post-closure steady-state conditions with pit lake formation to evaluate long-term groundwater recovery, hydraulic re-equilibration and contaminant plume behaviour following cessation of dewatering with waste rock dumps (WRDs) and the TSF as continued potential sources of contamination. Post-closure steady-state simulations assume pit lake formation at approximately 1,000 mamsl, functioning as a long-term hydraulic sink.

While rising water levels may mobilise previously oxidised materials locally, the model does not predict outward large-scale contaminant migration under conservative assumptions. Long-term groundwater quality risks are therefore expected to remain predominantly local to the mine footprint and associated waste facilities

6.1.4 Geochemical Assessment

Static geochemical leach testing carried out as part of the ESIA confirmed that mine materials were predominantly non-acid forming (NAF), with negative net acid producing potential (NAPP) values and excess acid neutralising capacity (ANC). Acid mine drainage (AMD) risk was therefore low, with neutral mine drainage (NMD) representing the more plausible geochemical pathway under oxidising conditions.

Kinetic testing is currently underway on samples to fully confirm these findings and quantify any potential NMD from the mined materials.

As a follow-up to the work carried out in the ESIA, samples from the bulk metallurgical leaching tests were assessed using the Synthetic Precipitation Leaching Procedure (SPLP, EPA Method 1312). These tests confirmed that the tailings are NAF with a negative NAPP and excess ANC. The first round of tests, which utilised nitric acid stabilisation of liquor prior to analysis, demonstrated elevated levels of nitrates (likely from the nitric acid) and arsenic (As), however repeat tests on the same samples without nitric acid showed low levels of nitrates (as expected) and no As in solution. These results indicated a potential for NMD of As, however the conflicting results were not conclusive.

Kinetic testing of the tailings is currently underway to provide a more comprehensive understanding of the NMD characteristics of As and to determine whether the addition of nitric acid may have induced the re-mobilisation of As.

6.2 Hydrology

The Project site is located within the upper catchment of the Omaruru River basin, an ephemeral river in western central Namibia. It originates in the Etjo Mountains, runs through the town of Omaruru, and flows into the Atlantic Ocean north of Henties Bay. The Project area is located on an upper area of the watershed as indicated in Figure 6.4. No perennial rivers are located within the site.

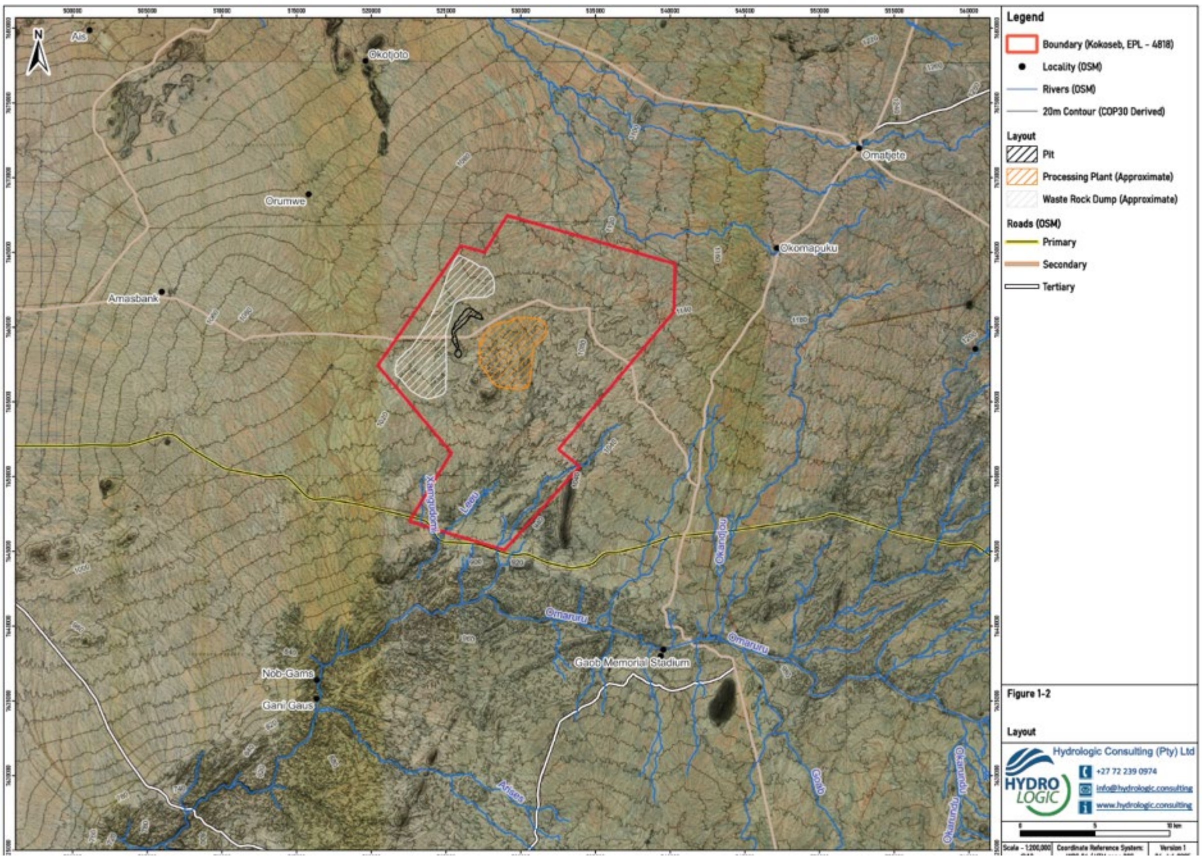


Figure 6.4: Regional Site Layout

6.2.1 Climate

6.2.1.1 Design Rainfall and Evaporation

Data from four rainfall stations operated by the Namibia Meteorological Service were analysed to determine design rainfall and evaporation criteria. The Omaruru station (60 km east of the Project) was selected as the primary data source due to its 90-year observation record, despite two closer stations being available.

The precipitation depths for 24-hour storm events with return periods of 2 to 10,000 years were estimated using statistical methods. The Log Pearson Type 3 distribution method provided a slightly better fit to the upper tail of the data and was therefore adopted as the primary basis for the design 24-hour rainfall depths.

The Project and the Namibia Meteorological Service stations used in this study are all situated within the Benguela-dominated climatic regime (refer Section 3.4) and are therefore considered appropriate for defining the Project's design 24-hour rainfall depths and associated frequency curves.

The annual estimates for reference evapotranspiration (ET_0) from the Food and Agriculture Organisation's AQUASTAT tool indicates an annual ET_0 of 2,489 mm, and the Atlas of Namibia (2020) indicates a potential annual evapotranspiration of 2,509 mm.

6.2.1.2 Climate Change

Due diligence was conducted to assess the potential impacts of climate change at the station and to determine whether a comprehensive study was warranted for the Project. In the Southern African region, simulations are available from the Coordinated Regional Climate Downscaling Experiment (CORDEX-Africa). The greenhouse gas (GHG) forcing scenario, RCP8.5, commonly categorised as "business as usual" benchmark, was chosen to capture upper end of potential precipitation changes. An extract from the Climate Information Portal (2024) analysis for the year 2100 indicates that, for the December to March period, rainfall is projected to decrease by approximately 1 to 3 mm for extreme daily events, corresponding to an approximate increase of 1% to 5% in design 24-hour depths. Given the relatively modest projected changes and the lack of a clear signal, a more detailed climate change impact assessment was not considered necessary at this stage

6.2.2 Flooding Assessment

The hydrological assessment was prepared in accordance with the following standards and guidelines:

- Anglo American Technical Standard AA TS 602 001 (2019), for TSF and stormwater design criteria, consequence classification.
- Namibia Drainage Manual, for stormwater design criteria.
- Global Industry Standard on Tailings Management (GISTM), to inform a risk-based approach and climate resilience considerations.
- International Commission on Large Dams Bulletin 82, for flood design guidelines for tailings dams.

Six sub-catchments, ranging from 3.6 km^2 to 11.2 km^2 were delineated to assess stormwater impacts on mine infrastructure and to support the design of clean water diversions. The catchment boundaries were defined in relation to natural drainage divides identified from the topographic survey as presented in Figure 6.5.

The consequence classification assessment was undertaken in accordance with the Anglo-American standard, which classified the facility as “Moderate”, broadly equivalent to a “Significant” consequence classification under GISTM. A 1:20-year average recurrence interval (ARI) event was adopted for the site and roads diversion works in general, with appropriate freeboard provided. A 1:1,000-year ARI was adopted for the TSF and for diversion works associated with the TSF and WRDs.

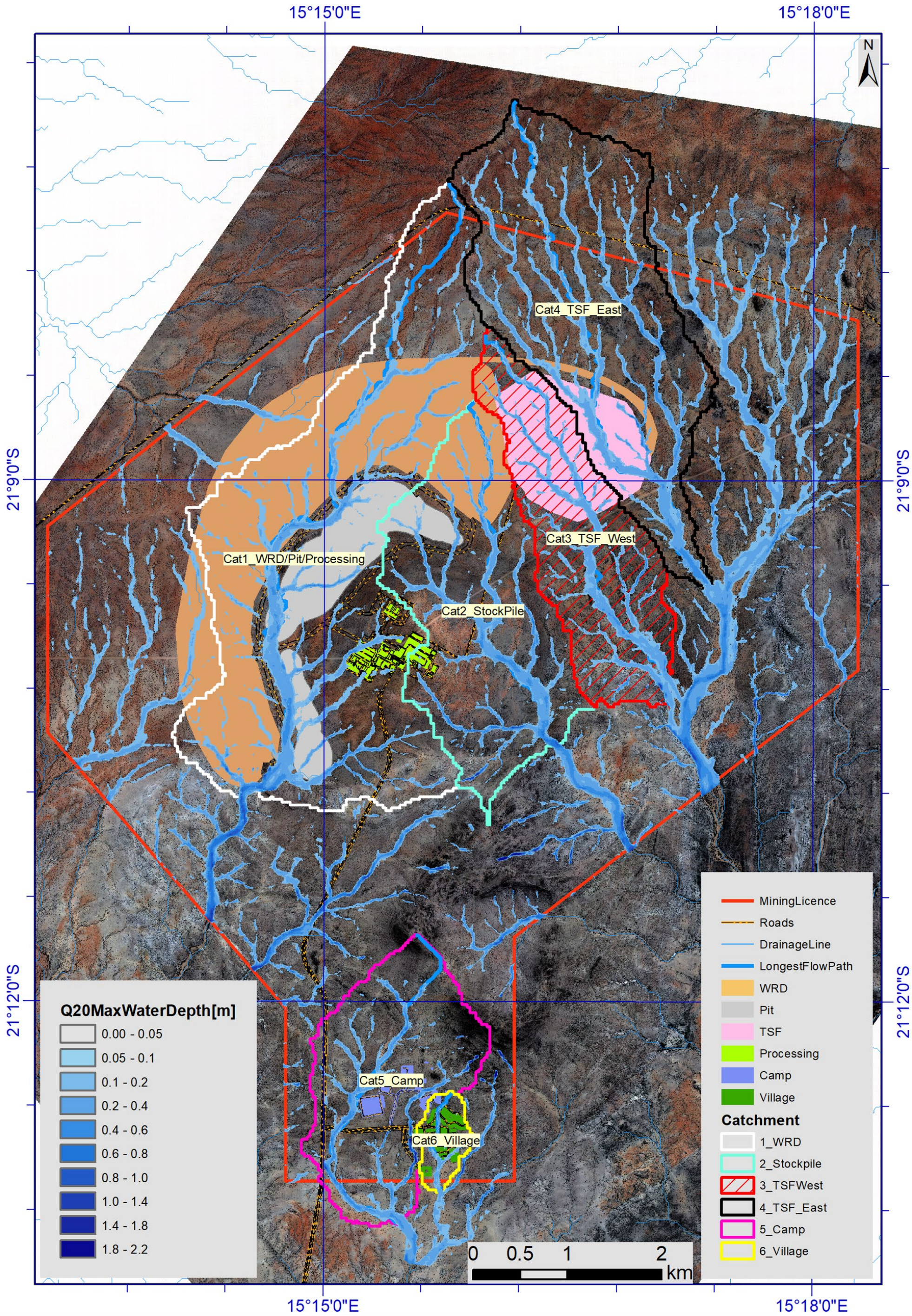


Figure 6.5: Simulated 1:20-year ARI Water Depth

Under existing conditions, simulation shows that natural drainage flows generally follow topographic lows in a southerly direction across the site. Maximum water depths reach 0.8 m to 2.2 m in concentrated flow paths, with velocities ranging from 0.2 m/s to 1.5 m/s, depending on local slope. The proposed locations of the TSF, WRD and processing facilities would be directly impacted by flows from Catchments 4, 3 and 1, respectively, without intervention. Flow from Catchment 6 traverses the proposed village location.

The results confirm that clean water diversions are required to route external runoff around mine infrastructure as flood flows would contact operational areas, potentially creating environmental risks from contaminated runoff and operational risks from flooding of critical facilities. The TSF is particularly vulnerable, with flows from both Catchments 3 and 4 naturally draining toward the TSF location.

6.2.3 Surface Water Management

The surface water management strategy was designed to ensure effective separation of contact and non-contact water, the containment and storage of contact water, and the accommodation of stormwater flows within the DFS mining and site infrastructure layout.

Hydraulic modelling of non-contact water runoff was undertaken for a 1:1,000-year ARI event for critical TSF and WRD diversion structures, and a 1:20 year ARI event for general site channels. This approach ensures that key water management infrastructure is appropriately designed to manage extreme storm events while maintaining the integrity and operability of the Project's facilities.

6.2.3.1 Non-contact Surface Water Management

The WRD will be developed in a phased manner around the north and west sides of the open pits and TSF to divert non-contact stormwater from Catchment 1 and 4. As indicated in Figure 6.6, the WRD will divert water from Catchment 1 to the west to rejoin the natural watercourses to the south of the pits. Water from Catchment 4 will be diverted to the eastern side around the TSF.

The WRD diversion will be completed together with the Phase 1 TSF starter embankment and lining to prevent damage to the TSF. Although velocities will exceed the 1.5 m/s permissible limit for unlined sandy channels along approximately 70-80% of the TSF diversion length, it is acceptable engineering practice as channels are designed to maintain capacity and prevent overtopping during extreme events, with post-event repairs addressing localised erosion for the rare event.

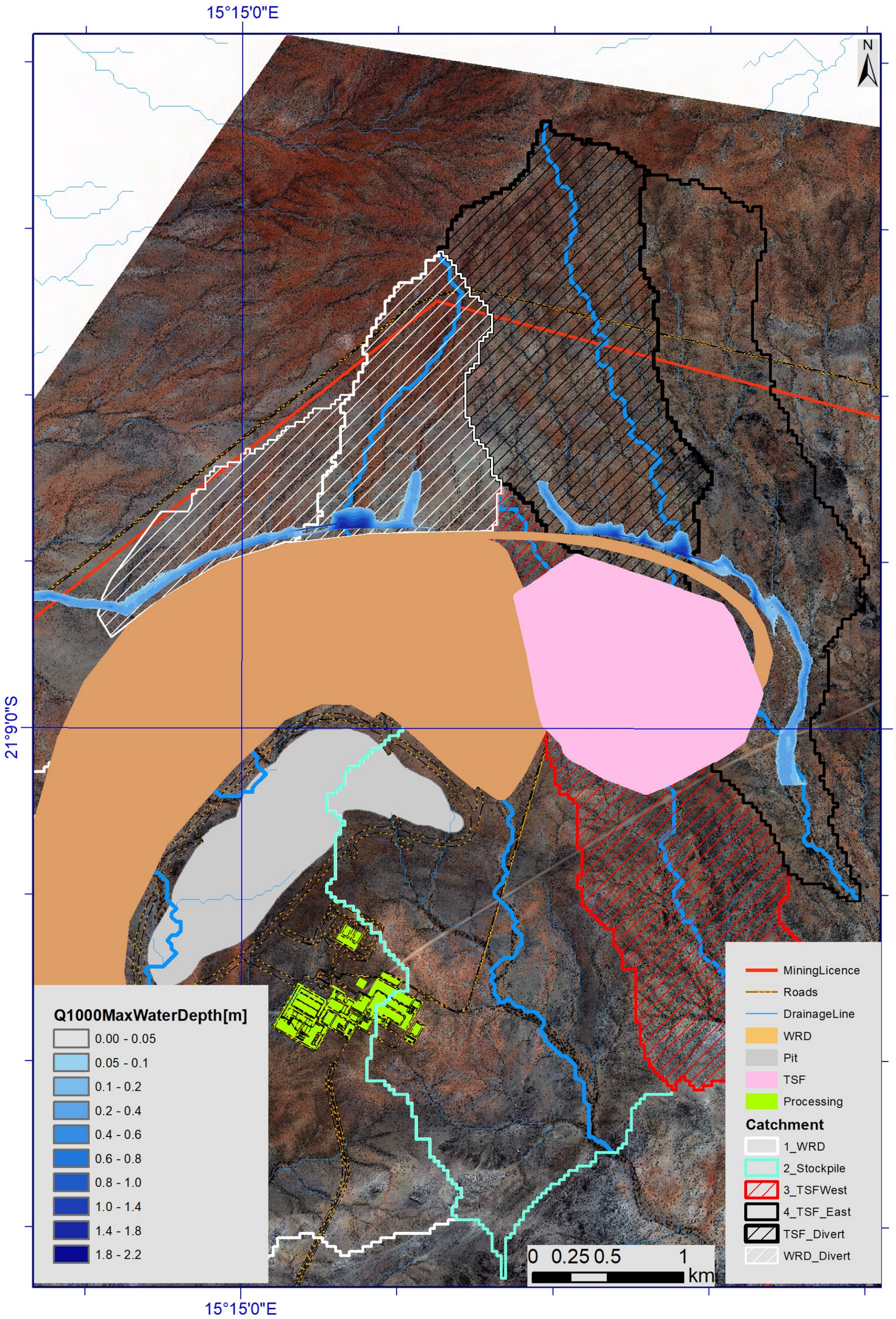


Figure 6.6: Simulated 1:1000-year ARI Water Depth

The process plant and associated infrastructure is strategically placed close to the watershed between Catchment 1 and 2, and the influence of non-contact water is minimal on the plant as protected from flow from larger catchment areas by the proposed WRD and pits. Minor non-contact water collected in the plant area will be divided with stormwater channel/berm combinations.

External runoff from upstream areas will be diverted around the village perimeter using trapezoidal earth channels on the eastern boundary. Internal village drainage will utilise earth-drains along roads and between building blocks to collect rainfall from hardstand areas, roofs, and parking zones. These drains will route runoff from the smaller internal watersheds to defined discharge points, preventing erosion within the residential areas. The earth-drains will discharge to natural drainage courses that form part of the larger catchment system.

6.2.3.2 Contact Water Surface Water Management

Containment of runoff contact water within the TSF will be in a lined downstream seepage / storage pond on the southwestern side of the final TSF footprint. The pond will be sized to accommodate the 1 in 1000-year ARI, 24-hour inflow design flood, as well as seepage water which will be drained from the lined tailings facility.

Contact water within the process plant will be managed with bunds and channels and contained in a pollution control dam on the eastern side of the plant. For the non-processing infrastructure, sheet flow contact water from the parking and workshop areas of the mining infrastructure area will be diverted with channels and internal berms to two pollution control dams to the western and southern side of the plant area.

Water from the TSF storage pond and the pollution control dams will be reclaimed by pumps into the process plant process water circuit, offsetting external water requirements.

7 METALLURGICAL TESTWORK

7.1 Introduction

The scoping study for the Project, completed in September 2025, was based on initial testing of two composite samples. The initial program covered the following testing and investigations and drew the following conclusions:

- Mineralogical characterisation.
- Standard comminution parameters suitable for circuit design.
- Grind and leach testing to define the grind product size P_{80} of 63µm, with additional work recommended.
- Gold extraction of greater than 90% was achieved for the two samples tested.
- Standard gravity recoverable gold testing was undertaken, justifying the inclusion of gravity gold recovery.
- Initial testing of a range of conditions that defined the following leaching conditions:
  - Initial cyanide concentration of 500 ppm with downstream additions to maintain 150 ppm.
  - Pre-oxidation with oxygen is not recommended, although additional investigation was recommended.
  - Preg-robbing is not apparent and therefore full carbon-in-leach (CIL) is not required.
  - 24 hours leach residence time was deemed sufficient based on the limited testing.
  - Carbon loading is not likely to be impeded by competing elements and six stages of contact in 24 hours is suitable.
- Pre-leach and tailings thickening is recommended to minimise reagent consumption and improve the filtration rates on tailings.
- Belt filtration is not suitable for tailings dewatering and pressure filtration achieved 14% moisture in the tailings.

Following on from the scoping study, a comprehensive testwork program was developed to support the DFS, including:

- Sourcing an additional 15 samples selected to be representative in range of grade and the spatial breadth of the resource.
- Extension of the comminution dataset for design with additional SAG mill comminution (SMC), crushing work index (CWi), Bond ball work index (BBWi), abrasion index (Ai) and Geopyora tests.
- Gravity recovery for all 15 variability samples prior to leaching.
- Gravity tails leaching to improve the definition of the metallurgical response to the following variables:

- Leach residence time.
- Grind product across a P_{80} of 75 µm and 63 µm.
- Pre-oxidation and impact on reagent consumption and extraction.
- Leaching in air or oxygen and the impact on reagent consumption and extraction.

- Testing of composites representing the bulk lithologies (weathered and fresh) to define:
  - Bulk ore material handling characteristics.
  - Oxygen uptake rates.
  - Carbon loading.
  - Cyanide destruction conditions.
  - Paste and conventional thickening characteristics.
  - Pressure and vacuum filtration performance.
  - Slurry rheology.

The results of the testwork provides the justification for the flowsheet and design conditions for the feasibility study process design.

The majority of testwork was executed at Maelgwyn South Africa (Johannesburg) or sub-contracted to appropriate specialist laboratories in South Africa.

7.2 Sample Selection

Across the two testwork programs, a total of 17 samples of continuous core were selected from 12 drill holes spanning the breadth of the minable inventory and surrounding waste as demonstrated by the drill hole locations presented within the context of the early-2025 resource pit shell in Figure 7.1. This shows the two original drill holes (KDD044 and KDD046) that the metallurgical samples were taken from for the scoping study and the additional holes for the extended program.

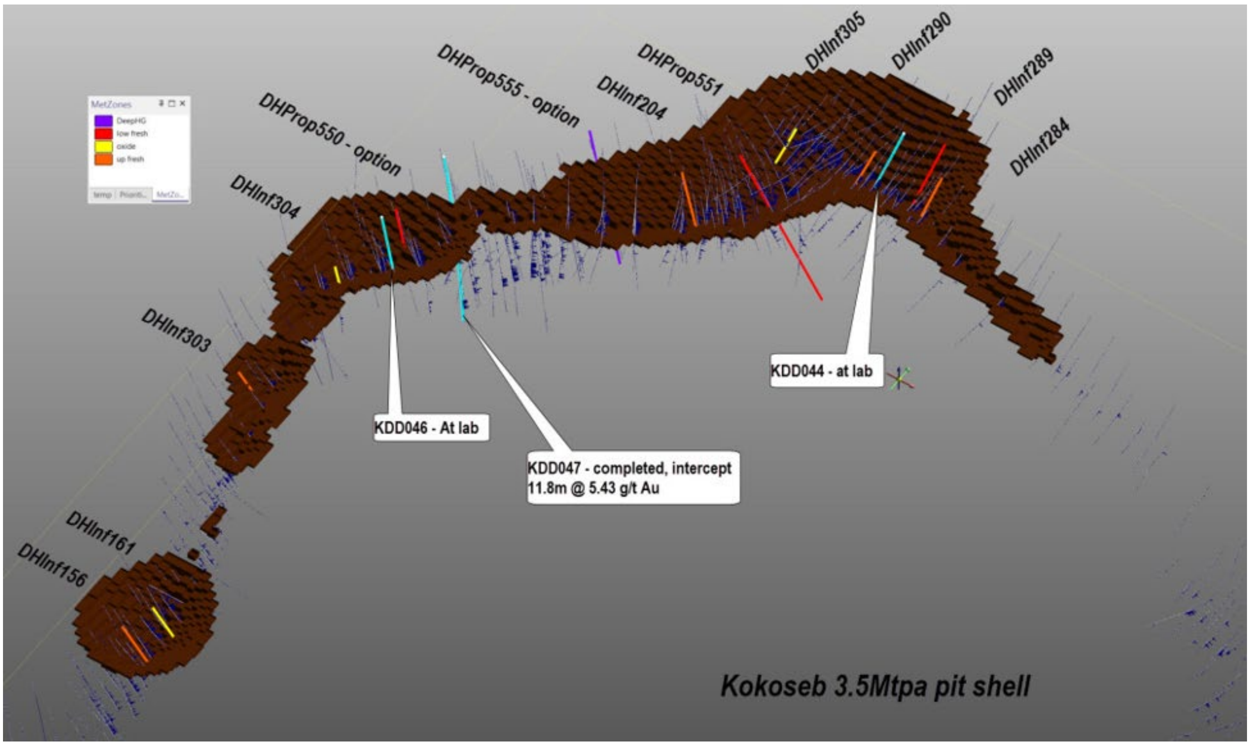


Figure 7.1: Metallurgical Drill Hole Locations

The sample suite was biased towards fresh ore, which represents the majority of the resource, however, near surface weathered samples were specifically targeted from the available drill holes. Three bulk samples of high- and low-grade fresh ore and weathered ore were composited from the selected 17 samples for physical testwork such as materials handling, thickening and filtration.

Table 7.1 provides the details of all samples generated for the DFS testwork program outlining the borehole identification, depth (down hole) and the estimated grade based on the geological sampling and assaying. Table 7.2 provides the head assays from the prepared samples (including the three bulk samples).

Table 7.1: Sample Drillhole Details

Sample ID	Drillhole	BHID	From (m)	To (m)	Description	Estimated Au (g/t)
Sample 1	DHProp555	KDD056	370.8	385.3	Deep Fresh	0.50
Sample 2			385.3	399.3	Deep Fresh	4.25
HW1			262.9	265.9	Hanging Wall	-
FW1			399.3	402.3	Footwall	-
Sample 3	DHInf156	KDD057	156.4	169	Low Fresh	0.58
HW2			46.2	49.3	Hanging Wall	-
FW2			173	176	Footwall	-
Sample 4	DHInf204	KDD058	146.7	167.65	Low Fresh	0.78

Sample ID	Drillhole	BHID	From (m)	To (m)	Description	Estimated Au (g/t)
Sample 5	DHProp551	KDD060	152.6	174.8	Low Fresh	1.15
Sample 6	DHInf284	KDD061	115	132.4	Up Fresh	0.90
Sample 7	DHInf290	KDD062	82.9	103.9	Up Fresh	1.51
Sample 8	DHInf289	KDD063	208	232	Low Fresh	1.66
Sample 9			232	249	Low Fresh	0.94
HW3			162	165	Hanging Wall	-
FW3			279	282	Footwall	-
Sample 10	DHInf305	KDD066	9	37	Weathered	1.73
Sample 11			72.9	80.9	Up Fresh	1.19
Sample 12	DHInf161	KDD069	120	142	Up Fresh	0.62
Sample 13			0	24	Weathered	0.62
Sample 14			33	55	Shallow Fresh	1.52
Sample 15	DHInf304	KDD070	9.4	29	Weathered	0.94

Table 7.2: Sample Head Assay Analysis

Sample ID	Measured Duplicates						Average	Exploration Sections
	1	2	3	4	5	6		
Sample 1	0.74	0.63	0.66	0.64			0.67	0.50
Sample 2	3.73	4.44	4.45	4.46			4.27	4.25
Sample 3	0.98	1.79	1.75	1.84			1.59	0.58
Sample 4	0.85	0.97	1.04	1.06			0.98	0.78
Sample 5	0.94	1.14	1.14	1.25			1.12	1.15
Sample 6	0.84	0.55	0.53	0.58			0.63	0.90
Sample 7	1.43	1.38	1.41	1.35			1.39	1.51
Sample 8	1.07	1.01	1.00	1.45	1.55	1.16	1.21	1.66
Sample 9	0.91	0.87	0.83	0.77			0.85	0.94
Sample 10 (weathered)	1.24	0.96	1.12	1.06	1.28		1.13	0.94
Sample 11	2.17	1.42	2.48	1.50	1.98		1.91	1.73
Sample 12	0.70	0.81	0.79	0.89			0.80	1.19
Sample 13 (weathered)	0.53	0.68	0.70	0.64			0.64	0.62
Sample 14	1.27	1.24	1.18	1.17			1.21	1.52
Sample 15 (weathered)	0.84	0.73	0.72	0.78			0.77	0.94
Weathered Composite	1.75	2.85	1.87	2.01			2.12	0.90
Fresh Low-Grade Composite	0.84	0.86	0.81				0.84	0.91
Fresh Average-Grade Composite	1.13	1.05	1.09				1.09	1.37

Whilst the majority of samples show good correlation between the measured gold grades and those estimated from the exploration sections, a number of outliers exhibit significant deviations. In addition, variability is observed between duplicate assay results. This discrepancy between expected and measured grades, as well as between duplicate assays, is considered a consequence of the nuggety nature of the gold mineralisation, which can result in locally erratic grade distribution and reduced assay repeatability. To minimise the impact of this variability on metallurgical testwork outcomes, all leach testwork undertaken as part of the DFS was conducted on gravity tails. This approach substantially reduces the influence of coarse free gold and improves the consistency and reliability of assay results

In addition to the head assays, the samples were analysed for sulphur and carbon with the results presented in Table 7.3.

Table 7.3: SG, Sulphur and Carbon Speciation

Sample ID	SG	Total S (%)	Sulphide S (%)	Total C (%)	Graphitic C (%)	Organic C (%)
Sample 1	2.78	0.30	0.22	0.37	0.23	0.01
Sample 2	2.79	0.24	0.21	0.44	0.29	0.01
Sample 3	2.71	0.18	0.17	0.18	0.10	<0.01
Sample 4	2.79	0.32	0.31	0.40	0.29	<0.01
Sample 5	2.78	0.24	0.23	0.43	0.30	<0.01
Sample 6	2.79	0.29	0.27	0.18	0.06	<0.01
Sample 7	2.80	0.38	0.34	0.21	0.05	<0.01
Sample 8	2.78	0.35	0.30	0.25	0.08	0.04
Sample 9	2.78	0.19	0.19	0.22	0.10	<0.01
Sample 10	2.75	0.02	0.02	0.32	0.20	<0.01
Sample 11	2.78	0.27	0.26	0.22	0.13	<0.01
Sample 12	2.78	0.22	0.15	0.25	0.12	0.04
Sample 13	2.73	0.02	0.02	0.48	0.35	0.01
Sample 14	2.74	0.29	0.27	0.28	0.12	<0.01
Sample 15	2.73	0.03	0.02	0.42	0.30	0.02
Average	**2.77**	**0.22**	**0.20**	**0.31**	**0.18**	**<0.01**
Median	**2.78**	**0.24**	**0.22**	**0.28**	**0.13**	**<0.01**
Weathered Composite	2.77	0.07	0.05	0.38	0.27	<0.01
Fresh Low-Grade Composite	2.81	0.24	0.20	0.27	0.17	<0.01
Fresh Average-Grade Composite	2.80	0.25	0.18	0.27	0.16	<0.01

These results indicate relatively low levels of sulphide sulphur and insignificant levels of organic carbon suggesting that acid forming and preg-robbing are unlikely to be issues requiring action during the process plant design.

7.3 Comminution Testwork

Comminution test work was carried out on all of the variability samples developed with remnants from the comminution testing returned to the samples for further leaching and other testwork.

The test work carried out included:

- Testing of seven samples, including fresh, moderately weathered and completed weathered samples, using:
  - SMC testing.
  - CWi.
  - BBWi.
  - Ai.
- Testing of all samples, including those tested above as paired samples, using the Geopyora testing method.

Testing of seven samples using both the SMC/Bond comminution test suite and Geopyora test methodology enabled calibration of the Geopyora results, thereby increasing confidence in the derived comminution parameters. A comparison of the specific grinding energies calculated from the two test methods is presented in Figure 7.2. The results demonstrate excellent correlation between the datasets, providing strong validation of the Geopyora test results and supporting their use across the broader sample set.

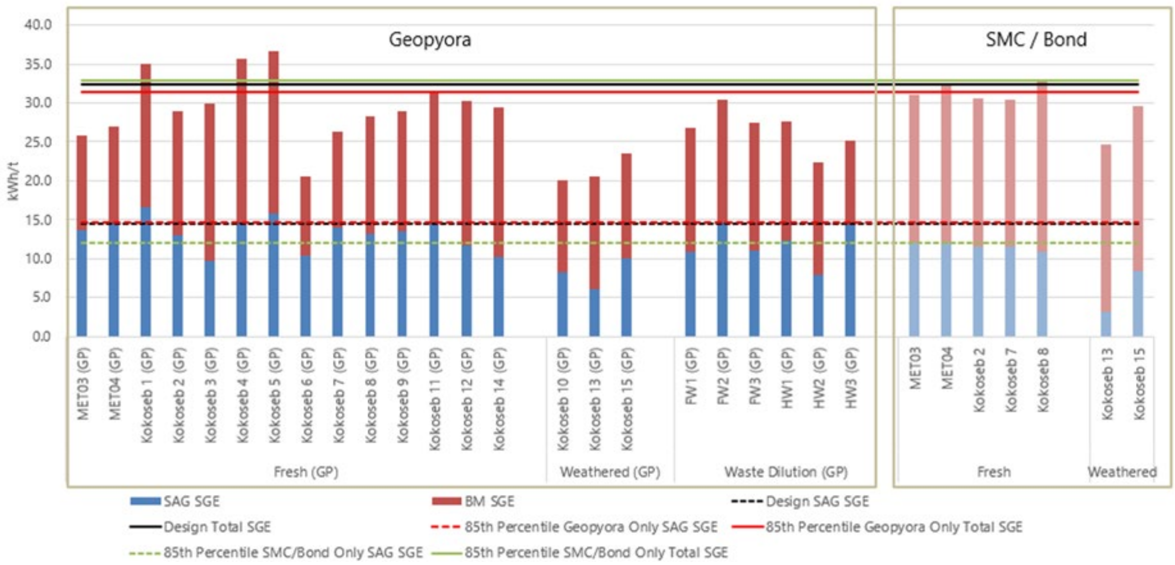


Figure 7.2: Comparison of Specific Grinding Energy Requirements

Based on the full dataset the comminution design parameters were developed and are summarised in Table 7.4.

Table 7.4: Comminution Testwork Summary

	Unit	Fresh	Weathered	Waste	Design
Number of samples		3	14	6	
CWi (Range)	kWh/t	9.4 – 17.4	6.4 – 7.6	-	17.4
RWi (85th Percentile)	kWh/t	14.3	-	-	14.3
BBWi (85th Percentile)	kWh/t	20.5	17.9	15.7	19.2
Axb (15th Percentile[1])		25.7	42.3	25.9	25.9
Ai (g)	g	0.176	0.087	-	0.150
SG (Average)		2.76	2.66	2.76	2.74

Notes:

1. Higher Axb represents softer material, consequently the 15th percentile represents the 85th percentile hardest ore.

The conclusions drawn from these results include:

- Fresh material reflects high to extremely competent comminution characteristics (low Axb) with waste material generally consistent with the fresh material.
- Weathered material reflects low to moderate competency.
- Fresh material is moderately abrasive, whereas weathered material has low abrasion characteristics.
- Both fresh and weathered materials are moderate to high hardness (higher BBWi) with the waste material being slightly softer on average.

7.4 Gravity Gold Recovery

The scoping study and the analysis of the DFS samples identified the likelihood of nuggety gold which is consistent with significant gravity gold recovery. To test this, and confirm the results from the scoping study, the DFS testwork program included the following testing on all samples:

- Preparation and grinding of a 10 kg sub-sample to a P_{80} of 212 µm.
- Processing of the ground sample using a Knelson MD3 bench scale gravity recovery unit to recover approximately 100 g of gravity concentrate from each 10 kg feed sample.
- Leaching of the gravity concentrate in a bottle roll with a cyanide concentration of 2% NaCN along with hydrogen peroxide and Leachaid addition.

The results of this testing is summarised in Table 7.5

Table 7.5: Gravity Testing Results

Sample	Au Head Grade (g/t)		Gravity/ Intensive Leach Recovery (%)	Gravity Tails Au Grade (g/t)
	Assay	Calculated		
Sample 1	0.67	0.69	50.0	0.35
Sample 2	4.27	4.17	28.6	2.97
Sample 3	1.59	1.57	65.9	0.54
Sample 4	0.98	0.80	44.6	0.44
Sample 5	1.12	1.26	43.6	0.71
Sample 6	0.63	0.62	45.8	0.33
Sample 7	1.39	1.16	45.2	0.64
Sample 8	1.21	1.10	44.0	0.62
Sample 9	0.85	0.90	72.2	0.25
Sample 10	1.13	1.10	49.9	0.55
Sample 11	1.91	1.94	55.4	0.86
Sample 12	0.80	0.77	38.7	0.47
Sample 13	0.64	0.73	13.8	0.63
Sample 14	1.21	1.15	61.4	0.44
Sample 15	0.77	0.89	32.9	0.60

No clear relationship is evident between feed grade and gravity/intensive leach recovery that could be used to reliably predict gravity gold recovery. Across all samples tested, the median gold recovery achieved through gravity concentration was 45.2%. This represents a high level of gravity recoverable gold and is consistent with the results of the earlier metallurgical testwork, including extended gravity recoverable gold (EGRG) testing, undertaken on the two samples evaluated during the scoping study. The DFS testwork therefore validates the strong gravity recovery response identified in the initial metallurgical programme.

7.5 Leach Testwork

Given the nuggety nature of the gold mineralisation and the high proportion of gravity recoverable gold, all leach testing was carried out on gravity tails. Each variability sample was processed in accordance with the methodology outlined above to produce a gravity tail sub-sample. The gravity concentrate generated for each variability sample was subjected to intensive leaching as outlined above with the residue from the intensive leach recombined with the gravity tail for use in leach testing.

To further mitigate the effects of grade variability associated with the nuggety gold distribution, all leach tests were performed in duplicate. Leach tests also included sub-sampling at 2, 4, 8 and 24 hours to provide kinetic leaching information.

7.5.1 Optimisation Testing

Following on from the testing of the samples in the scoping study, a selected group of five variability samples (including weathered and fresh samples) were tested to optimise the process parameters, including:

- Pre-oxidation.
- Use of oxygen or air for oxidation in leaching.
- Carbon-in-leach.

These leaching tests were all carried out on gravity tails, as outlined above, and at standard conditions including:

- 50% solids.
- 63 µm grind.
- Cyanide addition to an initial dose of 1,000 kg/t NaCN and additions to maintain 500 ppm NaCN.
- Lime addition to achieve pH 9.5 to 10.

The results of these tests, and the conclusions drawn outlined below.

7.5.1.1 Pre-Oxidation

Tests were carried out both with and without pre-oxidation with oxygen with leaching results and reagent consumption results outlined in Table 7.6 and Table 7.7.

Table 7.6: Pre-Oxidation Tests Leaching Results

Sample		Head Au	48-hour Au Residual (g/t)		48-hour Extraction (%)	
		(g/t)	Pre-Ox	No Pre-Ox	Pre-Ox	No Pre-Ox
Sample 2	Fresh	4.270	0.150	0.120	95.0%	96.0%
Sample 7	Fresh	1.393	0.050	0.060	92.2%	90.6%
Sample 8	Fresh	1.207	0.140	0.140	77.3%	77.3%
Sample 13	Weathered	0.637	0.040	0.050	93.7%	92.1%
Sample 15	Weathered	0.768	0.080	0.090	86.8%	85.1%
Average – Fresh			**0.113**	**0.107**	**88.2%**	**88.0%**
Average – Weathered			**0.060**	**0.070**	**90.2%**	**88.6%**

Table 7.7: Pre-oxidation Tests Reagent Consumption Results

Sample		Head Au	CN Consumption (kg/t)		Lime Consumption (kg/t)	
		(g/t)	Pre-Ox	No Pre-Ox	Pre-Ox	No Pre-Ox
Sample 2	Fresh	4.270	0.455	0.577	0.900	0.900
Sample 7	Fresh	1.393	0.480	0.547	0.500	0.500
Sample 8	Fresh	1.207	0.490	0.528	0.550	0.550
Sample 13	Weathered	0.637	0.613	0.658	0.950	0.950
Sample 15	Weathered	0.768	0.450	0.580	0.700	0.700
Average – Fresh			**0.475**	**0.551**	**0.650**	**0.650**
Average – Weathered			**0.532**	**0.619**	**0.825**	**0.825**

These result show that while there is no material benefit on extraction or lime consumption from the inclusion of pre-oxidation, there is a material reduction in cyanide consumption from pre-oxidation of approximately 0.08 kg/t cyanide. This reduction represents an approximate operating cost saving of US$1.1m/a.

7.5.1.2 Oxygen versus Air in Leaching

Tests were carried out with either oxygen or air used for aeration in the leach with leaching results and reagent consumption results outlined in Table 7.8 and Table 7.9.

Table 7.8: Oxygen versus Air Leaching Results

Sample		Head Au	48-hour Au Residual (g/t)		48-hour Extraction (%)	
		(g/t)	Oxygen	Air	Oxygen	Air
Sample 2	Fresh	4.270	0.120	0.120	96.0%	96.0%
Sample 7	Fresh	1.393	0.060	0.060	90.6%	90.6%
Sample 8	Fresh	1.207	0.150	0.140	75.7%	77.3%
Sample 13	Weathered	0.637	0.040	0.050	93.7%	92.1%
Sample 15	Weathered	0.768	0.085	0.090	86.0%	85.1%
Average – Fresh			**0.110**	**0.107**	**87.4%**	**88.0%**
Average – Weathered			**0.063**	**0.070**	**89.8%**	**88.6%**

Table 7.9: Oxygen versus Air Reagent Consumption Results

Sample		Head Au	CN Consumption (kg/t)		Lime Consumption (kg/t)	
		(g/t)	Oxygen	Air	Oxygen	Air
Sample 2	Fresh	4.270	0.551	0.577	0.900	0.900
Sample 7	Fresh	1.393	0.506	0.547	0.500	0.500
Sample 8	Fresh	1.207	0.513	0.528	0.550	0.550
Sample 13	Weathered	0.637	0.695	0.658	1.000	0.950
Sample 15	Weathered	0.768	0.576	0.580	0.700	0.700
Average – Fresh			**0.523**	**0.551**	**0.650**	**0.650**
Average – Weathered			**0.635**	**0.619**	**0.850**	**0.825**

These results indicate that there is no material difference in extraction on samples of fresh material with a small increase in extraction on weathered samples. However, since weathered material does not represent a large proportion of the Mineral Resources it was determined that there is no benefit through the use of oxygen in leaching. No material difference in reagent consumption was demonstrated.

7.5.1.3 Carbon-in-Leach

Tests were carried out with and without the addition of carbon into the leaching to determine if there is any preg-robbing effect evident. The leaching results are outlined in Table 7.10.

Table 7.10: Carbon-in-Leach (CIL) Leaching Results

Sample		Head Au	48-hour Au Residual (g/t)		48-hour Extraction (%)	
		(g/t)	CIL	No CIL	CIL	No CIL
Sample 2	Fresh	4.270	0.130	0.120	95.7%	96.0%
Sample 7	Fresh	1.393	0.070	0.060	89.0%	90.6%
Sample 8	Fresh	1.207	0.130	0.140	78.9%	77.3%
Sample 13	Weathered	0.637	0.030	0.050	95.2%	92.1%
Sample 15	Weathered	0.768	0.070	0.090	88.4%	85.1%
Average – Fresh			**0.110**	**0.107**	**87.9%**	**88.0%**
Average – Weathered			**0.050**	**0.070**	**91.8%**	**88.6%**

These results suggest that there is no consistent improvement in gold extraction on fresh samples with carbon in leach, however there is a marginal improvement on tests on the weathered samples (Samples 13 and 15). This implies that preg-robbing is not a factor for fresh material, however there may be a minor impact on weathered material. While this impact is noted for the weathered material, given the weathered material does not represent a large proportion of the Mineral Resources, it was determined that the benefit of using a full CIL circuit was not warranted for the Project.

7.5.2 Variability Leach Testing

Variability leach testing was carried in duplicate out on the gravity tails form all variability samples as outlined above. The testing conditions included:

- 50% solids.
- Cyanide addition to an initial dose of 1,000 kg/t NaCN and additions to maintain 500 ppm NaCN.
- Lime addition to achieve pH 9.5 to 10.
- No pre-oxidation and air sparging in the leach.
- No carbon addition.
- Leaching for 48-hours with sub-samples at 2, 4, 8, 24 hours.

All 15 variability samples were subjected to testing using grind sizes of P_{80} 63 µm and 75 µm. The results of these tests show in Table 7.11 and Table 7.12. Figure 7.3 presents these results graphically.

Table 7.11: Results of Variability Leach Testing at 63 µm

Sample	Weathering	P_{80} (µm)	Head Au Grade (g/t)		Residual Au Grade (g/t)		Au Extraction (%)			Reagents	
			Assay	Calc.	24-h	48-h	Gravity	24-h	48-h	NaCN	Lime
Sample 1	Fresh	63	0.668	0.693	0.134	0.095	50.0%	80.7%	86.3%	0.53	0.40
Sample 2	Fresh	63	4.270	4.169	0.302	0.120	28.6%	92.8%	97.1%	0.55	0.90
Sample 3	Sight/Moderate	63	1.590	1.574	0.062	0.050	65.9%	96.1%	96.8%	0.53	1.25
Sample 4	Fresh/Slightly	63	0.980	0.800	0.163	0.105	44.6%	79.7%	86.9%	0.60	1.10
Sample 5	Fresh	63	1.118	1.257	0.227	0.135	43.6%	81.9%	89.3%	0.65	0.50
Sample 6	Fresh	63	0.625	0.615	0.176	0.120	45.8%	71.4%	80.5%	0.50	0.45
Sample 7	Fresh	63	1.393	1.164	0.081	0.060	45.2%	93.0%	94.8%	0.51	0.50
Sample 8	Fresh	63	1.207	1.100	0.155	0.150	44.0%	85.9%	86.4%	0.51	0.55
Sample 9	Fresh	63	0.845	0.903	0.044	0.050	72.2%	95.1%	94.5%	0.70	0.40
Sample 10	Moderate	63	1.132	1.096	0.048	0.040	49.9%	95.7%	96.4%	0.55	0.65
Sample 11	Moderate/Slight	63	1.909	1.936	0.081	0.075	55.4%	95.8%	96.1%	0.60	0.60
Sample 12	Fresh	63	0.798	0.773	0.082	0.085	38.7%	89.4%	89.0%	0.50	0.45
Sample 13	Completely	63	0.637	0.729	0.038	0.040	13.8%	94.8%	94.5%	0.69	1.00
Sample 14	Slightly	63	1.214	1.151	0.088	0.080	61.4%	92.3%	93.1%	0.60	0.55
Sample 15	Moderate	63	0.768	0.890	0.102	0.085	32.9%	88.6%	90.4%	0.58	0.70

Table 7.12: Results of Variability Leach Testing at 75 µm

Sample	Weathering	P_{80}	Head Au Grade (g/t)		Residual Au Grade (g/t)		Au Extraction (%)			Reagents	
		(µm)	Assay	Calc.	24-h	48-h	Gravity	24-h	48-h	NaCN	Lime
Sample 1	Fresh	75	0.668	0.693	0.117	0.085	50.0%	83.1%	87.7%	0.38	0.35
Sample 2	Fresh	75	4.270	4.169	0.349	0.165	28.6%	91.6%	96.0%	0.43	0.75
Sample 3	Sight/Moderate	75	1.590	1.574	0.067	0.055	65.9%	95.7%	96.5%	0.38	1.10
Sample 4	Fresh/Slightly	75	0.980	0.800	0.177	0.095	44.6%	77.9%	88.1%	0.45	1.00
Sample 5	Fresh	75	1.118	1.257	0.225	0.135	43.6%	82.1%	89.3%	0.56	0.45
Sample 6	Fresh	75	0.625	0.615	0.167	0.120	45.8%	72.9%	80.5%	0.43	0.50
Sample 7	Fresh	75	1.393	1.164	0.113	0.078	45.2%	90.3%	93.3%	0.45	0.45
Sample 8	Fresh	75	1.207	1.100	0.166	0.160	44.0%	84.9%	85.5%	0.47	0.50
Sample 9	Fresh	75	0.845	0.903	0.047	0.045	72.2%	94.8%	95.0%	0.60	0.40
Sample 10	Moderate	75	1.132	1.096	0.057	0.045	49.9%	94.8%	95.9%	0.41	0.55
Sample 11	Moderate/Slight	75	1.909	1.936	0.129	0.115	55.4%	93.3%	94.1%	0.51	0.55
Sample 12	Fresh	75	0.798	0.773	0.090	0.094	38.7%	88.3%	87.9%	0.45	0.50
Sample 13	Completely	75	0.637	0.729	0.078	0.065	13.8%	89.3%	91.1%	0.62	0.90
Sample 14	Slightly	75	1.214	1.151	0.110	0.090	61.4%	90.5%	92.2%	0.55	0.50
Sample 15	Moderate	75	0.768	0.890	0.109	0.085	32.9%	87.7%	90.4%	0.53	0.60

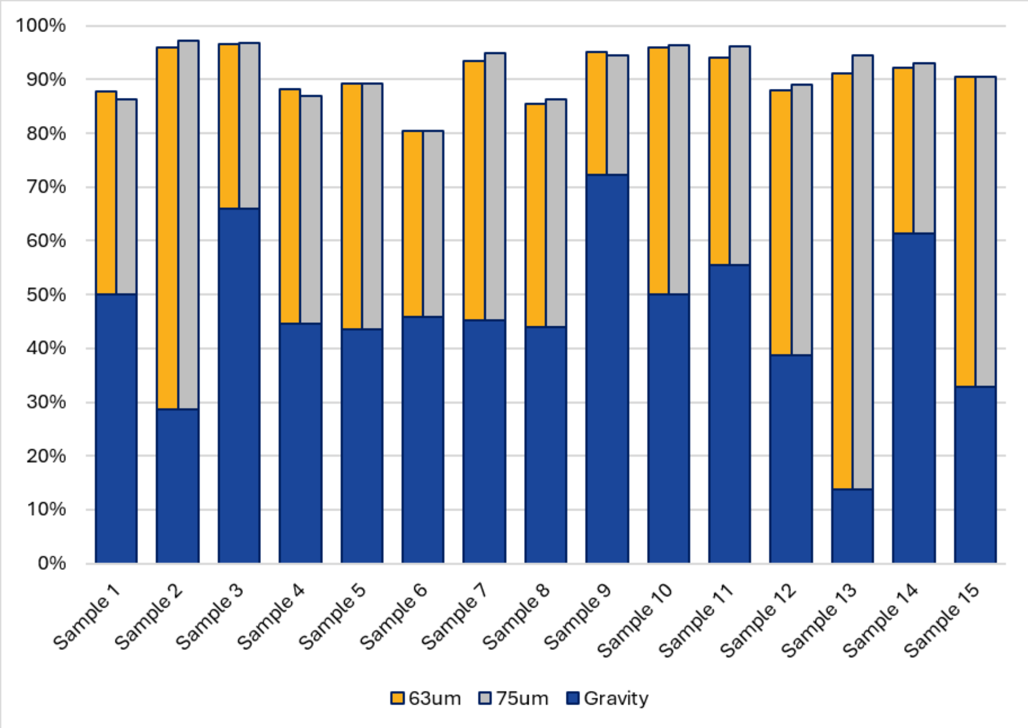


Figure 7.3: Gravity/Leach Summary of All Variability Tests

In addition to these 48-hour extraction results, the results from the paired liquor and solids sub-samples taken at 4, 8, 24 and 48 hours provide a kinetic leaching profile across the leach duration. Figure 7.4 presents the residual gold grade over time and includes the line of best fit from both the 75 µm and 63 µm datasets.

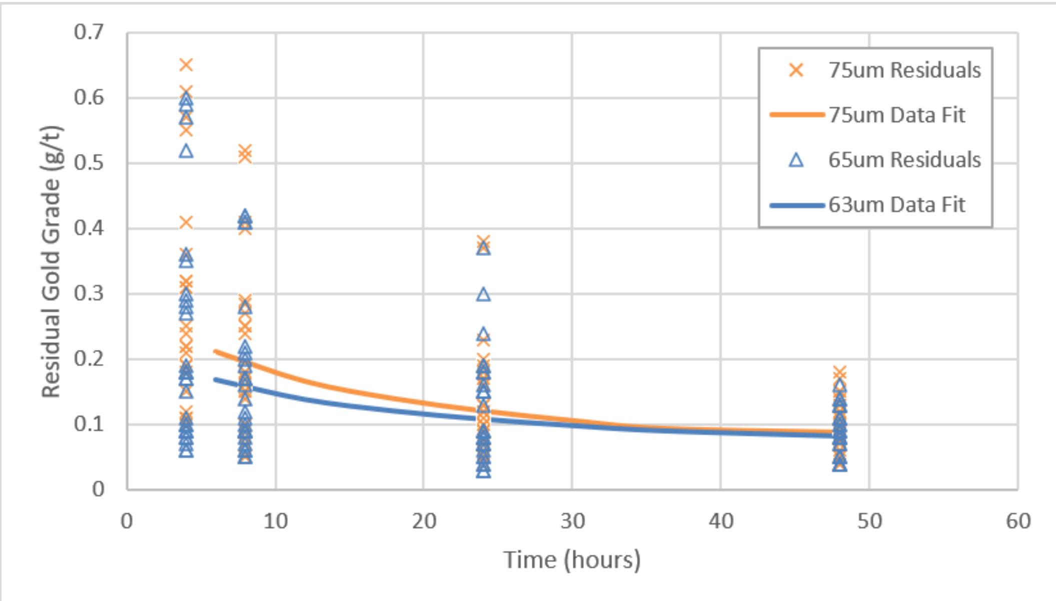


Figure 7.4: Gravity/Leach Performance versus Leach Residence Time

These results demonstrate a clear trend of lower residual gold grade, increased gold extraction, with increased leach residence time. In addition, the results show significant variability between individual samples, due to the nuggety gold effect outlined in Sections 7.2 and 7.4, and therefore the data fit of the results was used to draw conclusions, which could otherwise be "hidden" in the natural error that presents in individual tests.

These data fit results have the following line-of-best-fit equations, indicating that there a reduction in the leach residual grade of 0.005 g/t at 63 µm versus 75 µm:

- 63 µm gold residual (g/t) = -0.0431 x ln (Residence Time -hrs) + 0.2449, minimum 0.083.
- 75 µm gold residual (g/t) = -0.0655 x ln (Residence Time -hrs) + 0.3.283, minimum 0.088.

Analysis of the result also indicated that a fixed tail model, as encompassed in the above lines-of-best-fit, are the only statistically significant relationship that can be used to estimate gold extraction. Table 7.13 presents the calculated extraction across a range of gold head grades.

Table 7.13: Estimated Au Extraction: Impact of Head Grade, Grind Size and Residence Time

Au Head Grade	Au Extraction: P_{80} 63µm		Au Extraction: P_{80} 75µm	
g/t	24-Hours	48-Hours	24-Hours	48-Hours
0.88	87.7%	90.6%	86.4%	90.1%
1.00	89.2%	91.8%	88.0%	91.3%
1.22	91.1%	93.2%	90.2%	92.8%
Fixed Tail Estimate	**0.108 g/t**	**0.083 g/t**	**0.120 g/t**	**0.088 g/t**

7.5.3 Leach Liquor Assays

Detailed assays were undertaken on the liquor from the variability leach tests, presented in Table 7.14. Conclusions that can be drawn from these assays include:

- Relatively low levels of Ag and Cu in solution indicating that is unlikely that there will be an impact on the Au loading level on carbon.
- Low levels of base metals (Fe, Ni, Zn) which influence the reagent consumption in cyanide destruction.
- No measured presence of Hg indicating no specific mercury handling required in the design and no potential health, safety or environmental impacts from mercury.

Table 7.14: Variability Leach Testing – Liquor Assays

Sample	Ag	Al	As	Ca	Co	Cr	Cu	Fe	K	Mn	Na	Ni	Hg	S	Zn
	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
1	0.40	0.58	0.28	19.58	<0.3	<0.1	11.15	24.50	110.00	0.62	486.75	0.23	<1	46.00	2.98
2	0.18	0.78	1.03	26.08	<0.3	<0.1	10.48	20.75	113.25	0.56	549.50	0.70	<1	48.25	2.45
3	0.14	0.95	4.58	20.40	<0.3	<0.1	5.85	11.30	131.25	0.58	495.00	0.15	<1	34.50	2.08
4	0.12	1.00	2.93	18.45	<0.3	<0.1	8.68	24.08	131.50	0.89	607.00	0.47	<1	41.25	2.00
5	0.15	0.93	1.98	19.85	<0.3	<0.1	8.80	21.15	151.75	1.47	491.00	0.40	<1	46.50	1.90
6	0.11	0.78	1.80	19.73	<0.3	<0.1	9.63	22.60	159.75	0.62	452.00	0.40	<1	40.75	1.88
7	0.11	0.90	4.03	19.15	<0.3	<0.1	9.08	35.63	131.00	0.43	600.00	0.60	<1	47.75	1.88
8	0.11	1.63	4.25	18.18	<0.3	<0.1	9.58	43.35	126.50	0.51	535.50	0.70	<1	47.75	2.33
9	0.14	1.25	2.03	19.15	<0.3	<0.1	9.35	23.05	144.25	1.55	517.00	0.13	<1	31.25	2.43
10	0.22	0.83	1.23	17.98	<0.3	<0.1	13.38	20.68	127.75	0.77	548.00	0.35	<1	23.25	2.13
11	0.08	1.08	3.55	20.20	<0.3	<0.1	9.75	26.45	129.50	0.69	662.75	0.65	<1	33.50	2.33
12	0.16	0.95	2.10	19.73	<0.3	<0.1	9.08	23.48	145.75	2.32	550.00	0.50	<1	29.00	2.40
13	0.24	0.30	5.75	26.35	<0.3	<0.1	1.30	23.38	131.75	2.67	426.75	0.00	<1	22.75	1.85
14	0.13	0.23	11.18	27.88	<0.3	<0.1	2.98	20.45	122.50	1.50	418.75	0.40	<1	85.50	1.65
15	0.11	0.53	3.63	22.73	<0.3	<0.1	5.33	23.58	134.25	1.69	409.75	0.00	<1	22.00	1.70
Average	**0.16**	**0.85**	**3.35**	**21.03**	**<0.3**	**<0.1**	**8.29**	**24.29**	**132.72**	**1.12**	**516.65**	**0.38**	**<1**	**40.00**	**2.13**
Median	**0.14**	**0.90**	**2.93**	**19.73**	**<0.3**	**<0.1**	**9.08**	**23.38**	**131.25**	**0.77**	**517.00**	**0.40**	**<1**	**40.75**	**2.08**

7.5.4 Conclusions

The key conclusions from the leaching program included:

- Leach residence time of 48 hours is appropriate, and the longer leach residence time also minimises the gold extraction difference between 63 μm and 75 μm grind size.
- Fixed tail model is the only statistically significant predictor of leach extraction of gold.
- Slurry density of 50% solids is acceptable for leaching.
- Pre-oxidation for four hours with oxygen reduces cyanide consumption.
- Air is suitable for leach aeration with no additional benefit from oxygen aeration.
- No preg-robbing is evident and therefore full CIL is not required.
- Liquor analysis for the leach testing shows no concentration of elements of concern either for carbon loading (Ag, Cu), cyanide detoxification (base metals) or for health, safety and environment (Hg).

7.6 Bulk Leaching Testwork

7.6.1 Carbon Loading

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

The bulk leaching (approximately 13 kg of sample) was conducted at intentionally high cyanide concentrations (above 500ppm NaCN) to ensure that copper loading did not interfere with gold loading on the subsequent carbon loading tests.

Triple carbon contact tests achieved greater than 1,400 ppm gold loading on carbon and equilibrium loading carbon tests with varied carbon concentration (for isotherm development) achieved greater than 2,000 ppm gold loading on carbon from each of the sample. For particularly high-grade samples that generated elevated gold in solution from the leach, the isotherm the maximum gold loading was greater than 10,000 ppm gold.

7.6.2 Oxygen Uptake

Oxygen uptake rate tests were completed on the three composite samples at 50% solids by measuring the rate of oxygen withdrawal from the system after an initial fast addition of oxygen to achieve a dissolved oxygen concentration of 15 to 20 ppm. The oxygen decay was measured for 60 minutes, and the cycle repeated a further three times so that average oxygen uptake can be calculated, and the results are presented in Table 7.15.

Table 7.15: Oxygen Uptake Rate Summary

Sample ID	Average OUR	Max OUR	Avg Oxygen consumed	Cum. Oxygen consumed
	mg/L/min per cycle	mg/L/min	g/t ore/hr	g/t ore/hr
Fresh Low-grade	0.078	0.086	7.048	28.190
Fresh Average-grade	0.106	0.122	9.526	47.630
Weathered	0.098	0.103	8.777	43.883

These tests provided the design point for the oxygen addition to the leach, based on the maximum of three tests.

7.6.3 Tailings Detoxification

The slurry from the triple carbon contact tests was then detoxified successfully to less than 5 mg/L weak acid dissociable (WAD) cyanide using the following conditions:

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

7.7 Thickening and Filtration

Thickening and filtration testing was carried out on three bulk composite samples representing fresh material, weathered material and a blend of fresh and weathered material.

Thickening testwork was conducted to establish design parameters for both the pre-leach and tailings thickening. Testing was performed on ground samples (P_{80} 63 µm) using a continuous laboratory hi-rate thickener at solids loading (flux) rates from 0.25 t/m^{2}h to 1.5 t/m^{2}h. All three composites achieved acceptable overflow clarity and underflow densities exceeding 55% solids at a solids loading rate of 1.0 t/m^2/h, demonstrating favourable thickening characteristics.

Filtration testwork was undertaken to establish design parameters for tailings dewatering, as filtered tailings disposal was considered likely due to the limited availability of water within the Project region. The program evaluated thickened material generated during the thickening testwork using both vacuum and pressure filtration techniques, utilising laboratory vacuum filtration and single-chamber pressure filtration equipment.

Vacuum filtration was not successful in achieving an acceptable product for filtered stack tailings disposal on any of the samples tested.

In contrast, pressure filtration achieved acceptable product cake consistency on all bulk samples using a 50 mm chamber depth, achieving product moisture in the range of 14.9% to 15.7% for all samples and a range of chamber depths.

The chamber flux rate was consistently greater than or equal to 240 kg DS/m^2/h for all samples, supporting the suitability of pressure filtration for the Project.

The bulk composite sample comprising entirely of shallower weathered material produced filtration and thickening results comparable to those obtained from the fresh and blended composites, indicating that there is no separate design considerations required for the two identified lithologies.

7.8 Other Testwork

Materials handling testwork was carried out at a specialist laboratory to inform detailed stockpile and chute design. This testing did not measure significant differences in flowability between the samples and peak cohesive strength was measured in the 8% to 9% moisture range. Although not considered significant, the weathered material is slightly more cohesive, but all materials tested are considered suitable for mass flow and funnel flow storage facilities.

From the testing, apron feeders were recommended for ROM ore, however belt feeders were considered suitable for crushed ore with a top size less than 75 mm. Whilst top size must be considered at transfer points, bulk streams with self-cleaning large lumps and fines can be designed with dead boxes.

Rheology testing was conducted on each of the three bulk samples (weathered, fresh and blend) with all demonstrating similar behaviour and presenting as shear thinning, but with increasing viscosity as the slurry density increases. At the target leach operating point of greater than 50% solids the viscosity is within typical ranges and was used for agitator and pumping design.

Metallurgical testwork undertaken to date has utilised laboratory tap water at the South African testing facility. Testing with water from the Project water supply is currently planned, however, at the time of testing the source of water for the Project had not been finalised. Initial analysis of the water from the proposed range of sources, with the most saline being classified as brackish with a salinity of 3,500 mg/L, does not indicated a high level of risk for gold processing.

7.9 Geometallurgical and Design Parameters

The metallurgical testwork programme established the following key geometallurgical and process design parameters for the Project:

- Fresh and weathered ore samples tested exhibited comparable physical characteristics in testing (Section 7.8), with no concerns identified relating to material handling, mass flow, storage or transfer points. Both material types are considered suitable for gyratory crushing and stockpile reclamation with appropriate feeder selection and chute design.
- A design grind size (P_{80}) of 75 µm was selected based on the analysis in Section 7.5.2. Comminution circuit modelling demonstrated that the proposed circuit has sufficient installed power to achieve an average grind product size (P_{80}) of 63 µm for material of average ore hardness at the design duty power draw.

- Comminution circuit (crushing and milling) power demand, derived from the comminution parameters presented in Table 7.4 and comminution circuit modelling, varies by lithology, with estimated specific energy consumptions of:
  - Fresh ore, 32.26 kWh/t.
  - Weathered ore, 25.43 kWh/t.
- Gold recovery by gravity concentration of approximately 45%, based on the median gravity recovery achieved across all variability samples outlined in Section 7.4.
- Four hours of pre-oxidation with oxygen addition as demonstrated by the testwork outlined in Section 7.5.1.1.
- Air sparging for leach tank aeration, as supported by the testwork outlined in Section 7.5.1.2.
- Conventional thickening technology capable of achieving thickener underflow densities of 62% solids (Section 7.7), with rheology testing confirming suitability for leaching at 50% solids (Section 7.8).
- Pressure filtration of tailings, as outlined in Section 7.7, to produce a filtered tailings with moisture content of approximately 13-16% (w/w).
- Cyanide consumption in leaching of 0.44 kg/t based on median consumption recorded across all variability tests detailed in Section 7.5.2, adjusted to reflect the inclusion of pre-oxidation stage as described in Section 7.5.1.1.
- Lime consumption in leaching of 0.50 kg/t based on the median consumption recorded across all variability tests detailed in Section 7.5.2.
- Leach residual grades defined by the following relationships between residue grade and leach residence time, developed from the variability leach testwork results outlined in Section 7.5.2:
  - 63 µm gold residual (g/t) = -0.0431 x ln (Residence Time -hrs) + 0.2449, Minimum 0.083 g/t.
  - 75 µm gold residual (g/t) = -0.0655 x ln (Residence Time -hrs) + 0.3283, Minimum 0.088 g/t.
- Effective cyanide destruction can be achieved using the operating conditions established in Section 7.6.2:
  - 3 g SO_2/g WAD CN, added as $Na_2S_2O_5$.
  - 10 mg/L Cu excess, added as $CuSO_4.5H_2O$.
  - pH 8.5.
  - 2-hour residence time.

These parameters form the basis for process plant design, equipment sizing, and operating cost estimation for the DFS.

8 MINING AND ORE RESERVES

8.1 Mining Strategy

The Project development strategy was based on establishing a practical and optimised open pit mining sequence that supports early gold production while maintaining a practical material movement profile. The strategy builds on the work completed during the Scoping Study, with the DFS schedule being developed based on the updated sub-blocked Mineral Resource model and at a greater level of detail to improve the timing of pre-production mining, ore release, stockpile management and mining fleet requirements.

The mine schedule was optimised to maximise recovered gold production during the first five years of operations, thereby enhancing project economics and accelerating capital repayment. To facilitate this outcome, higher value ore was preferentially scheduled where practicable, while maintaining operational flexibility through the use of a simple LOM fixed grade binning strategy.

A scheduling cut-off grade of 0.4 g/t Au was adopted. This cut-off is higher than the estimated economic breakeven cut-off grade and is intended to support the early production strategy. Indicated Mineral Resources above 0.4 g/t Au were treated as the main process plant feed, however it was further binned into medium-grade ore between 0.4 g/t Au (inclusive) and 0.7 g/t Au and high-grade ore equal to or greater than 0.7 g/t Au.

Material above the economic breakeven cut-off grade but below 0.4 g/t Au was classified as low-grade mineralised material for scheduling purposes. This material will be mined when required by the pit sequence and stockpiled separately as mineralised waste. The mineralised waste provides a potential future source of mill feed, either towards the end of the LOM or through any future increase in plant milling capacity.

For scheduling purposes, the Inferred Mineral Resources above 0.4 g/t Au were also binned separately. The Inferred Mineral Resources will be stockpiled separately to avoid unintended blending with Ore Reserves during reclaim.

The DFS design was developed as a staged open pit. Strategic schedule scenarios were used to test stage boundaries and ore release timing, with the preferred staging arrangement selected to maximise recovered gold over the first five to seven years while maintaining a practical mining sequence. The design includes eight mining stages, including two starter pits, developed to provide early access to the higher value areas of the deposit.

A sensitivity assessment was also completed to consider the potential future underground mining. To reduce potential interference with future underground access or stoping, and to support the required minimum mining areas at the pit floors, the pit designs have ignored the bottom 10 m of optimisation shells. This adjustment did not reduce open pit value and was retained in the DFS design basis.

Pre-production mining is planned to establish sufficient ore stocks prior to plant commissioning and to provide waste for early project earthworks. The pre-production mining target is to mine and stockpile approximately three months of ore prior to commencement of processing, together with waste required for construction of the ROM pad and other early surface water management infrastructure.

8.2 Operations Strategy

The mining operating strategy is based on contract mining, with the owner retaining responsibility for overall management, statutory obligations, geology, grade control interpretation, mine planning and scheduling. The mining contractor will be responsible for the execution of the open pit mining works in accordance with the mine plan and the agreed scope of work.

The contractor's scope will include:

- Clearing and stripping of pit, WRD, stockpile and ROM pad areas.
- Establishment and maintenance of ex-pit roads.
- Bulk earthworks for the ROM pad behind structural fill and embankment works constructed by the bulk earthworks contractor.
- Grade control drilling and sampling.
- Drill and blast.
- Load and haul of ore and waste.
- ROM pad and stockpile rehandle.
- Pit dewatering.
- Dust suppression.
- Rehabilitation of final waste dump surfaces and obsolete haul roads.
- Provision of infrastructure to support the contractor's activities as outlined in Section 12.5.

Waste mined during the pre-production period will be used for construction of the ROM pad, initial surface water management protection for the pits and TSF and other infrastructure where directed. After the initial construction requirements are met, waste will be hauled to the nearest waste rock dump. During operations, waste will also be used for additional lifts, expansion of and progressive closure of the TSF.

Ore will be hauled to the ROM pad or to designated stockpiles adjacent to the pit and processing plant area. Ore delivered to the ROM pad will be tipped onto stockpile fingers or direct tipped to the crusher where practical. Stockpiled ore will be reclaimed using a front-end loader or loader and truck combination.

Pit dewatering will be undertaken by the mining contractor using in-pit sumps and pumps. Water will be pumped to a surface water dam adjacent to the open pit or used directly for dust suppression.

8.3 Mining Method

Conventional open pit truck-and-shovel mining was selected for the Project. This method is considered appropriate for the near-surface gold mineralisation and is consistent with common open pit mining practice in Namibia.

Grade control drilling will be completed in advance of mining using RC drilling. The grade control drilling will comprise RC holes drilled at a 60° inclination to an approximate vertical depth of 20 m. Samples will be collected at 2.5 m vertical intervals, with a splitter used to reduce sample mass to approximately 2 kg.

The mining method was designed to selectively mine ore zones using backhoe-configured excavators on a 2.5 m flitch to minimise dilution and ore loss. Ore mining will be undertaken using 220 t class backhoe excavators supported by 90 t payload haul trucks. Bulk waste mining will be undertaken using 300 t class face shovels matched with 140 t payload haul trucks.

Production drilling will be undertaken using Epiroc SmartROC D65 or equivalent down-the-hole drill rigs for primary blast-hole drilling and Epiroc SmartROC T45 or equivalent top hammer drill rigs for presplit and trim blasting applications. Grade control drilling will be completed using contractor-supplied reverse circulation drill rigs in accordance with the grade control requirements.

All weathered and fresh in-situ material was assumed to require drill and blast. Blasting will be undertaken on 10 m benches, with presplits and trim shots used adjacent to final walls to reduce backbreak. The pits were designed with berms at 10 m vertical intervals in weathered material and 20 m vertical intervals in transitional and fresh rock in line with the geotechnical design requirements outlined in Section 5.1.

Waste rock dumps will be constructed in 10 m lifts. Rehabilitation of final waste rock dump surfaces will commence as soon as practicable using dozers to reprofile the landform, front-end loaders and mine trucks to place topsoil, and contour ripping to reduce run-off and erosion.

The proposed open pit mining fleet is summarised in Table 8.1. Fleet selection was based on contractor submissions received during the DFS, with the primary loading and hauling fleet adjusted to suit the selective ore and bulk waste mining strategy. The excavator capacities are presented by operating weight class, haul trucks by payload class, and drill rigs by drilling method and hole diameter range. Fleet quantities represent the estimated peak or steady-state mining requirements over the life of the operation.

Table 8.1: Proposed Open Pit Mining Fleet

Category	Type	Model (or equivalent)	Specification		Number
			Unit	Value	
Loading	Excavator	Cat 6020 B	op. weight (t)	220	2
	Face Shovel	Cat 6030 FS		300	3
	FEL	Cat 992 G	bucket (m^3)	11.5	2
Hauling	Dump Truck	Cat 777	payload (t)	90	10[1]
		Cat 785		140	32[1]
Drilling	DTH Production drill	Epiroc SmartROC D65	hole dia.(mm)	110-229	7
	Top hammer drill	Epiroc SmartROC T45		89-140	2
	Grade control drill	Contractor Specified		60° holes to 20 m	1
Support	Track Dozer	Komatsu D375A	power (kW)	391	4
		Cat D8T		264	2
	Motor Grader	Caterpillar 18M	blade length (m)	5.5	3
	Water Truck	75 kL water truck	tank size (kL)	75	2
	Roller/ Compactor	Dynapac CA6000D	op.weight (t)	20	1
	Rockbreaker	Caterpillar 352 with hydraulic hammer	op. weight (t)	50	1
	Fuel Truck	20 kL diesel truck	tank size (kL)	20	3
	Ore Transport Truck	Caterpillar 777	payload (t)	90	6
	Dewatering Pumps[2]	Sykes XH250	flow @ max head (L/s @ m)	300 @ 240	1
		Sykes XH100		75 @ 140	2
		Sykes XH80		25 @ 160	2
		Sykes MH150		160 @ 50	1
		Sykes MH130		75 @ 100	10
Total Fleet					**81**

Notes:

1. Truck numbers are the maximum operating requirements over the LOM and will be less during early mining and later stockpile processing.
2. Dewatering pumps excluded from total fleet number.

8.3.1 ROM and Stockpile Operations

The ROM operating strategy is based on ore being hauled to a central ROM pad adjacent to the primary crusher. Where practical, Ore will be direct tipped into the crusher feed bin, otherwise it will be stockpiled on designated ROM fingers for subsequent reclaim and feed to the crusher. For purposes of the DFS it is assumed that approximately 30% of ore will be direct tipped from the pit with the remainder rehandled via the ROM fingers. All ore reclaimed from stockpiles will be direct tipped.

Reclaiming of ore to the crusher will be undertaken using a front-end loader or a front-end loader and truck combination, depending on the stockpile location and haul distance. During periods of reduced ore availability, or towards the end of the mine life, low-grade stockpiles may be reclaimed to supplement crusher feed and maintain plant utilisation. The timing and extent of low-grade stockpile processing will be determined as part of the final processing strategy and prevailing economic conditions.

8.4 Pit Optimisation

Preliminary optimisation scenarios were completed during the DFS using an interim Mineral Resource model to support strategic schedule development and pushback selection. The final optimisation used for pit design was completed using the block model for the Mineral Resource estimate released co-currently with the DFS and as outlined in Section 4.8 and Table 4.1. The selected optimisation shell was then used as the target shell for the ultimate pit and stage designs.

A block-based dilution method was applied to the resource model prior to optimisation. The approach swaps a lateral skin of material between adjacent 5 m by 5 m by 5 m blocks in both the east-west and north-south directions. The total skin width applied was 0.5 m, equivalent to transferring 0.25 m of material from each block's edge to its adjacent block and is considered appropriate for the selected mining method, equipment sizes and mine design. The transferred skin carries the specific gravity and metal content of the source block.

The purpose of the dilution method is to smooth grades across mining bench edges and to mimic the dilution expected when blasting and mining ore mark-outs that are larger than the current 5 m by 5 m selective mining unit. The method allows sufficient additional dilution while retaining spatial control of metal and material expected to move between adjacent ore and waste blocks.

Table 8.2 compares the resource at 0.4 g/t Au cutoff, before and after dilution. The equivalent ore loss is calculated from the reduction in contained metal after dilution, expressed as equivalent tonnes at the undiluted grade. The equivalent dilution is calculated as the additional diluted material required to reconcile the diluted tonnes after allowing for the equivalent ore loss. On a total Indicated/Inferred Resource basis, the applied dilution results in an equivalent ore loss of around 2% and equivalent dilution of around 4%.

Table 8.2: Dilution and Ore Loss Assessment (at 0.4 g/t Au Cut-off)

Resource Category	Undiluted			Diluted			Ore Loss (%)	Dilution (%)
	Tonnes (Mt)	Au (g/t)	Au (koz)	Tonnes (Mt)	Au (g/t)	Au (koz)		
Indicated	86.8	0.882	2,462	88.7	0.854	2,436	1%	3%
Inferred	85.0	0.804	2,197	86.5	0.768	2,135	3%	5%
Total	**171.8**	**0.843**	**4,659**	**175.2**	**0.812**	**4,571**	**2%**	**4%**

Table 8.3 summarises the key parameters used for the final pit optimisation. The optimisation was based on Indicated material only, with a minimum cut-off grade of 0.4 g/t Au enforced in Whittle. Low-grade mineralised material below 0.4 g/t Au and Inferred material were excluded from the optimisation ore category and reported as waste for optimisation purposes.

Table 8.3: Pit Optimisation Parameters

Area	Item	Unit	Value	Basis
Basic Inputs	Resource model	-	June 2026 block model	Final DFS optimisation model
	Resource category	-	Indicated only	Low-grade and Inferred reported as waste
	Minimum cut-off grade	g/t Au	0.4	Enforced in Whittle
	Processing capacity	Mtpa	5.25	Plant throughput
	Capital allowance	US$m	445.7	SS project capital allowance
Dilution	Dilution skin	m	0.5	
	Equivalent ore loss	%	2.4	
	Equivalent dilution		4.1	
Geotechnical	Slope angle - oxide	degrees	34.4	All
	Slope angle – fresh		39	East walls
			45	North, south and west walls
Mining costs	Load and haul waste	US$/bcm	Variable	Bench and material dependent
	Drill and blast waste	US$/bcm	Variable	Bench and material dependent
Ore costs	Grade control	US$/t ore	0.12	Included in ore mining premium
	ROM rehandle		0.59	Included in ore mining premium
	Stockpile reclaim		0.74	Differential for stockpile reclaim
	Ore mining premium (OMP)		Variable	Ore differential Drill & Blast and Load & Haul cost
Processing	Processing variable cost		10.36	Updated DFS processing input
	General and administration		2.35	Updated DFS input
	Tailings cost		1.07	Updated DFS input
	Total process cost		13.78	Processing only
	Total process cost + OMP		15.55	Input to Whittle
Revenue	Base gold price	US$/oz	3,000	Revenue factor applied in shell selection
	Transport and refining	US$/oz	2.75	Deducted from revenue
	Royalty and export levy	%	4	Deducted from revenue
	Discount rate	%	5	Discounted cashflow basis

The SCN411 nested shell results are shown in Figure 8.1. The selected shell is Shell 48 at a revenue factor of 0.87, equivalent to a gold price of US$2,600/oz against the US$3,000/oz base price. The selected shell lies near the plateau of the average discounted cashflow curve and captures the main high-value portion of the pit while avoiding lower-value incremental material in larger shells.

Shells immediately below the selected shell retain similar average value but contain less ore and metal, while shells above the selected shell add material for limited incremental value and a declining average case DCF trend. Shell 48 was therefore selected as a practical guide for pit design because it provides a strong value outcome, retains the required project scale, and maintains a close relationship between shell size, contained metal and material movement.

Following selection of Shell 48, the ultimate pit and stage designs were completed using the selected shell as the design guide. Practical design constraints, including ramps, geotechnical berms, minimum mining widths, stage access and final bench access, were then applied to convert the Whittle shell into a mineable open pit design.

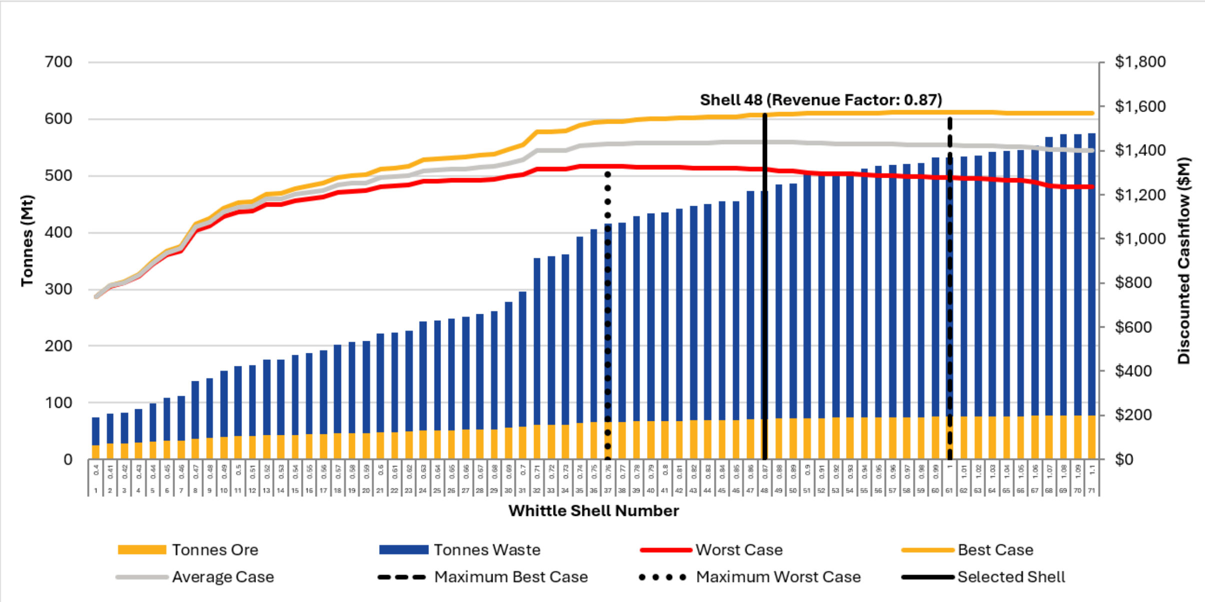


Figure 8.1: SCN411 Pit-by-Pit Optimisation Results

The selected shell results are summarised in Table 8.4. The tonnes reported in this table include only Indicated Resources above the 0.4 g/t Au cut-off grade. Low-grade mineralised material and Inferred Resources are included as waste material for optimisation reporting.

Table 8.4: Selected Optimisation Shell Results

Item	Unit	Value
Scenario	-	SCN411
Selected shell	#	48
Revenue factor	-	0.87
Equivalent gold price	US$/oz	2,600
Indicated Resources Mined	Mt	71
Indicated Resource Grade	g/t Au	0.875
Contained gold	koz	1,997
Recovered gold	koz	1,808
Waste	Mt	402.5
Total material	Mt	473.5
Strip ratio	waste:ore	5.67
Mine life	years	13.5
Best case DCF	US$m	1,562
Worst case DCF	US$m	1,316
Average case DCF	US$m	1,439

Table 8.5 compares the selected optimisation shell with the ultimate pit design. The difference is reported as design variance relative to the selected shell. For consistency with the optimisation, only Indicated Resources above 0.4 g/t Au is reported as ore in the design comparison with low-grade mineralised material and Inferred material are treated as waste.

Table 8.5: Pit Design versus Selected Shell

Source	Ore (Mt)	Au (g/t)	Au (koz)	Waste (Mt)	Total (Mt)	Strip Ratio
Selected Optimisation Shell	71.0	0.875	1,997	402.5	473.5	5.67
Ultimate Pit Design	69.8	0.87	1,953	401.2	471.0	5.74
Difference	-1.6%	-0.5%	-2.2%	-0.3%	-0.5%	1.3%

The ultimate pit design is closely aligned with the selected shell, with the design reporting approximately 1.6% less ore tonnes, 2.2% lower contained gold, almost same waste tonnes and it contains as little as 0.5% less total material than the selected Whittle shell. This provides a reasonable reconciliation between the optimisation shell and the practical design, recognising the inclusion of ramps, geotechnical criteria and minimum mining widths in the design process.

8.5 Pit Design

The pit design process converted the selected shell into practical mineable pit and stage designs using the applicable geotechnical criteria, ramp design parameters, minimum mining widths and stage sequencing requirements. The focus was on maintaining alignment with the selected shell while providing practical ramp access, suitable operating widths, controlled final wall geometry and sufficient stage access to support the early production strategy.

8.5.1 Strategic Scheduling and Pit Staging

Pit staging was developed iteratively using a combination of optimisation shell guidance, practical pit design considerations and strategic schedule testing. Maptek Evolution Phase was used to support the development and assessment of mining pushbacks. The resulting stage concepts were then tested using Evolution Strategy to assess the impact of stage sequence, mining rate, vertical advance, ore release and stockpile strategy on project value and early recovered gold production.

Approximately 30 strategic schedule scenarios were assessed during the DFS. The scenario work tested alternative stage layouts, mining rates, vertical advance limits, plant throughput sensitivities and ore definition assumptions.

The final design comprises eight mining stages. Stages 1 and 2 form the initial starter pits which will be developed to provide early access to the highest value ore areas. Subsequent stages were designed to maintain ore continuity, control waste stripping and maintain practical ramp access as the pits are expanded. The general arrangement of the stages is shown in Figure 8.2. The stage configuration provides two starter pits, with immediate cutbacks allowing higher value ore to be accessed early while lower value cutbacks can be deferred until later in the mining sequence.

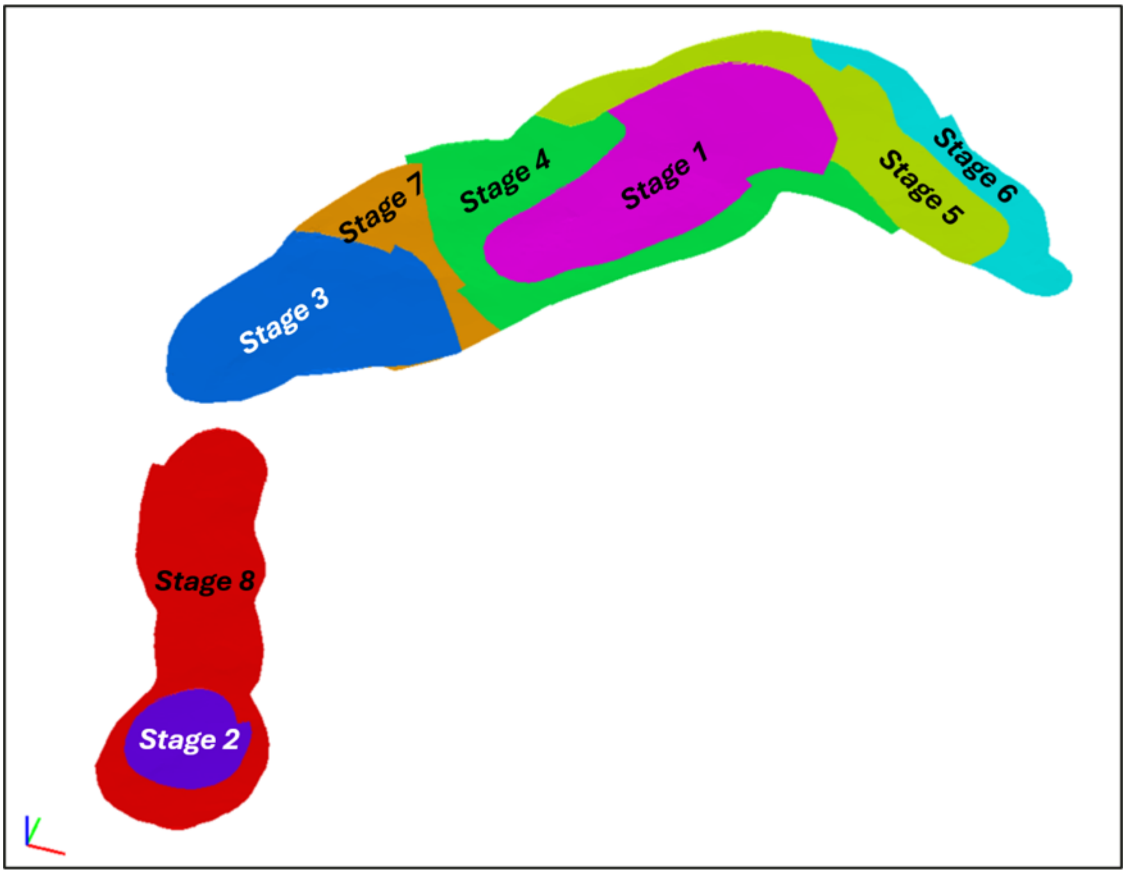


Figure 8.2: Pit Stage Layout

8.5.2 Basis of Pit Design

The pit design parameters were based on the selected optimisation shell, geotechnical slope criteria, the selected mining fleet, practical ramp geometry and minimum mining width requirements. The design criteria used for the ramp and road designs in the open pit are summarised in Table 8.6.

The open pit geotechnical criteria, as discussed in Section 5.1, are summarised in Table 5.1. These parameters define the bench height, batter angle, berm width and indicative inter-ramp slope geometry by weathering domain and wall orientation.

Table 8.6: Ramp and Road Design Criteria

Item	Basis	Value
Two-way pit ramp width	Cat 785D operating width and 33R51 tyre basis	31 m
One-way pit ramp width	Cat 785D operating width and 33R51 tyre basis	18 m
Two-way surface road width	Double-bunded surface road	38 m
Ramp gradient	Dual-lane pit ramps	10%
Ramp gradient	Single-lane final bench access	12%
Minimum cutback width	Target operating width	60 m
Minimum mining width	Limited local operating width	30 m
Minimum pit floor mining width	Pit base design	25 m
Good-bye cut	At the pit floors	5 m

The pit design includes the following additional operating assumptions:

- All weathered, transitional and fresh in-situ material is assumed to require drilling and blasting.
- Final walls are to be protected using presplit and trim blasting where required.
- Ore mining is planned on 2.5 m flitches to support selective mining and grade control.
- The pit floors include 5 m deep good-bye cuts below the working floor to support recovery of final ore blocks prior to pit abandonment.

8.5.3 Pit and Stage Designs

The ultimate pit design is shown in Figure 8.3. The ultimate pit is split into two separate pits, the northern pit and the southern pit. The northern pit is approximately 2.3 km long and 690 m wide at its widest point. The northern pit reaches to 775 mRL, or approximately 300 mbgl. The southern pit is approximately 1.4 km long and 430 m wide at its widest point. The southern pit reaches to 885 mRL, or approximately 170 mbgl.

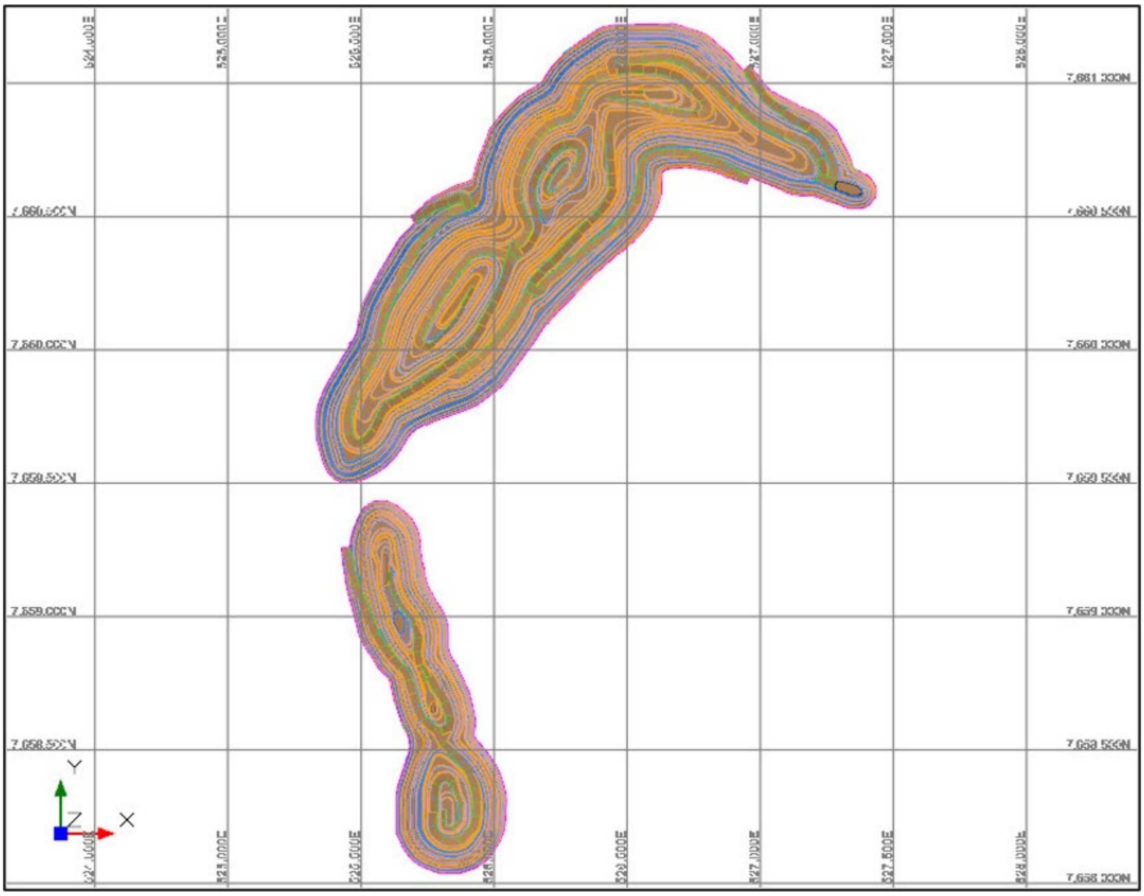


Figure 8.3: Ultimate Pit Design

Stage 1 is the starter pit in the northern pit (Figure 8.4) and is approximately 1 km long and 450 m wide reaching a depth of 160 mbgl. Access is from the east side of the pit close to the plant using a dual-lane ramp from surface before a dual-lane switchback connects to a second dual-lane slot access. The final 250 m long single-lane ramp provides access to the final benches.

Stage 2 is the starter pit the southern pit (Figure 8.5) and is approximately circular with a width of 360 m and a depth of approximately 85 mbgl. Access is via a dual-lane ramp from surface at wrapping clockwise around the pit before merging into single-lane access towards the pit stage floor.

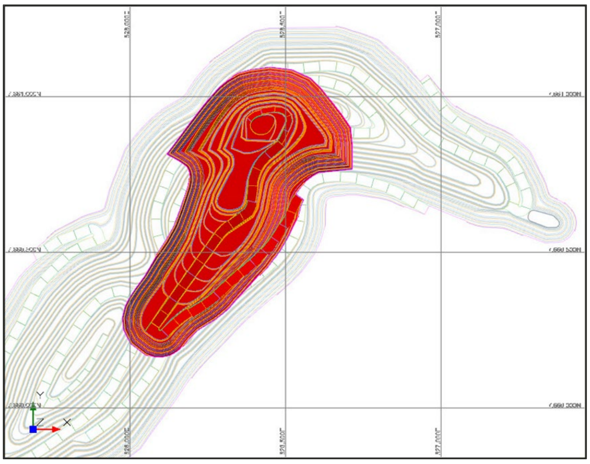


Figure 8.4: Stage 1

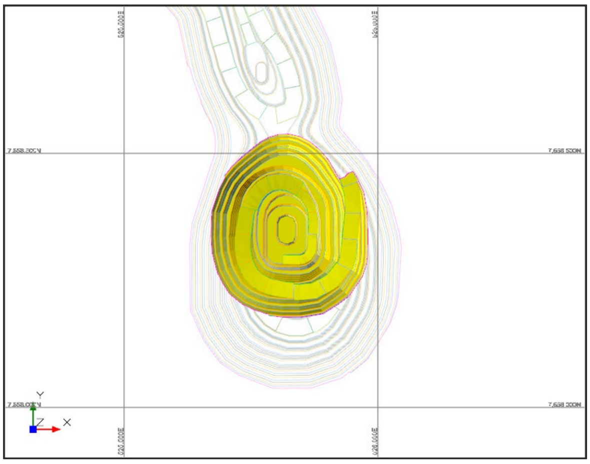


Figure 8.5: Stage 2

Stage 3 (Figure 8.6) is an initial cutback on the south of the northern pit approximately 800 m long and 500 m wide. The pit access faces north, which is beneficial for minimising ex-pit haulage costs for both ore and waste. The ramp system located predominantly on the northern temporary wall continues west and merges with the final pit wall ramp at approximately 995 mRL. The ramp leaves the final wall at approximately 915 mRL and is located on the northern temporary wall again for the remaining lower benches, turning into single-lane access near the pit floor.

Stage 4 (Figure 8.7) is a cutback to Stage 1, extending it to the final eastern wall and partly to the final western wall. The cutback maintains at least 65 m mining width from Stage 1 and targets high value ore immediately below the Stage 1 starter pit. Stage 4 is approximately 1 km long and approximately 500 m wide at surface. The stage is relatively deep, reaching approximately 230 mbgl, but does not yet reach the final ultimate pit floor.

Stage 4 makes use of two separate ramp systems from surface. An ore access with a short haul to the plant and a waste access on the temporary wall which exits West toward the WRD.

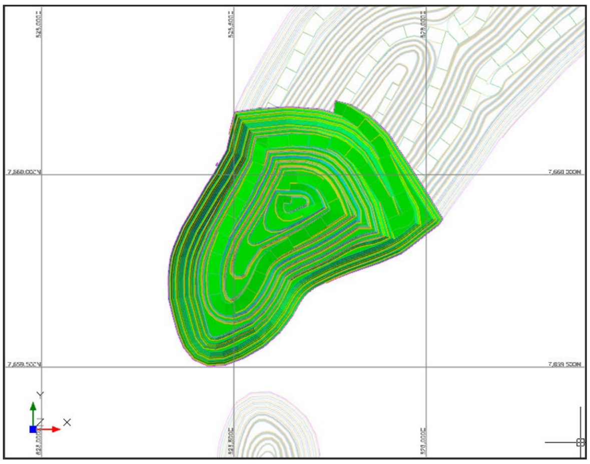


Figure 8.6: Stage 3

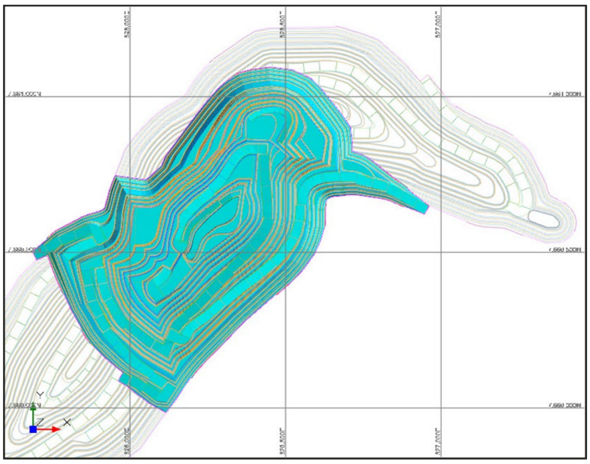


Figure 8.7: Stage 4

Stage 5 (Figure 8.8) is a cutback to both Stage 4 and Stage 1, extending them to the final western wall and the ultimate pit floor. The mining width between Stage 5 and the earlier stages is generally greater than 80 m, with a few pinch points of approximately 50 m near the base of Stage 1. Access is via a temporary dual-lane ramp on the northern wall before joining a permanent dual-lane ramp on the western wall of the ultimate pit. From this point, the remaining access to the pit floor is via permanent ramps, mostly along the western wall. The surface pit exit on the northern wall is positioned to provide short waste haulage to the WRD while maintaining a reasonable ore haulage route to the ROM pad.

Stage 6 (Figure 8.9) is the northernmost cutback of the northern pit. It pushes Stage 5 walls back to the final northern walls and targets the value left below that stage. The pushback is generally wider than 90 m to 100 m, with a few pinch points around 60 m wide. The access concept is similar to Stage 5.

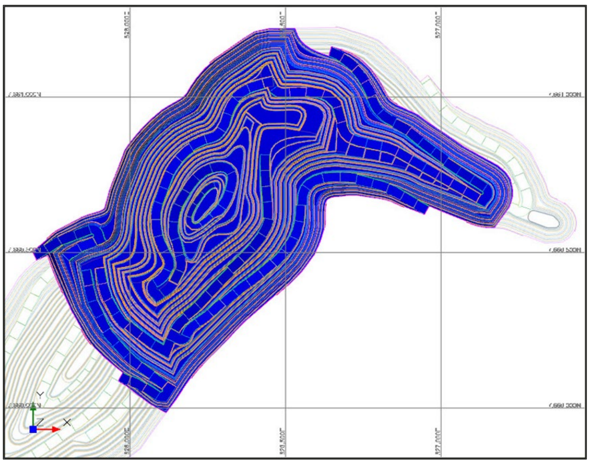


Figure 8.8: Stage 5

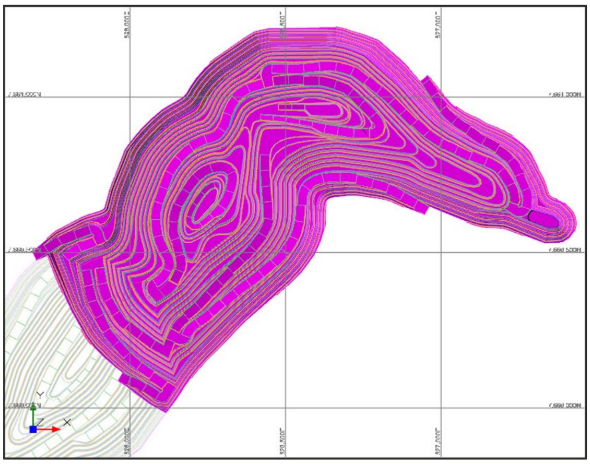
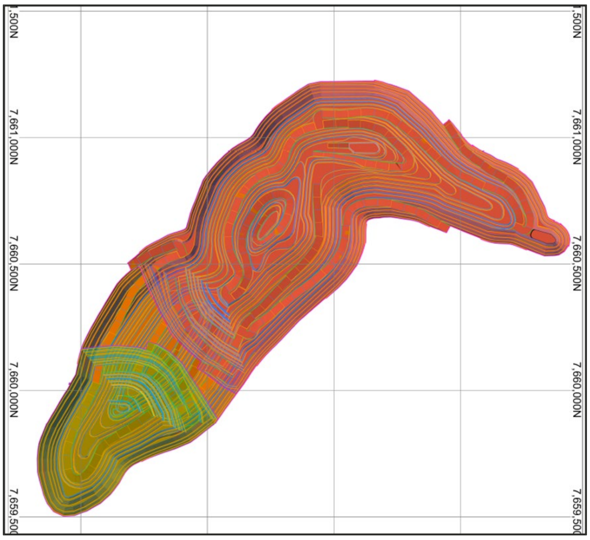


Figure 8.9: Stage 6

Stage 7 (Figure 8.10) completes mining of the northern pit by mining the lower value pushback between Stage 3 to the south and Stage 4 to the north. The stage uses a combination of retreat-mining, temporary accesses and permanent ramp segments.

Stage 8 (Figure 8.11) is the final cutback of the Stage 2 starter pit in the southern pit. It expands the starter pit to the final shell boundaries and is a lower value cutback that is deferred until late in the mine life. The pit is relatively long, approximately 1.4 km in length and approximately 300 m wide at surface. The pushback is generally wide, although one narrow pinch point of approximately 30 m is present on the western wall between starter pit and final pit wall. Access is from the northern end with a ramp down the western final wall to a saddle point where it branches into northern and southern pods. The northern pod continues anticlockwise on the eastern wall, transitioning to single-lane access as it nears the pit floor at 920 mRL. The southern pod is accessed by a separate clockwise dual-lane ramp that transitions to single-lane access at as it nears the final pit floor at 885 mRL.



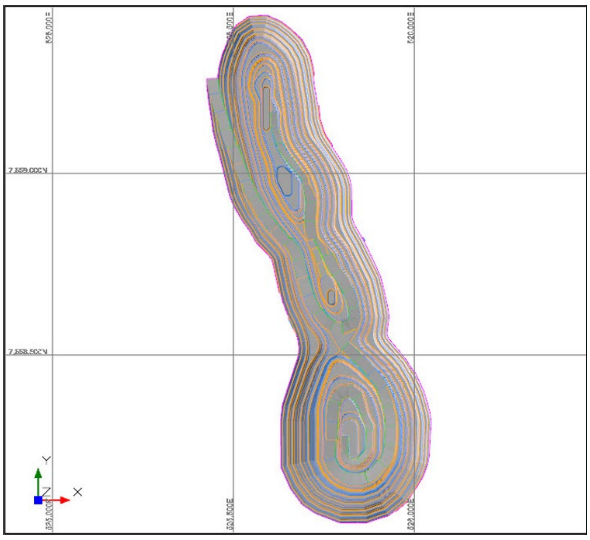


Figure 8.10: Stage 7

Figure 8.11: Stage 8

Representative sections through the main mineralised zones show the relationship between the pit walls, staged designs and mineralisation. Figure 8.12shows the section locations. The sections were arranged to cover the northern pit and the southern pit, including areas where the stage sequence and final pit limits are most important for understanding the design.

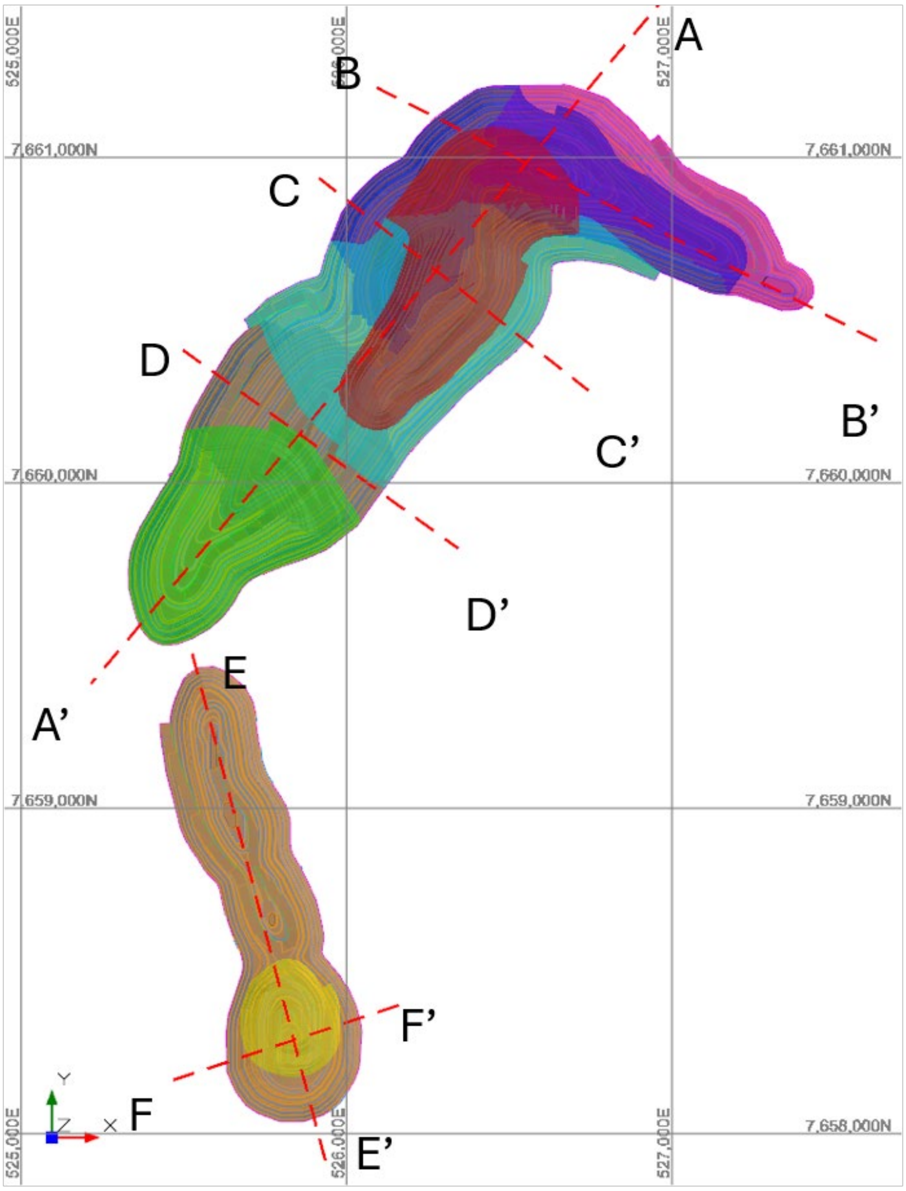


Figure 8.12: Representative Sections Through Main Mineralisation and Pit Design

Sections A-A' and B-B' (Figure 8.13 and Figure 8.14) cut through the main northern pit and illustrate the relationship between the selected pit geometry, stage designs and the mineralised zones through the northern part of the deposit.

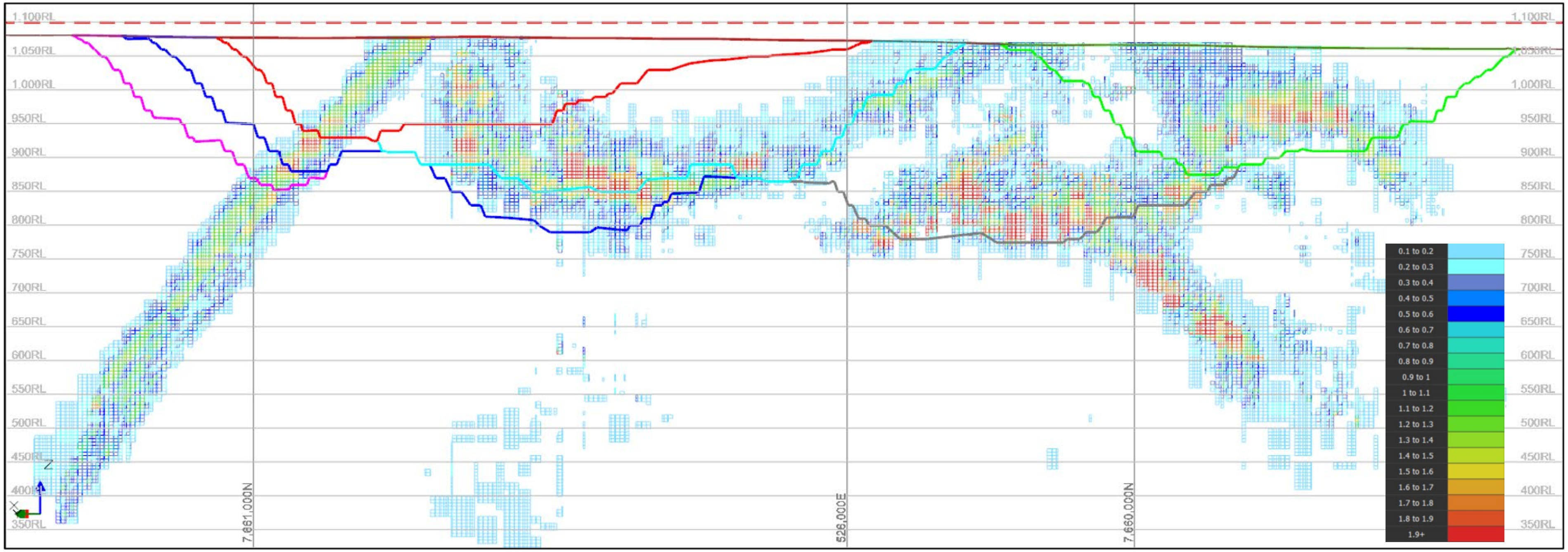


Figure 8.13: Section A-A'

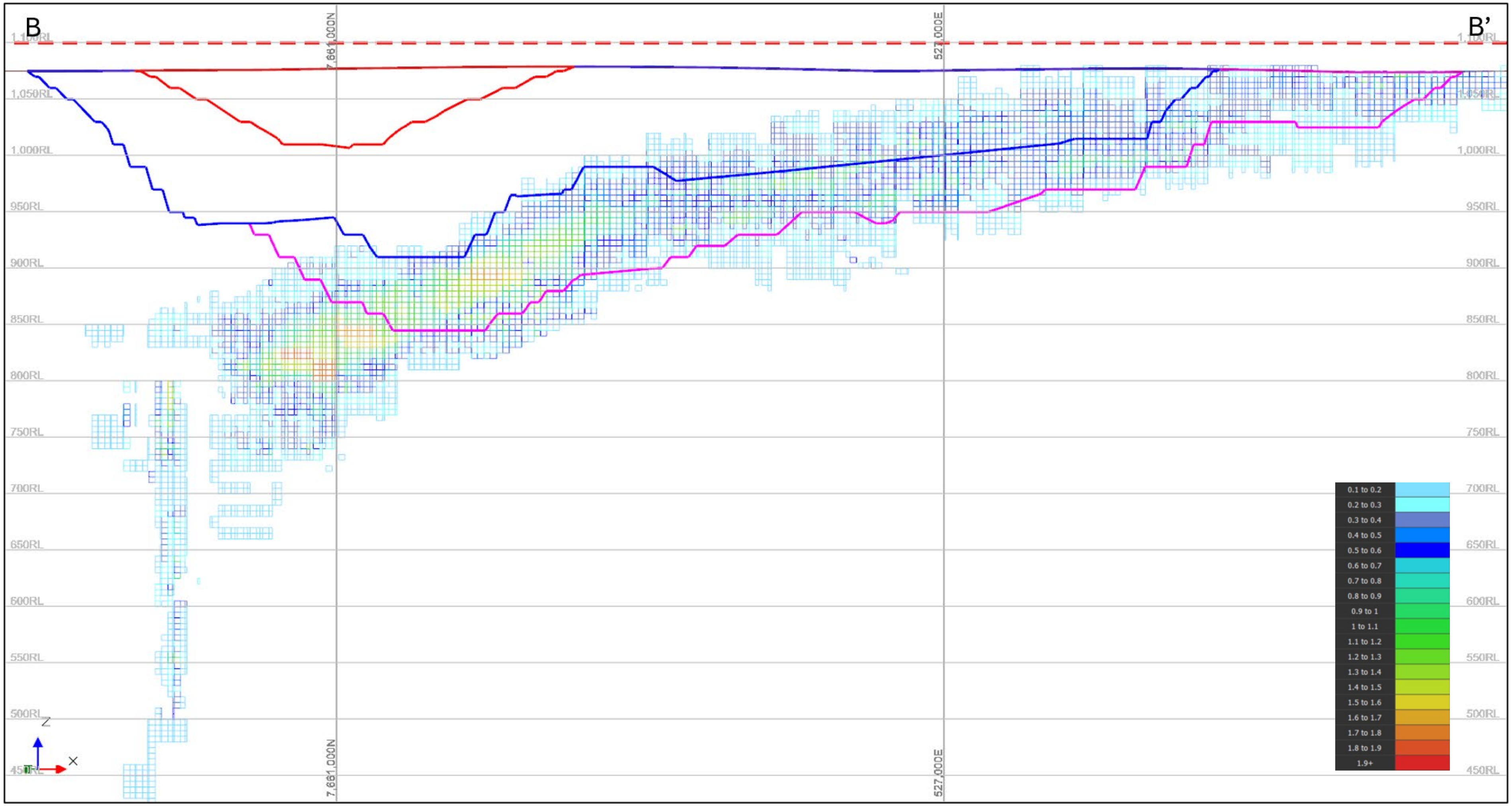


Figure 8.14: Section B-B’

Sections C-C' and D-D' (Figure 8.15 and Figure 8.16) provide cross-pit views through the central and deeper areas of the northern pit, where section D-D' intersects with the lower value Stage 7 with most value realised at the base of the pit and sections C-C' representing an interaction with Stages 1, 4 and 5 proving how the onion skin type expansion on the western wall is helping to defer waste mining while targeting shallower mineralisation in earlier stages.

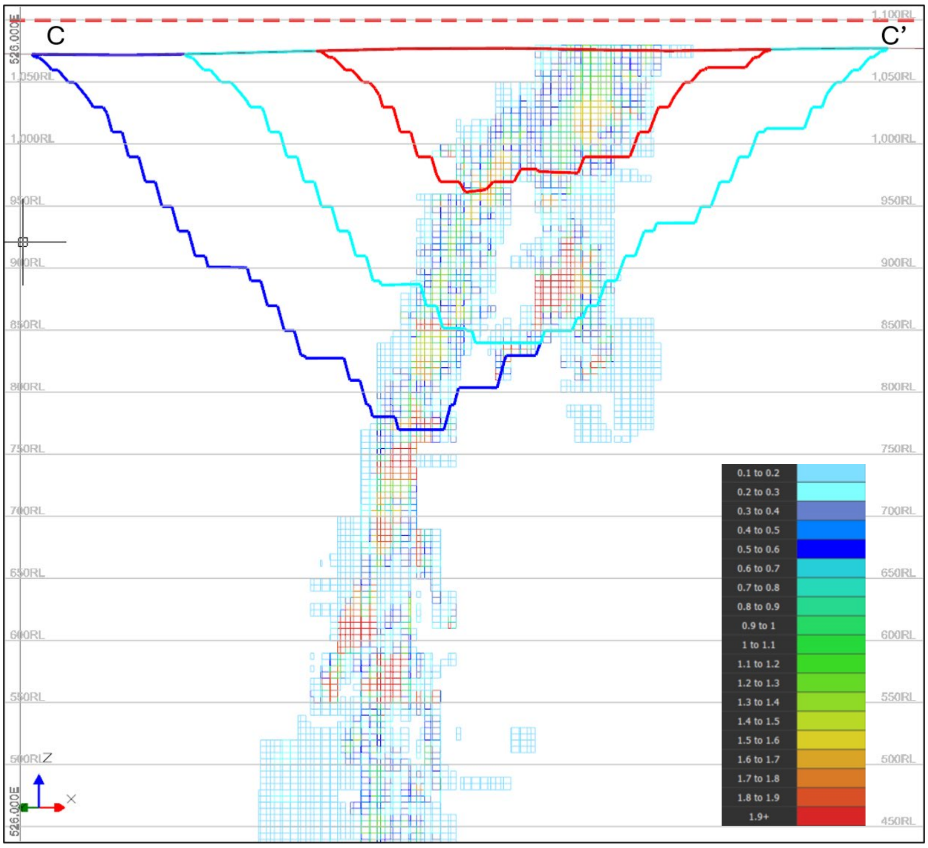


Figure 8.15: Section C-C'

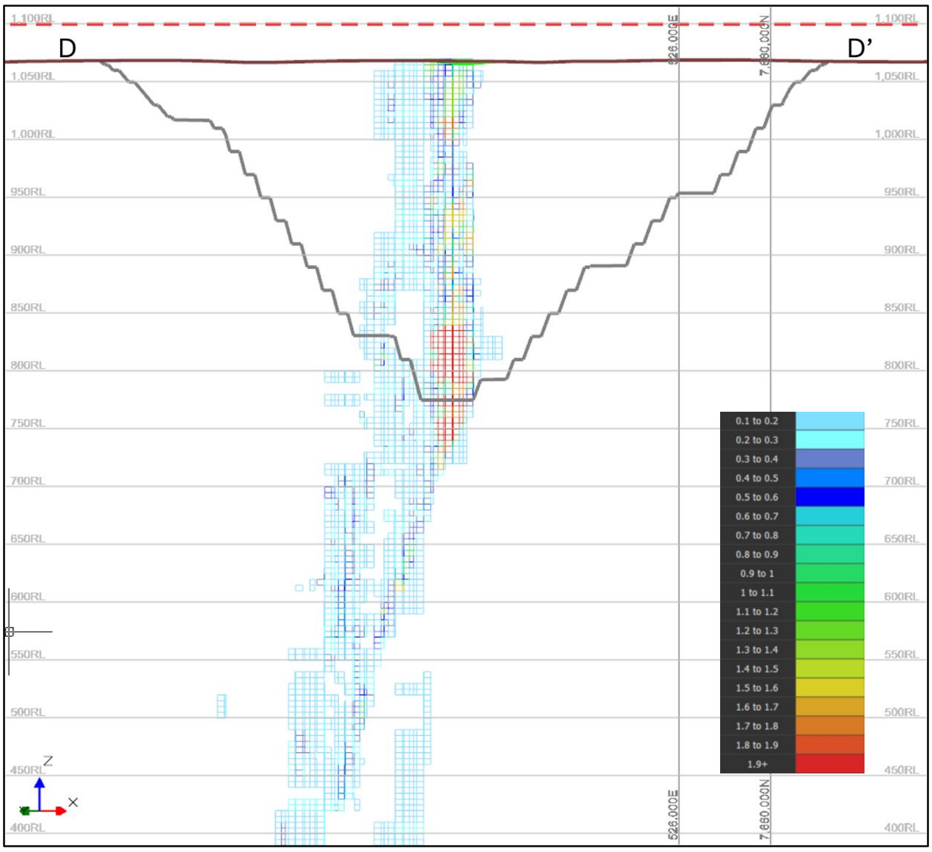


Figure 8.16: Section D-D'

Sections E-E' and F-F' (Figure 8.17 and Figure 8.18) provide representative views through the southern pit and the broader mineralised domain, showing the Stage 2 starter pit and the final pit boundaries relative to the mineralisation and the optimisation shell.

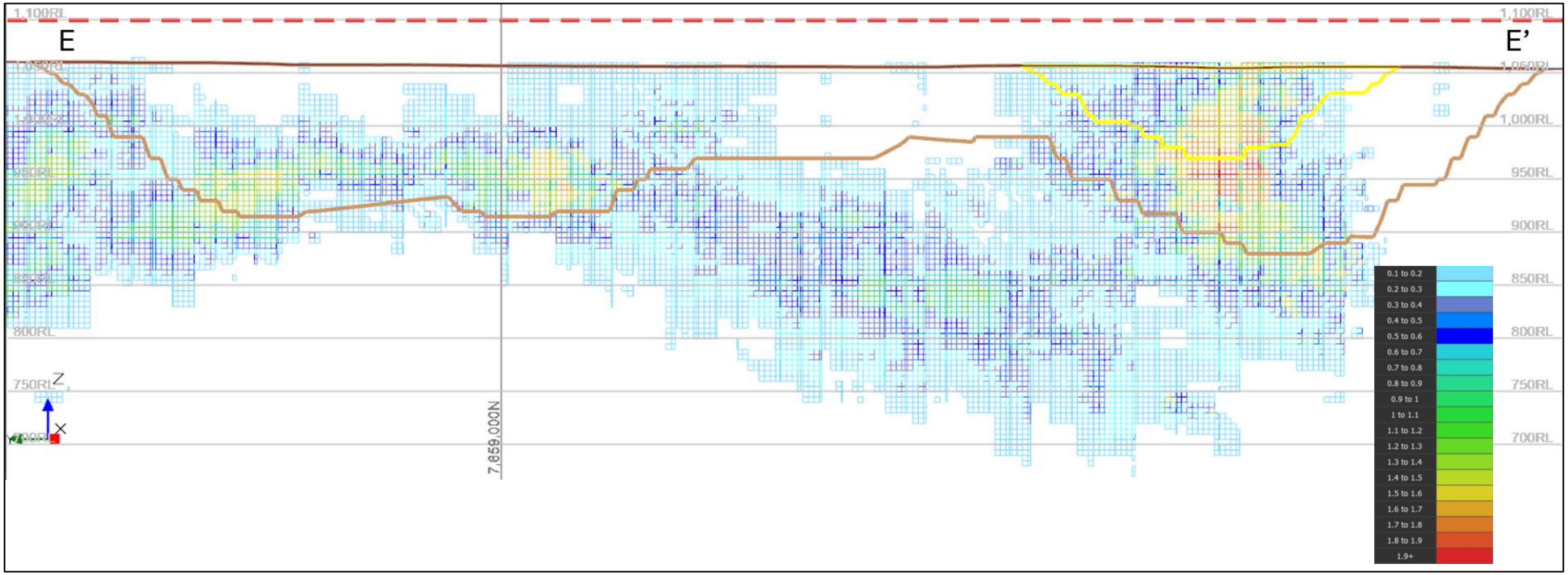


Figure 8.17: Sections E-E'

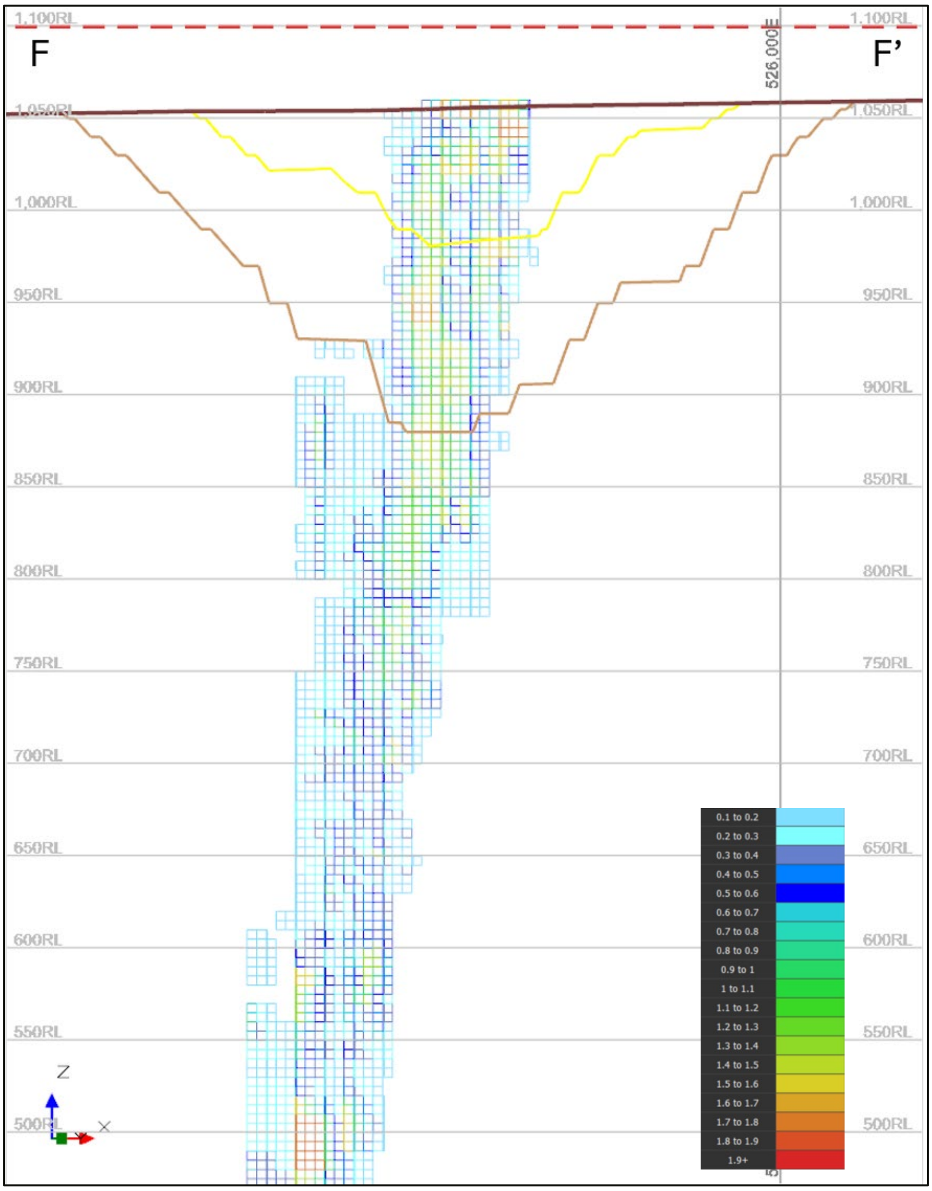


Figure 8.18: Section F-F’

8.5.4 Mining Inventory

The mining inventory contained within the ultimate pit design is summarised by material type in Table 8.7. The inventory is reported separately for Indicated Mineral Resources, low-grade mineralised material, Inferred Mineral Resources and waste to ensure transparent tracking and reporting of each material category throughout the mine planning and scheduling process.

Table 8.7: Ultimate Pit Design Inventory by Material Type

Material Type	Tonnes (Mt)	Volume (Mbcm)	Au (g/t)	Au (koz)	Reporting
Indicated Mineral Resources (>=0.4 g/t)	69.84	25.88	0.87	1,953	Ore
Inferred Mineral Resources (>=0.4 g/t)	3.10	1.16	0.77	77	Inferred Mining Inventory
Indicated & Inferred Mineral Resources (<0.4 g/t, above breakeven)	20.78	7.71	0.34	227	Mineralised Waste
Waste	377.29	140.53	0.05		Waste
Total	**471.01**	**175.28**			**Pit Inventory**

The pit inventory by pit stage is presented in Table 8.8.

Table 8.8: Pit Design Inventory by Stage

Stage	Ore			Mineralised Waste			Inferred Mining Inventory			Waste (Mt)	Total Material (Mt)	Strip Ratio
	Tonnes (Mt)	Au (g/t)	Au (koz)	Tonnes (Mt)	Au (g/t)	Au (koz)	Tonnes (Mt)	Au (g/t)	Au (koz)			
1	16.74	0.88	473	3.66	0.34	40	0.85	0.71	19	32.76	54.01	2.23
2	1.75	0.93	52	0.5	0.34	5	0.32	0.94	10	4.72	7.29	3.16
3	7.49	0.93	224	1.97	0.34	22	0.23	0.85	6	45.01	54.7	6.31
4	11.84	0.89	339	3.83	0.33	41	0.57	0.78	14	76.01	92.24	6.79
5	9.9	0.83	264	3.18	0.34	35	0.41	0.83	11	58.54	72.04	6.28
6	5.94	0.73	139	1.71	0.34	19	0.27	0.82	7	32.57	40.49	5.82
7	7.7	1	247	2.72	0.33	29	0.07	0.74	2	76.91	87.4	10.35
8	8.49	0.78	213	3.2	0.34	35	0.38	0.66	8	50.76	62.83	6.4
Total	**69.84**	**0.87**	**1,953**	**20.78**	**0.34**	**227**	**3.1**	**0.77**	**77**	**377.29**	**471.01**	**5.74**

1. Strip ratio calculated considering mineralised waste and inferred mining inventory as waste.

8.5.5 Waste Rock Dumps

The WRD design is based on locating waste storage close to the open pits while maintaining suitable haulage access to the ROM pad, plant area and filtered tailings storage facility. The WRD location wrapping around the pit and TSF also acts as the main surface water management structure for the pits, plant and other infrastructure.

Waste mined during pre-production will be used for construction of the ROM pad, required stormwater bund around the starter pits and TSF, but still within the WRD footprint and the filtered tailings storage facility starter embankment.

TSF designs indicate that approximately 0.83 Mm^3 of waste rock is required for engineered TSF embankments and staged raises, with an additional approximately 0.93 Mm^3 required for progressive waste rock cladding of the TSF. These cladding works are expected to be completed progressively as an operational mining activity and integrated with the surrounding WRD landform where practical.

The waste rock dump design criteria are summarised in Table 8.9. The final rehabilitated geometry assumes broad berms and flatter batter angles to support long-term stability and rehabilitation requirements.

Table 8.9: Waste Rock Dump Design Criteria

Item	Units	Value
Construction Lift Height	m	10
Construction Face Angle	°	36
Intermediate Construction Berm	m	10.2
Construction Berm Width Near Design Limit	m	22.2
Final Rehabilitated Berm Width	m	8
Final Rehabilitated Batter Angle	°	18
Final Rehabilitated Overall Slope Angle	°	16
WRD Ramp Width	m	31
WRD Ramp Gradient	%	10
Topsoil Replacement Depth	mm	150

Figure 8.19 shows the construction and rehabilitated waste rock dump profiles. The profile illustrates the 10 m construction lifts, steeper construction faces and wider construction berms before reprofiling to a final rehabilitated landform with 18° batters, 8 m berms and an approximate 16° overall slope.

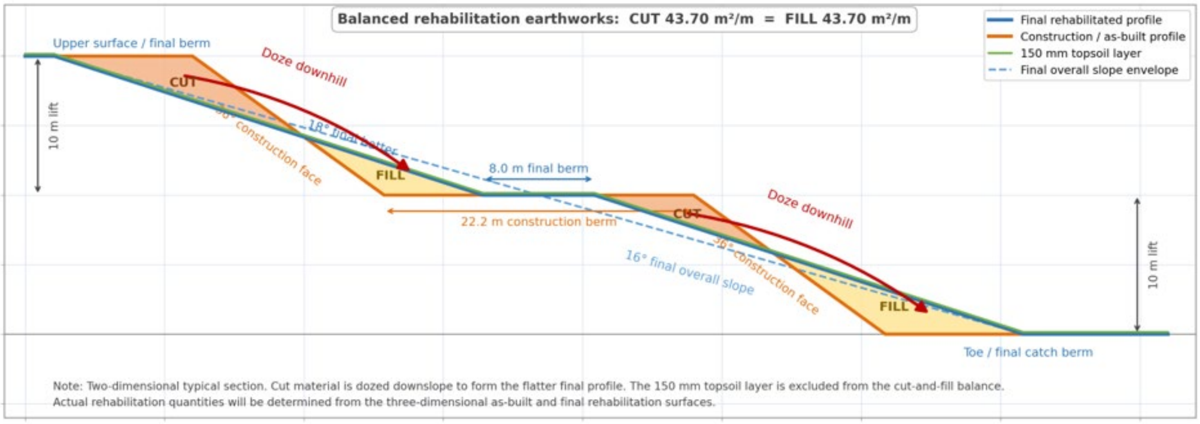


Figure 8.19: Waste Rock Dump Typical Construction and Rehabilitated Profile

Figure 8.20 shows the general mine layout showing the open pits, ROM pad, WRD and TSF.

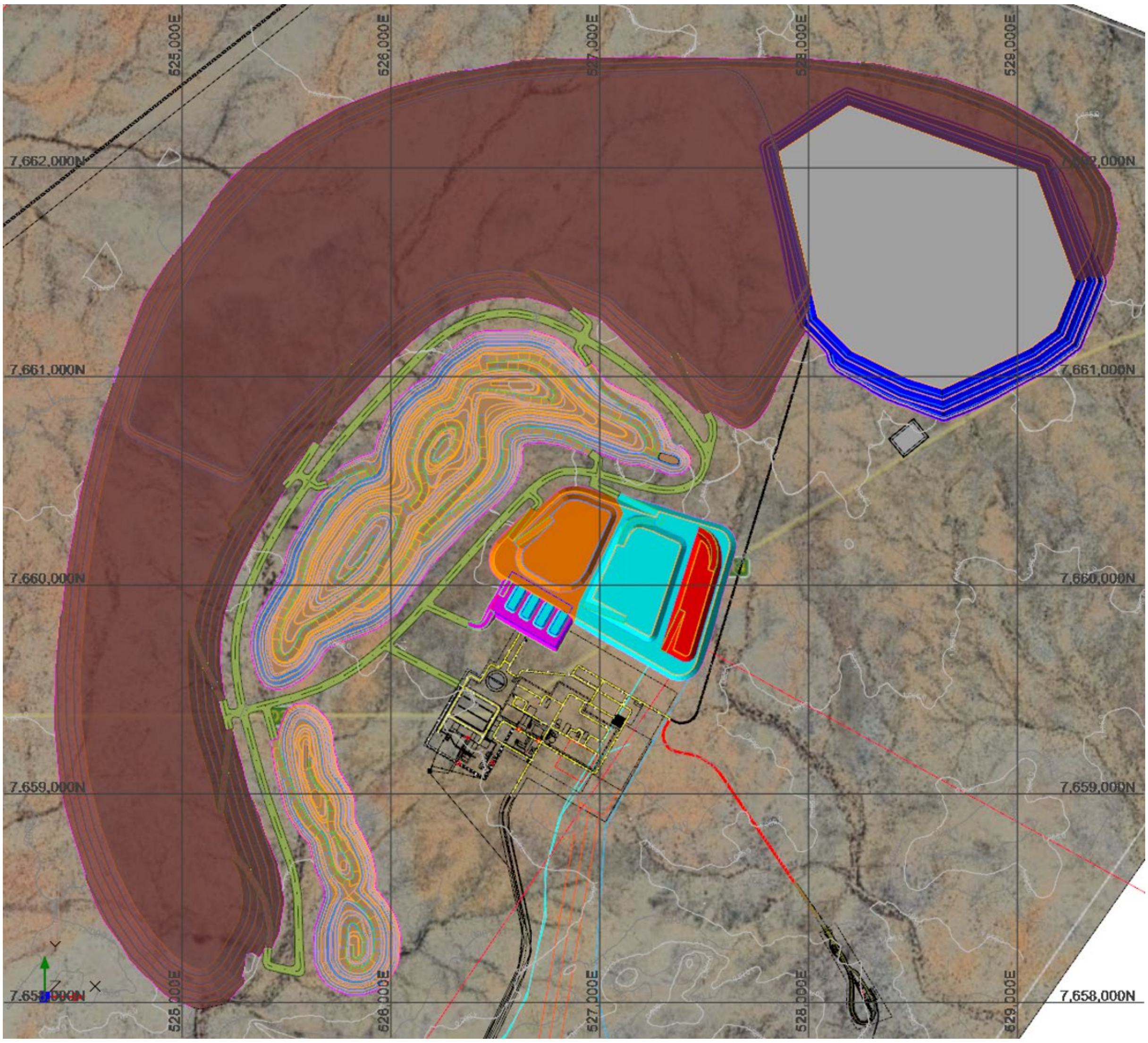


Figure 8.20: General Mine Layout Showing Pits, WRD, ROM Pad and TSF

8.6 Production Scheduling

The mining schedule, based on the mining inventory, includes mining rate ramp up over four months as the digging units progressively mobilised over this period to match the opening of mining areas within the initial pits.

Mining activities are scheduled to commence three months prior to plant commissioning, with commencement of processing designated as Year 1. Open pit mining is completed by Year 11, after which stockpile reclaim provides mill feed through to Year 14. Annual schedule outputs are presented below to summarise the material movement, stage sequence, mill feed, stockpile balance and recovered gold profile.

Over the life of the operation, the schedule includes the mining of approximately 471.0 Mt of material, excluding stockpile reclaim, comprising 377.3 Mt of waste and 93.7 Mt of mineralised material. The process plant treats 72.9 Mt at an average grade of 0.87 g/t Au, producing approximately 1.84 Moz of recovered gold.

8.6.1 Scheduling Approach and Constraints

The detailed schedule was developed using Hexagon MinePlan Schedule Optimiser® with the objective of maximising discounted cash flow subject to a range of operational constraints. These included a maximum mining rate of 54.4 Mtpa, based on 5,400 operating hours and the fleet outlined in Table 8.1.

Pre-production and the first three years of production were scheduled monthly, Years 4 to 6 were scheduled quarterly, and the remainder of the mine life was scheduled annually.

8.6.2 Production Schedule

Summary of the annualised mining schedule was shown in Table 8.10. The schedule fully utilises the mining capacity across Years 1 to 7, after which the annual mining rate reduces to approximately half during Years 8 and 9 and then gradually reduces through to the end of Year 11.

Mining is completed in Year 11, while plant feed is maintained through stockpile reclaim until Year 14. Figure 8.21 shows total material movement by year illustrating the type of the material mined and reclaimed over the life of mine.

Table 8.10: Annual Mining Schedule

	Units	Total	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Mineralised Material																	
Ore	Mt	**69.8**	1.1	9.9	8.3	7.4	8.3	6.8	5.0	5.4	5.3	5.6	4.7	2.2			
Inferred Mining Inventory	Mt	**3.1**	0.2	1.1	0.2	0.2	0.1	0.3	0.1	0.6	0.2	0.0	-	0.2			
Total Mining Inventory	**Mt**	**72.9**	1.3	11.1	8.5	7.6	8.3	7.1	5.0	6.0	5.5	5.6	4.7	2.4			
Au Grade	g/t	**0.87**	0.76	0.84	0.91	0.85	0.81	0.87	0.93	0.90	0.95	0.89	0.72	0.93			
Mineralised Waste																	
Mineralised Waste	Mt	**20.8**	0.4	2.7	1.7	2.1	2.7	2.2	1.6	2.3	1.7	1.7	1.2	0.4	-	-	-
Au Grade	g/t	**0.34**	0.34	0.34	0.34	0.34	0.34	0.33	0.33	0.33	0.33	0.34	0.34	0.34			
Total Mineralised Material	**Mt**	**93.7**	1.7	13.8	10.2	9.6	11.0	9.3	6.7	8.2	7.2	7.2	5.9	2.8	-	-	-
Waste	Mt	**377.3**	5.9	40.0	43.6	44.2	42.8	44.5	47.3	45.6	26.1	22.7	12.6	1.9	-	-	-
Total Material	**Mt**	**471.0**	7.7	53.8	53.8	53.8	53.8	53.8	54.0	53.8	33.3	30.0	18.5	4.7	-	-	-
Strip Ratio[1]	-	**4.03**	3.40	2.91	4.26	4.58	3.90	4.79	7.08	5.54	3.62	3.14	2.14	0.68			
Reclaim	Mt	**20.9**	-	0.3	0.3	0.7	-	-	0.4	0.2	-	0.1	0.6	2.8	5.2	5.2	5.0

1. Strip ratio considers waste:mineralised material (including ore, inferred mining inventory and mineralised waste).

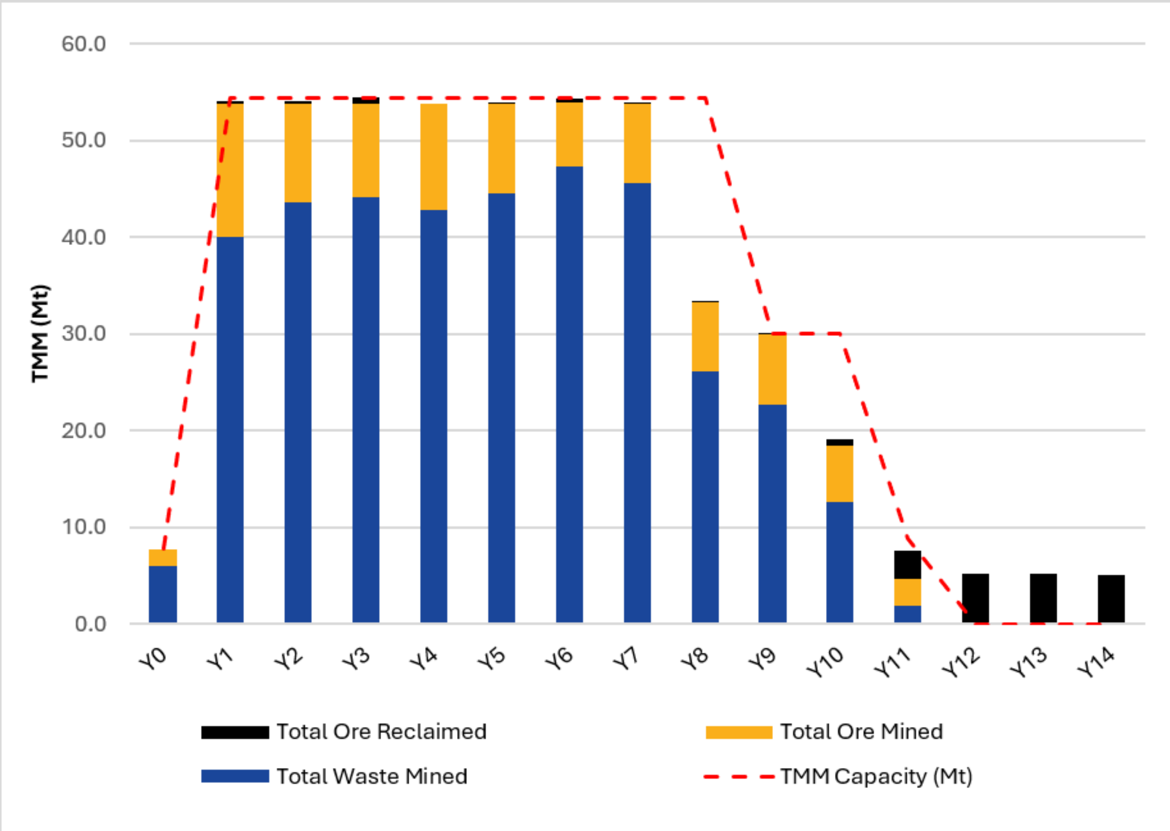


Figure 8.21: Annual Total Material Movement

The mining movement by pit stage is shown in Figure 8.22. The starter pits provide the initial material movement and ore release, with subsequent cutbacks sequenced to maintain mill feed and defer lower-value material where practical. Table 8.11 shows the stages' start and end mining period along with annual vertical advance peaks.

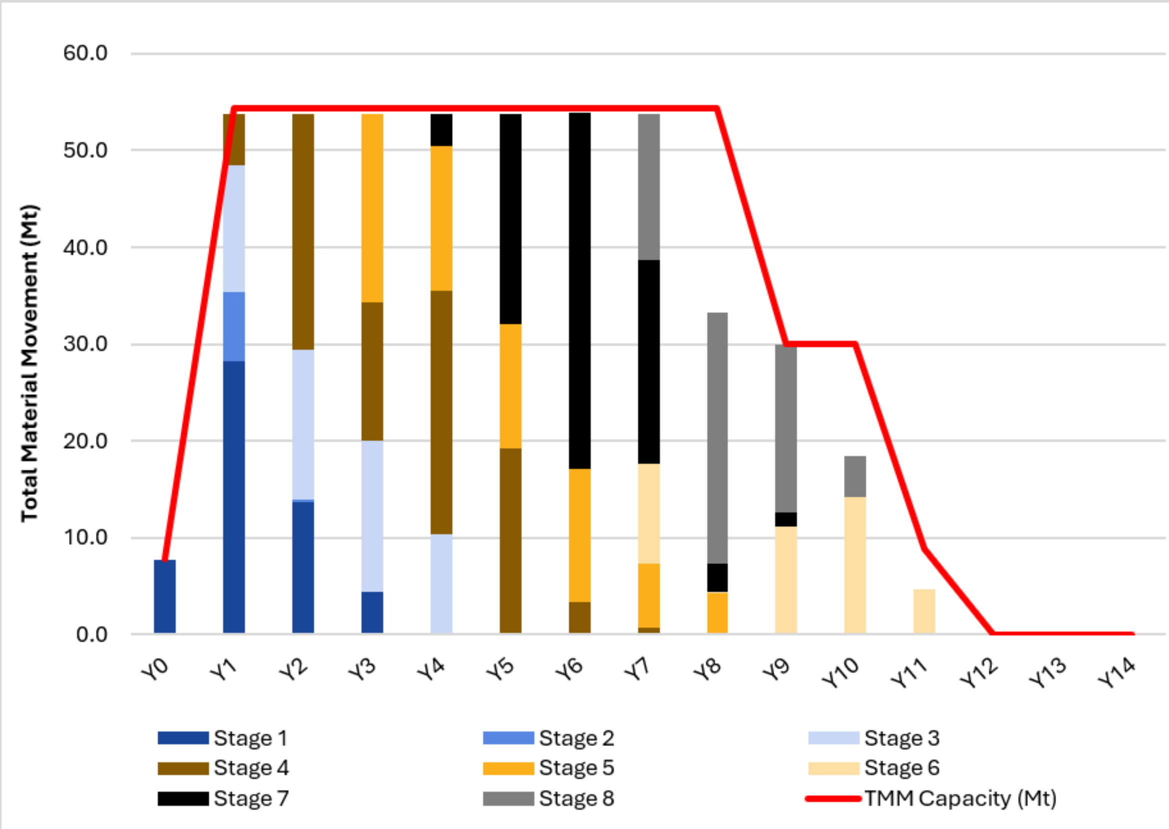


Figure 8.22: Annual Mining Movement by Pit Stage

Table 8.11: Stage Timing and Vertical Advance Summary

Stage	Start Period	End Period	Duration	Total Material (Mt)	Max Vertical Advance (m/a)	Year of Max Vertical Advance
1	Y-1M-3Q0	Y3M31Q11	2.7	54	55	Y1
2	Y1M1Q1	Y2M14Q5	1.1	7.3	75	Y1
3	Y1M4Q2	Y5Q20	4.5	54.7	80	Y4
4	Y1M9Q3	Y7	5.3	92.2	65	Y5
5	Y3M27Q9	Y8	4.8	72	80	Y8
6	Y7	Y11	4	40.5	75	Y10
7	Y4Q16	Y9	4.3	87.4	90	Y6
8	Y7	Y10	3	62.8	80	Y9

The processing and production schedule is summarised in Table 8.12. The plant feed is maintained at the nominal 5.25 Mtpa rate from Year 2 through Year 13, with a reduced final feed in Year 14 as the remaining stockpile inventory is depleted. The schedule delivers average production of 161 koz/a of gold over the first five years of production, 150.0 koz/a over the first ten years and 131 koz/a over the LOM for total production of 1.84 Moz. Annual recovered gold production is shown in Figure 8.25.

Figure 8.23 shows the annual mill feed split between direct feed and stockpile reclaim, with the corresponding mill feed grade. The schedule initially relies on direct pit feed, however after Year 9 feed from reclaim progressively increases until end of mine life.

The mill feed by resource category is shown in Figure 8.24. The plant feed is dominated by Ore (Indicated Mineral Resources), with Inferred Mining Inventory introduced from Year 6 onward. Mineralised waste is not processed in this base case schedule, however it will be separately stockpiled and tracked remaining available for later processing.

The stockpile balance by material is shown in Figure 8.26. Medium- and high-grade ore stockpiles are drawn down during the later years of processing to maintain mill feed and maximise mill feed grade. Inferred Mineral Resources are also tracked separately and progressively reclaimed from Year 6 onward.

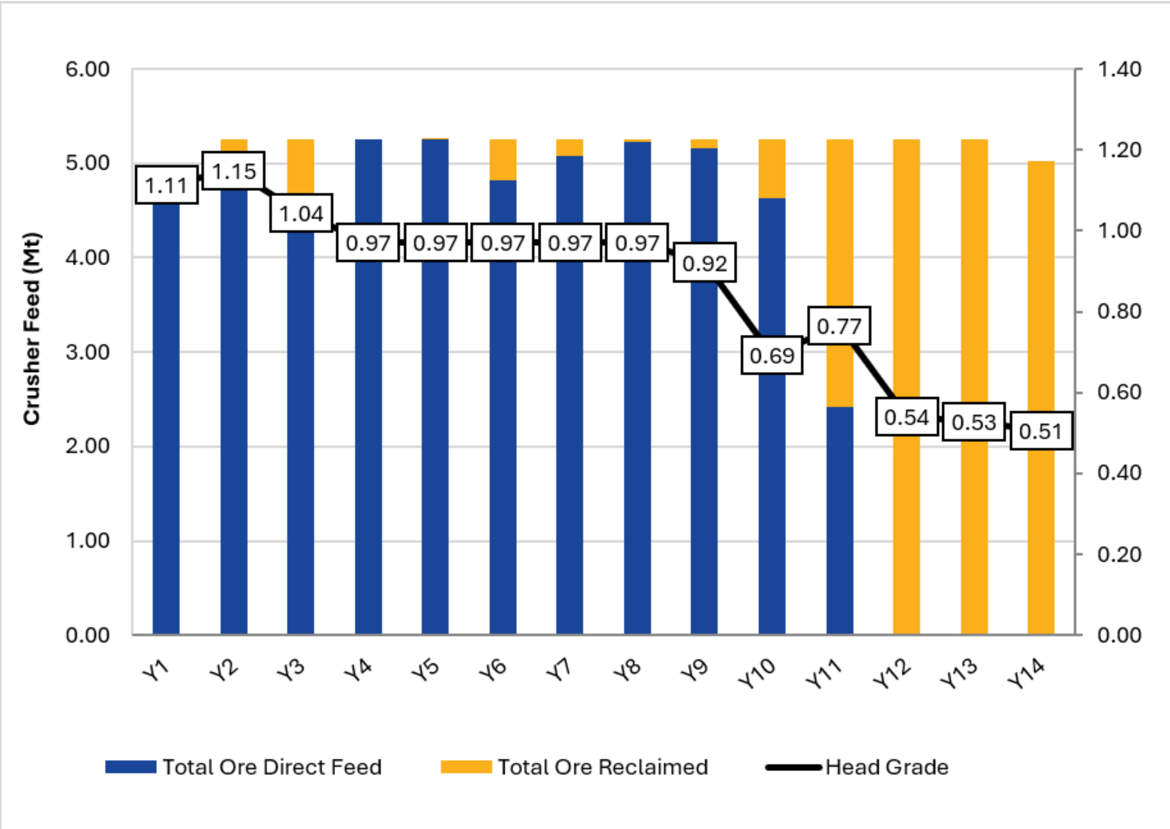


Figure 8.23: Annual Mill Feed and Grade

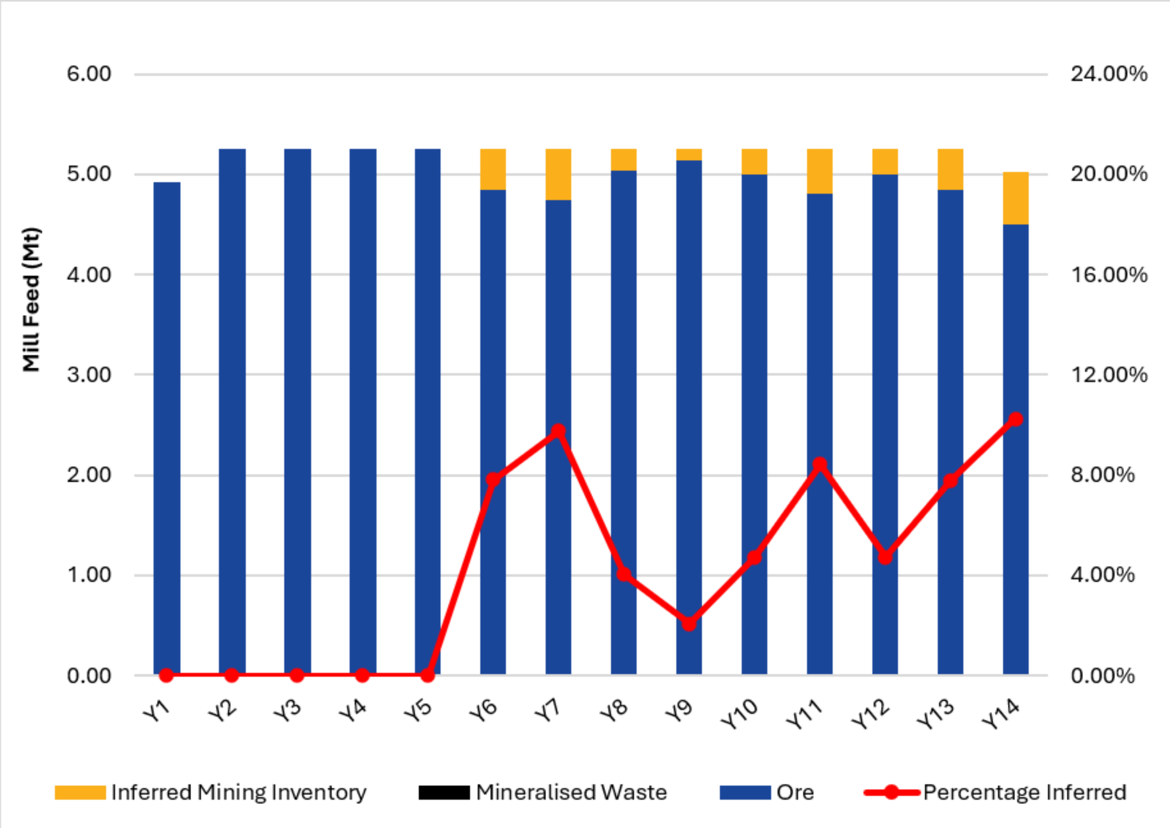


Figure 8.24: Annual Mill Feed by Resource Classification

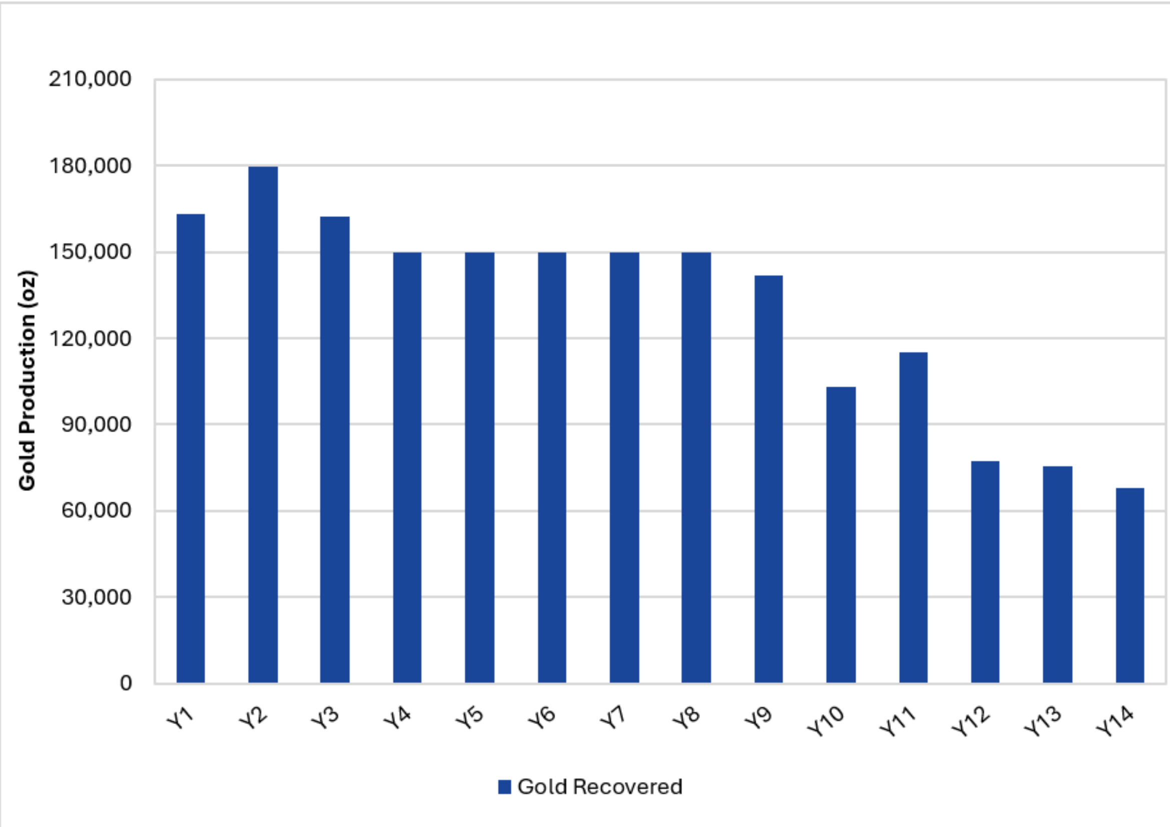


Figure 8.25: Annual Gold Production

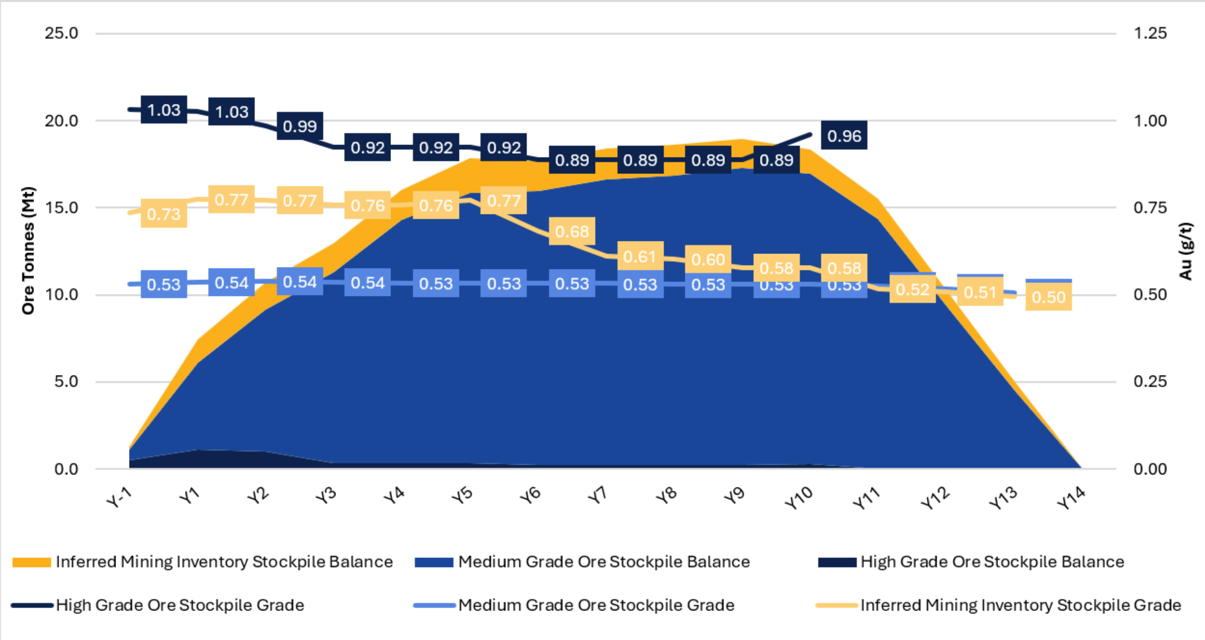


Figure 8.26: Annual Stockpile Balances

Table 8.12: Annual Processing and Production Schedule

	Units	Total	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Mill Feed																	
Direct Feed	Mt	52.0		4.65	4.95	4.57	5.25	5.25	4.82	5.08	5.25	5.15	4.63	2.42	0.01	0.00	0.01
Stockpile Reclaim	Mt	20.9		0.27	0.30	0.68	-	-	0.43	0.17	-	0.10	0.62	2.83	5.24	5.25	5.01
Total Mill Feed	**Mt**	72.9		4.92	5.25	5.25	5.25	5.25	5.25	5.25	5.25	5.25	5.25	5.25	5.25	5.25	5.02
Mill Feed Au Grade	g/t	0.87		1.11	1.15	1.04	0.97	0.97	0.97	0.97	0.97	0.92	0.69	0.77	0.54	0.53	0.51
Mill Feed Au Contained	**koz**	2,031		176	194	176	164	164	164	164	164	156	117	129	91	89	82
Ore	Mt	69.8		4.9	5.3	5.3	5.3	5.3	4.8	4.7	5.0	5.1	5.0	4.8	5.0	4.8	4.5
Inferred Mining Inventory	Mt	3.1		-	-	-	-	-	0.4	0.5	0.2	0.1	0.2	0.4	0.2	0.4	0.5
Inferred Proportion Tonnes	%	4.3		-	-	-	-	-	7.8	9.8	4.0	2.1	4.7	8.4	4.7	7.8	10.2
Inferred Proportion Au	%	3.8		-	-	-	-	-	9.1	10.0	3.5	2.3	3.9	9.8	4.7	7.8	10.0
Production																	
Recovery	%	90.4		92.5	92.8	92.1	91.5	91.5	91.5	91.5	91.5	91.0	88.0	89.2	84.7	84.3	83.6
Recovered Au	**koz**	**1,836**		163	180	162	150	150	150	150	150	142	103	115	77	75	68

8.7 Ore Reserve

The open pit Ore Reserve was derived from the DFS pit inventory and final open pit design, which was based on a pit optimisation undertaken using Indicated Mineral Resources only and constrained by a pit shell generated at a gold price of US$2,600/oz. The estimate was prepared for reporting in accordance with the JORC Code (2012) and includes only material classified as Indicated Mineral Resource that is located within the final open pit design and above the reserve reporting cut-off grade.

No Measured Mineral Resource is present in the current pit inventory and therefore no Proved Ore Reserve is reported. The Ore Reserve is therefore classified as Probable Ore Reserve, reflecting conversion of Indicated Mineral Resource after application of the relevant mining, metallurgical, geotechnical, economic, environmental and operational modifying factors.

The reserve reporting cut-off grade is 0.4 g/t Au was applied. Indicated Mineral Resources contained in the final open pit below 0.4 g/t Au but above the breakeven cut-off grade is classified separately as low-grade mineralised waste and is not included in the Ore Reserve estimate. Similarly, Inferred Mineral Resources contained in the final open pit above 0.4 g/t Au are reported separately in the mining inventory but is not converted to Ore Reserves. This approach is consistent with the requirements of JORC Code (2012), which does not permit Inferred Mineral Resources to Ore Reserves.

A LOM mining and production schedule was developed excluding all Inferred Mining Inventory. The schedule was subsequently optimised to maintain full process plant utilisation while relying solely on Ore Reserve material. This schedule was evaluated using the Project financial model to confirm that the open pit Ore Reserves remain economically viable without any contribution from Inferred Mineral Resources. The resulting financial outcomes demonstrated robust economics at the gold price assumption adopted for the Ore Reserve estimate, confirming that the declared Ore Reserve is both valid and supportable.

The resulting Ore Reserve estimate is summarised by JORC classification in Table 8.13. The estimate is reported on a dry tonne basis and reports contained gold prior to metallurgical recovery. Metallurgical recovery was applied in the production schedule and financial model, but recovered ounces are not used for classification of the Ore Reserve statement.

Table 8.13: Open Pit Ore Reserve Estimate

JORC Classification	Tonnes (Mt)	Au Grade (g/t)	Au Contained (koz)
Proved	-	-	-
Probable	69.8	0.87	1,953
Total	**69.8**	**0.87**	**1,953**

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$2,600/oz.
3. Ore Reserves are reported above a cut-off grade of 0.4 g/t Au.
4. Ore block grade and tonnage dilution was included into the model.
5. Only Indicated Mineral Resources were converted to Ore Reserves after consideration of the relevant Modifying Factors; no Measured Mineral Resources are present and therefore no Proved Ore Reserves are reported.
6. Ore Reserves are a subset of the Mineral Resources reported in Section 4.8 and are not additional to those Resources.
7. Inferred Mineral Resources are not converted to Ore Reserves and are excluded from the Ore Reserve estimate.
8. Low-grade mineralised material below 0.4 g/t Au is excluded from the Ore Reserve estimate and is treated as mineralised waste for reserve reporting.
9. Contained gold is calculated from ore tonnes and Au grade and is reported before metallurgical recovery.
10. Tonnages, grades and contained metal are rounded to reflect the level of accuracy of the estimate. Totals may not add due to rounding.
11. The Competent Person for the Reserve is Mr Jake Fitzsimons FAusIMM, Principal Consultant with Orelogy Consulting Pty Ltd. Mr Chris Jones, Principal Consultant with Orelogy Consulting Pty Ltd visited site in May 2026.

9 PROCESSING

9.1 Design

9.1.1 Trade-Off Studies

At the commencement of the DFS, various trade-off studies were undertaken on the process plant design, including:

- Leach residence time.
- Carbon-in leach (CIL) versus carbon-in-pulp (CIP), with CIP being selected due to reduced gold lock-up in inventory.
- Milling circuit design grind size.

9.1.1.1 Leach Residence Time

Metallurgical testwork identified both grind size and residence time as key drivers of gold extraction performance. Of the two variables, the effect of residence is more significant with an increase from 24-hours residence time (as per the scoping study) to 48-hours residence time resulting in an increase in extraction of approximately 3% based on feed grade of 1 g/t Au. The improved recovery translates to an increase in annual gold production of approximately 5 koz, equivalent to additional annual revenue of approximately US$18 million at a gold price of US$3,550/oz.

The additional cost of the leach tanks required to achieve the 48-residence time, based on high-level cost estimates, was determined to be approximately US$15m. Given the anticipated increase in gold production and a payback period of less than one year, the DFS design basis was updated to incorporate a 48-hour leach residence time.

9.1.1.2 Carbon-in-Leach versus Carbon-in-Pulp

With the increase in leach residence time to 48-hours, a trade off between CIL and CIP was revisited due to the significant carbon inventory required in a CIL circuit. The evaluation showed:

- Preferred CIL circuit configuration was 14 CIL tanks of approximately 3,100 m^3 with a carbon inventory of 450 t.
- Preferred CIP circuit configuration of 12 leach tanks of approximately 3,100 m^3 followed by six CIP tanks of approximately 900 m^3 with a carbon inventory of 135 t.
- Capital cost difference between the CIL and CIP option of less than US$1m.
- Operating cost increase for CIP of approximately US$120,000 per annum.
- CIL has an in-process gold inventory of approximately 8,150 oz, whereas CIP reduces this to approximately 2,280 oz.

With the capital and operating costs being similar between the CIL and CIP options, the determining factor was the increase in in-process gold lock up of 5,870 oz which translates to a deferral of approximately US$21m until the end of the LOM.

Based on this analysis, CIP was selected as the preferred option for the DFS.

9.1.1.3 Milling Circuit Design Grind Size

From the metallurgical testwork, the impact of finer grinding, at the 48-hour residence time, is greatly reduced. Using the analysis outlined in Section 7.4, a finer grind of 63 µm reduces the Metallurgical testwork demonstrated that, at a 48-hour leach residence time, the incremental benefit of finer grinding is significantly reduced. Based on the analysis presented in Section 7.4, reducing the grind size from 75 µm to 63 µm reduces the residual gold grade by only 0.005 g/t Au. This decrease in residual gold grade from leaching represents additional revenue of approximately US$3m/a at US$3,550/oz.

The comminution circuit design was based on the 85th percentile hardest ore, to achieve a design grind size of 75 µm and comprising the following circuit configuration:

- Primary gyratory crusher.
- SAG and ball mills both with 12 MW installed power.
- Pebble crusher treating SAG mill pebbles.

While the circuit was designed on the basis of the 85th percentile hardest ore, normal operations are expected to process ore with a significantly lower average hardness. Comminution modelling undertaken using average ore hardness parameters demonstrated that the selected milling circuit is capable of consistently achieving a grind size of approximately 63 µm.

Accordingly, although the milling circuit was designed to achieve a grind size of 75 µm under the most demanding operating conditions, typical operating performance is expected to achieve a finer grind. The resulting improvement in gold recovery and reduction in residual gold grade have been incorporated into the life-of-mine metallurgical performance. projections

9.2 Design Criteria

The processing plant will be designed to process 5.25 Mtpa of fresh open pit ore, operating seven days per week at a nominal treatment rate of 656 dry tonnes per hour on fresh ore with a grinding circuit utilisation rate of 91.3%.

The processing facility unit processes are based on proven processing technology for gold recovery and based on the following high-level design criteria outlined in Table 9.1.

Table 9.1: Key Process Design Criteria and Equipment Sizing

Parameter	Units	Value
Operating Schedule		
Primary Crushing Throughput (dry	t/h	800
Grinding and Plant Throughput (dry	t/h	656
Design Au Feed Grade	g/t	1.20
Design Gold Recovery	%	91.02
Design Ag Feed Grade	g/t	0.70
Design Ag Recovery	%	29.56
Maximum Production (equivalent rate)	koz/a	184
Maximum Production (equivalent rate)	koz/a	35
Physical Ore Characteristics (Design)		
SMC (Axb)		26
Bond Rod Mill Work Index	kWh/t	14.3
Bond Ball Mill Work Index	kWh/t	19.2
Bond Abrasion Index	g	0.15
Specific grinding power (P_{80} 75 µm)	kWh/t	29.8
Crushing		
Primary crusher		Gyratory Crusher - Superior MK-III 42-65 or Equivalent
Feed size, F100	mm	900
Product size, P100	mm	320
Crushed ore stockpile, live capacity	t	15,740
Grinding Circuit		
Circuit type		SABC
Feed size, F_{100}	mm	320
Product size, P_{80}	µm	75
SAG mill		10.4 m diameter x 5.24 m EGL, grate discharge, 12 MW installed, low speed VSD
Ball mill		7.6 m diameter x 10.49 m EGL, overflow discharge, 12 MW installed, low speed VSD
Pebble crusher		Cone crusher, Metso HP450 or equivalent
Gravity Circuit		
Gravity gold recovery	%	45.5
# of concentrators	#	2
Max Concentrator capacity	t/h/Unit	400
Intensive leach capacity	kg/day	5 260

Parameter	Units	Value
Pre-leach, Leach and Adsorption		
Pre-leach thickener settling rate	$t/h/m^2$	1.043
Pre-leach thickener		High rate, 35 m diameter
Pre-oxidation slurry residence time	h	4
Leach slurry residence time	h	48
CIP slurry residence time	h	6
Pre-oxidation tanks	#	1
Leach tanks	#	11
Nominal volume	m^3	3,300 – 3,600
CIP tanks	#	6
Nominal volume	m^3	900
Carbon loading (With or Without Gravity)	Au g/t	1,223 – 1,601
Carbon loading (With or Without Gravity)	Ag g/t	257 – 322
Elution, Electrowinning and Carbon Regeneration		
Elution process		Pressure Zadra
Acid wash and elution column	t	12
Elution frequency	per week	6
Carbon regeneration kiln		Diesel fired, 600 kg/h
Cyanide Detoxification		
Process		INCO SO_2 Cyanide Destruction
Detoxification discharge WAD	mg/L	20
Detoxification slurry residence time	h	1
Number of tanks		2
Tank volume, nominal	m^3	590

Parameter	Units	Value
Tailings Filtration		
Type		Plate and Frame Pressure Vertical plate Filters
Circuit Utilization	%	82.7
Tailings filtration throughput (dry solids)	t/h	725
Tailings filtration filter cycle time	minutes	12.8
Cake moisture content	%	15
Bulk solids density (max)	t/m^3	1.41 (1.86[1])
Filtration rate (max)	$t/m^2/h$	0.15 (0.20)
Filter chamber size	L x H x W	2.50 m x 2.50 m x 0.05 m
Number of filters/plates per filter		4 / 130
Design Filtration Area	m^2	6,500
Filtration Solids per cycle (max)	t	53.3 (70.3[1])
Estimated Excess Capacity (max)	%	37 (80[1])

Notes:

1. Vendor recommended bulk density is based on higher operating pressures possible in industrial equipment compared to laboratory scale equipment. Increased filter production per cycle and excess capacity is based on this recommended bulk density.

9.3 Flowsheet

The Kokoseb gold processing plant flowsheet incorporates primary crushing of run-of-mine ore to reduce large rock sizes (up to approximately 900 mm) to a manageable product of approximately a P_{80} of 150 mm for downstream handling. The crushed ore is then processed in a grinding circuit, which comprises of a semi-autogenous (SAG) mill followed by a ball mill and pebble crushing (SABC configuration), achieving a fine grind of around P_{80} of 75 µm to liberate gold-bearing minerals. Due to the design margins inherent in the design, however, it is anticipated that the grinding circuit will normally achieve the finer grind of 63 µm.

A portion of the cyclone feed from the grinding circuit is treated using centrifugal gravity concentrators to recover coarse free gold. The gravity concentrate is subjected to intensive cyanide leaching, and the dissolved gold is recovered via electrowinning. The remaining fine slurry (cyclone overflow) is thickened to 50% solids before entering a conventional leach and carbon-in-pulp (CIP) circuit. The leach section includes a pre-oxidation stage followed by multiple agitated tanks providing a residence time of 48 hours for gold dissolution. The CIP circuit then adsorbs dissolved gold onto activated carbon over six stages.

Loaded carbon is treated through acid washing and elution (using the pressure Zadra process) to strip gold into solution, followed by electrowinning to produce a gold-bearing precipitate. This precipitate is smelted to produce doré bullion. Stripped carbon is thermally regenerated and recycled back into the CIP process.

Tailings from the CIP circuit undergo cyanide destruction to reduce residual cyanide to environmentally acceptable levels, after which the slurry is thickened and filtered prior to trucking and deposition in the TSF.

Supporting systems include reagent preparation and dosing, comprehensive water management, oxygen plant and blowers.

Raw water is stored in the raw water pond, filtered, and distributed across the plant, with reverse osmosis treatment supplying high-quality process water where required. Potable and fire water systems are also integrated to support plant operations and safety. A schematic of the process flowsheet is provided in Figure 9.1.

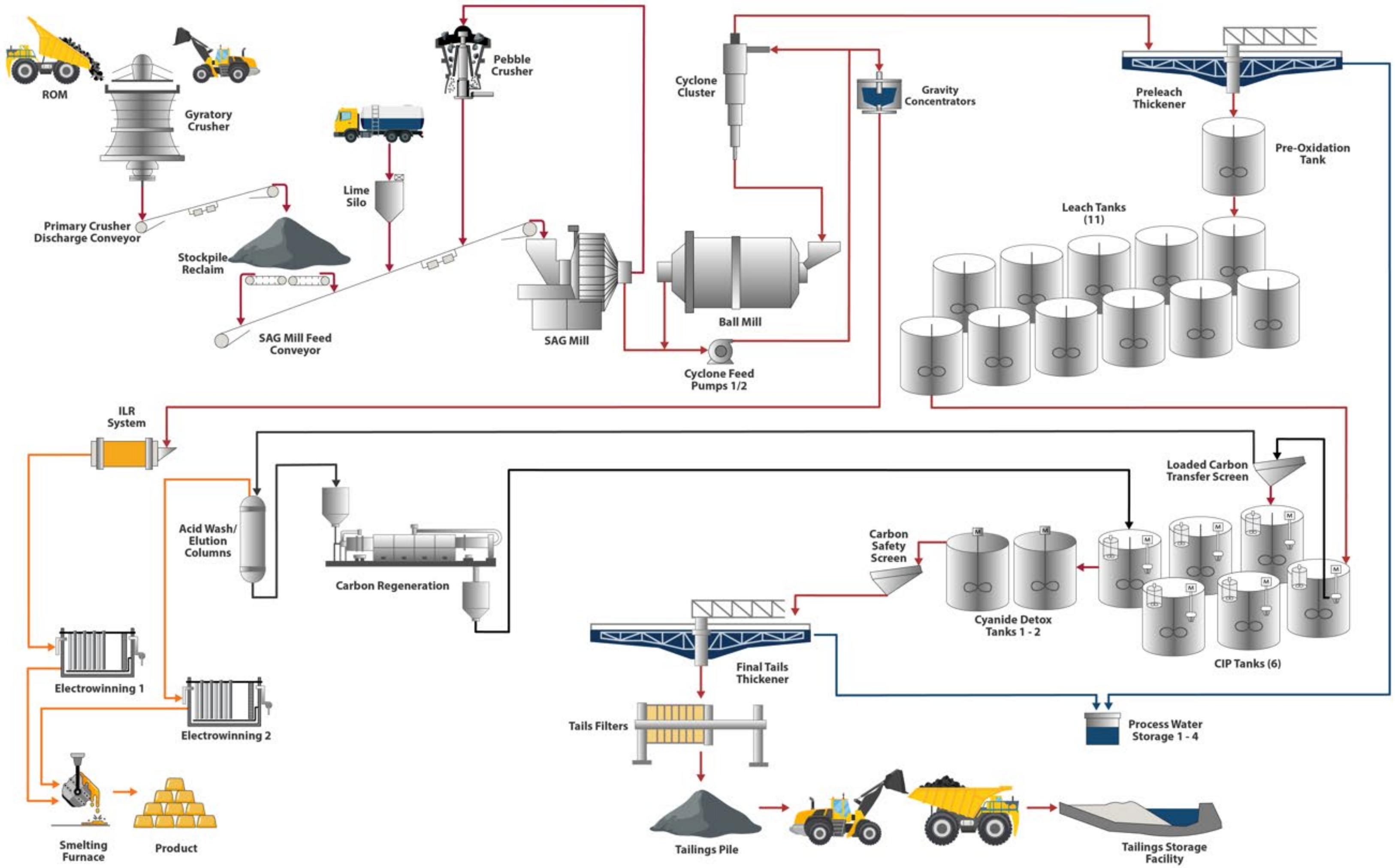


Figure 9.1: Process Flowsheet

9.4 Process Plant Layout

The layout of the gold processing facility was developed with a logical and efficient spatial arrangement to support safe operations, effective material flow, and ease of access to key infrastructure.

The processing plant is centrally located within the site, forming the core operational hub. This central positioning minimises haulage distances for run-of-mine (ROM) ore delivery and provides optimal connectivity to supporting infrastructure such as the TSF, water supply systems, and reagent storage areas. The layout facilitates a streamlined process flow from crushing through to gold extraction and tailings management, while also allowing for future expansion where required.

To the south-west of the processing plant, the mining workshops and maintenance facilities are situated. This location enables direct access to the mining areas and provides sufficient space for heavy equipment servicing, maintenance bays, parts storage and ancillary support functions. The separation of workshops from the processing plant enhances safety by reducing traffic congestion and isolating heavy vehicle activity from core plant operations.

The administration offices and support buildings are positioned to the west of the processing plant. This placement ensures a clear segregation between operational and non-operational personnel while maintaining convenient access to the plant. The administrative area typically accommodates offices, control rooms, laboratories and parking facilities, and is designed to provide a safe and comfortable working environment removed from high-noise and high-traffic plant areas.

Overall, the site layout reflects a considered balance between operational efficiency, safety, and future scalability, with clear zoning of processing, maintenance, and administrative functions. A schematic of the process plant layout is provided in Figure 9.1 and layout drawing Figure 9.2.

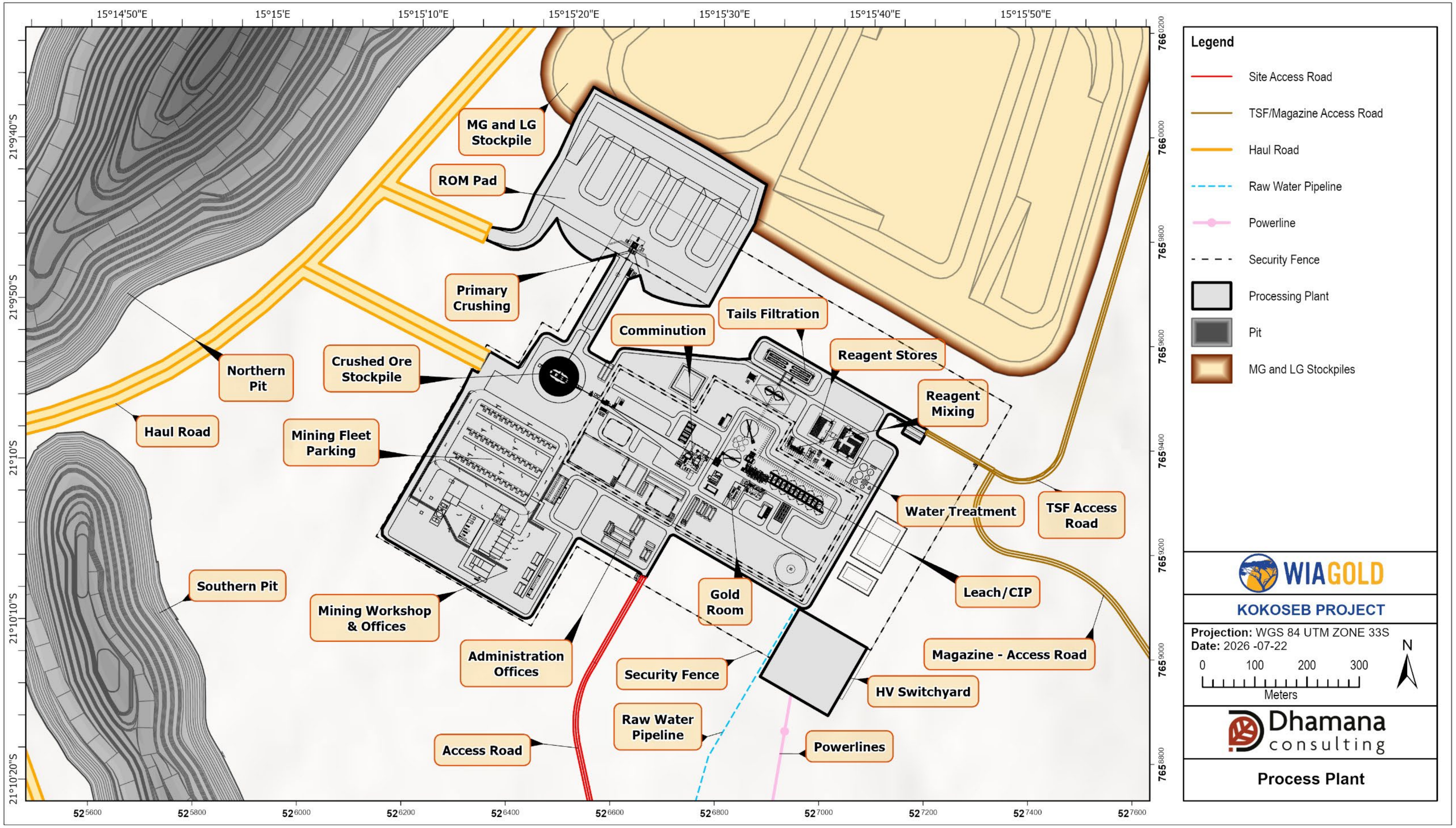


Figure 9.2: Process Plant Layout

10 TAILINGS DISPOSAL

10.1 Tailings Storage Facility Selection and Operation

A filtered stack TSF was adopted for the Project based on evaluation through a comprehensive multi-criteria alternative analysis (MCAA) conducted during the scoping study in 2025. The MCAA process also assessed several TSF locations, shown in Figure 10.1 and configurations with consideration of risks, environmental, social and business-related criteria in accordance with the GISTM Principle 3 requirements. TSF Option 5 was identified as the preferred location through this multi-criteria approach and was subsequently selected for advancement in the DFS.

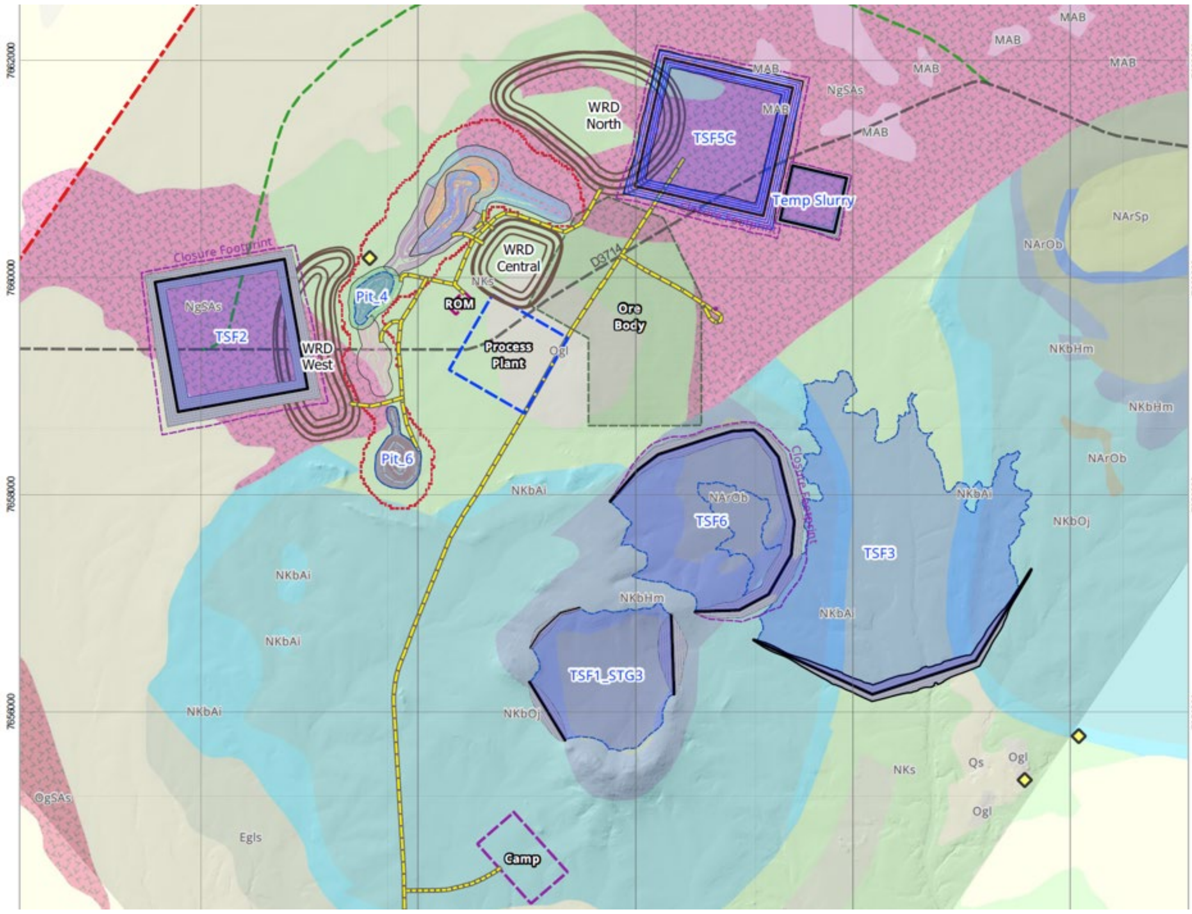


Figure 10.1: Initial Locations Considered for the TSF in the MCAA Process

As part of the DFS, the conceptual arrangement was revisited through a second MCAA process to compare different facility configurations, embankment construction methods, and tailings placement methodologies.

Preliminary geochemical assessment indicates the tailings are non- acid forming but may exhibit metal leaching characteristics, primarily arsenic, although this testing is currently inconclusive and detailed kinetic testing is ongoing.

A range of embankment raising strategies and geometric configurations were evaluated during the MCAA process resulting in a phased TSF configuration aligned to the natural topography and integrated with the WRD to facilitate stacking, support and closure. The design concept is show in Figure 10.2.

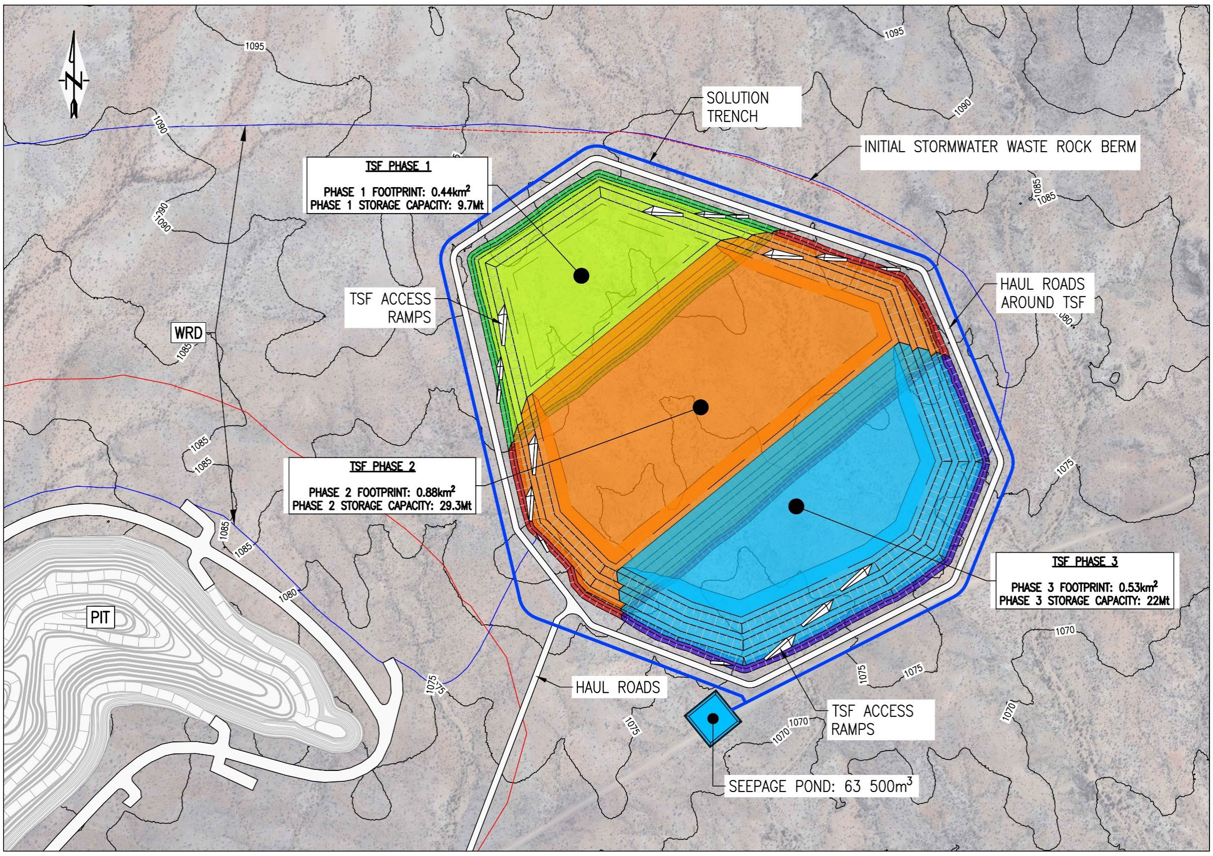


Figure 10.2: TSF Design Concept

10.2 Design and Construction

The TSF design basis and operating criteria are based on accepted national and international standards for TSF design and operation, including the Australian National Committee on Large Dams (ANCOLD) (2012 and 2019); GISTM, (2020) and applicable local guidelines such as the Best Practice Guide for Mining in Namibia (2019) and legislation in place such as the Namibian Environmental Management Act, 2007, the Water Act 1956 and Water Resources Management Act 2013.

The dam consequence classification assessment results in a facility that is classified as "Significant" due to the population at risk (PAR) and the potential metal leaching of the tailings, though the impacted area from a slope failure is relatively small. The earthquake design ground motion (EDGM) and inflow design flood thresholds used for the design reflect this consequence classification:

- **Inflow design flood**: Temporarily contain runoff from the 1 in 1 000-year ARI, 24-hour storm event.
- **Design earthquake**: The EDGM ground acceleration from the 1 in 1,000-year earthquake (5% probability of exceedance in 50 years).

The facility will be constructed and operated in three distinct phases, with each phase designed to be advanced to final height before commencement of the subsequent phase. Phase 1 will provide storage capacity of 9.1 Mt over approximately 1.75 years, reaching a final elevation of 1,108 mamsl. Phase 2 will extend southward and accommodate 27.3 Mt of tailings over approximately 5.25 years, also to an elevation of 1,108 mamsl. Phase 3 will extend further south and store the remainder of the LOM tailings. Completed phases will be progressively encapsulated with waste rock while subsequent phases are being developed, allowing for concurrent construction and closure activities.

As show in Figure 10.3, each TSF phase comprises a 5 m minimum height engineered starter embankment constructed from compacted waste rock and an interface layer for liner anchorage and tailings migration minimisation. The embankment provides containment, foundation stability, and structural support. An anchor trench will be excavated for secure liner anchorage, with an HDPE or equivalent geomembrane installed in the basin. Perimeter solution trench, stormwater diversion channels, seepage collection systems, and access ramps (potentially multiple routes per phase) complete the primary infrastructure. The progressive development of each phase includes liner extension, berm construction, and expansion of stormwater and seepage collection systems, with early collection implemented where feasible to support water recycling back to the process plant.

The design incorporates dedicated handling and drying areas within each phase, including contingency drying pads within structural zones for moisture management when required.

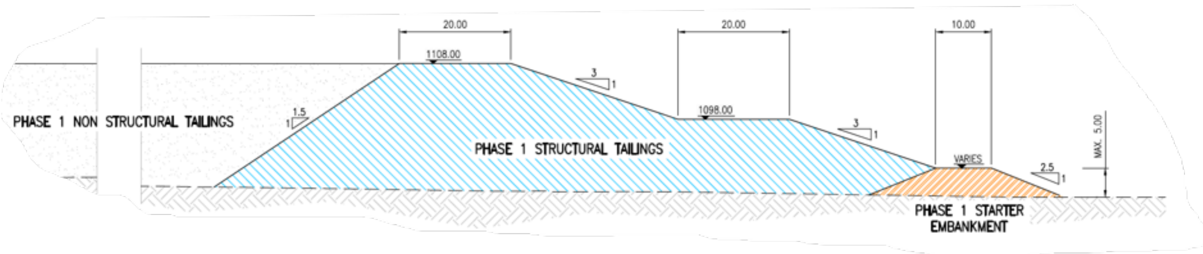


Figure 10.3: TSF Typical Cross Section

10.3 Operations

10.3.1 Tailings Placement Strategy

Operations will follow a systematic placement and compaction methodology. Filtered tailings will be transported from the process plant by truck and end-dumped at designated tipping points within the active deposition area. Dozers will spread material in controlled lifts, with lift thickness varying between structural zones, comprising thinner lifts of 0.3 m for enhanced compaction control, and non-structural zones where 0.5 m lifts are planned. In structural zones, graders will level surfaces and smooth-drum rollers will achieve specified compaction densities through a defined number of passes. Material placement will be controlled to target moisture content ranges determined during detailed design, with moisture testing conducted at specified frequencies. Once each 5 m lift is completed, the TSF will be raised again in a similar way all the way to final height. This operating sequence is outlined in Figure 10.4, Figure 10.5 and Figure 10.6.

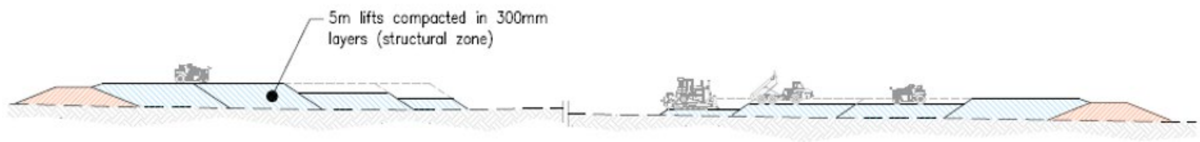


Figure 10.4: TSF Structural Zone First Lift

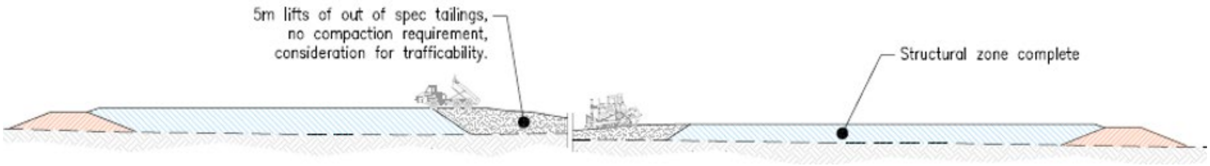


Figure 10.5: TSF Non-Structural Zone First Lift

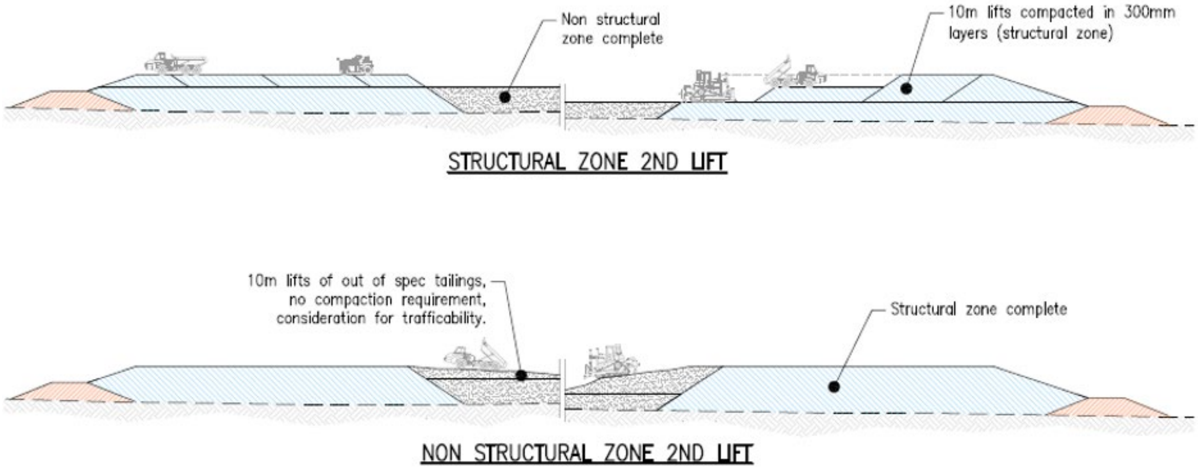


Figure 10.6: TSF Subsequent Lifts

A cell or checkerboard placement strategy may be employed whereby separate areas are placed and allowed to cure while adjacent cells are being developed, enabling return to previously placed cells after adequate drying periods. This approach maintains operational flexibility and production continuity while managing moisture-sensitive fine-grained tailings.

10.3.2 Tailings Management Fleet

Based on the proposed operating philosophy, the required annual tailings production rate, and allowing for moisture management, surface water management, dust suppression, haul road maintenance and routine operational support activities, the indicative operating fleet comprises:

- 7 off 40-t articulated dump trucks (ADT).
- 1 off CAT 980 (or equivalent) wheel loader.
- 1 off D6 (or equivalent) low ground pressure dozer.
- 1 off 14-t grader.

- 1 off 15-t smooth drum compactor.
- 1 off 16 kL water cart.

These equipment requirements were developed in conjunction with potential contractors for the tailings rehandle scope. The contractor will also provide suitable office and maintenance facilities for the management of the works and maintenance of the fleet.

Opportunities exist to reduce the cost of the haulage through the substitution of the articulated dump trucks for lower cost conventional on-road tipper trucks once the trafficability of the stacked tailings is better defined.

11 WATER SUPPLY

11.1 Operational Water Demand

An overall site water balance was developed using Project specific data and incorporates the principal water demand categories across the operation, including:

- Mining activities, primarily for dust suppression.
- Process plant operations, including losses to tailings and evaporation from the storage ponds and process vessels.
- Tailings rehandling and dust suppression requirements.
- Personnel usage associated with both the accommodation village and operational facilities.

Given the arid climatic conditions of the Project area and the limited availability of groundwater, including within the proposed open pit, the overall site water balance is forecast to be in deficit. Accordingly, a conservative basis was adopted for the design of the Project water supply system, whereby no contribution was assumed from the following potential supplementary water sources:

- Stormwater captured within the open pits, TSF or process plant areas.
- Groundwater abstracted from any pit dewatering bores or from sumps collecting seepage into the pit.

Where available, water recovered from these sources will be used to offset bulk water supply demand for the Project.

11.1.1 Mining

Mining water usage for dust management was estimated over the LOM based on the following parameters:

- Haul distances calculated from the detailed mining schedule.
- Average haulage times for in-pit and ex-pit travel (i.e. time actively on the roads/ramps) assuming:
  - Average in-pit speed of 20 km/h.
  - Average ex-pit speed of 45 km/h.
- Water cart usage of 12.5% of haul truck hours.
- Water cart loads of 45 kL and one load per hour.

This estimate was adjusted for rainfall based on the number of days where rainfall is anticipated to exceed 1 mm of rain in the day, which would result in reduced dust suppression requirements.

Based on this methodology the LOM estimated water usage in mining is presented in Figure 11.1.

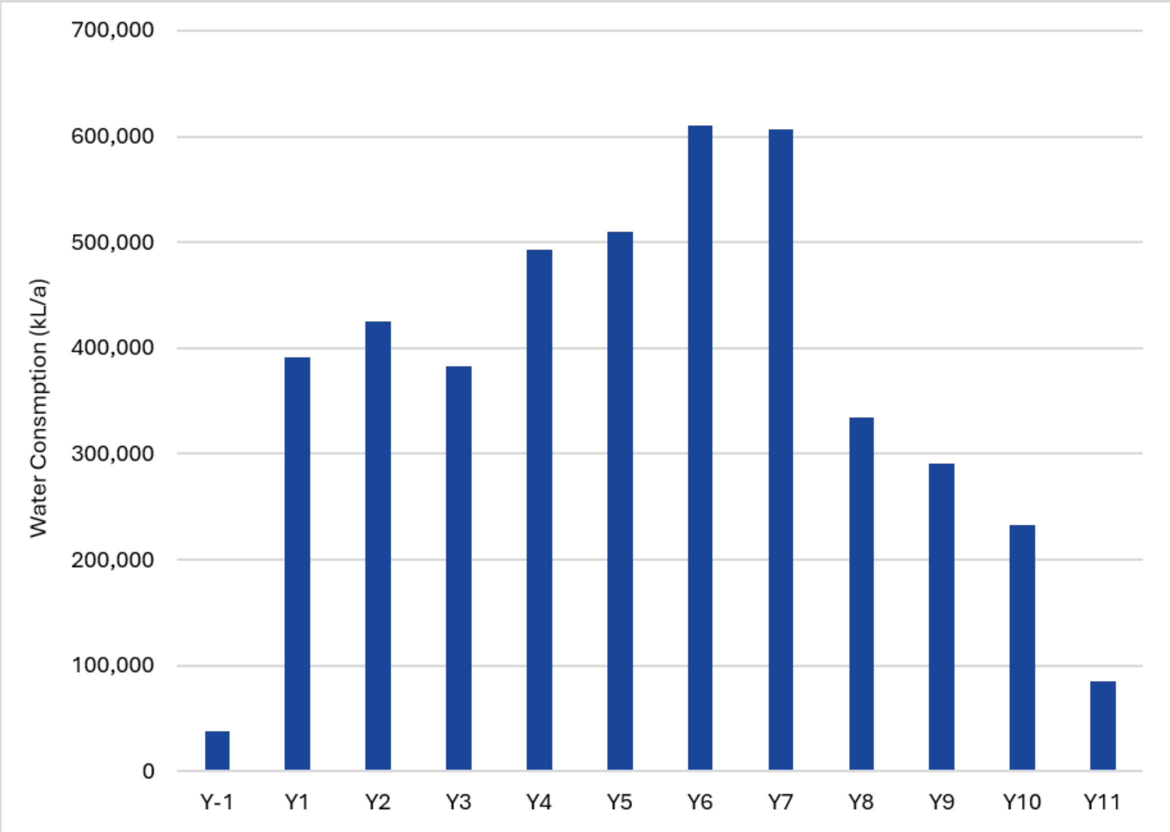


Figure 11.1: LOM Estimated Mining Water Usage

This analysis indicates a peak usage over the LOM of 1,755 kL/day which totals 640,000 kL/a. To validate this estimate, as part of the mining contract DFS tendering process contractors were requested to provide an estimate of water requirements with the responses for peak usage ranging from 890 kL/day to 1,500 kL/day.

There is also an opportunity to further reduce water demand through the use of a system such as Dust-A-Side®, however, this was not included in the determination of the design basis for the Project water supply.

11.1.2 Processing

The water demand for processing was calculated by determining the difference between the water in the ore being fed to the plant and the water contained in the filtered tailings. An additional check was carried out to ensure that specific raw water demands within the processing plant did not exceed this overall water demand, with filtered water being substituted where possible to bring the circuit into balance.

In addition, evaporation from the raw water dam and process tanks was calculated based on the area and the average annual evaporation rate.

Due to the use of filtered tailings and a filtered stack TSF, there is no excessive requirement for water on commencement of operations before water is returned from the TSF.

The total annual water demand for the process plant is approximately 653,500 kL/a.

11.1.3 Tailings Rehandle

Water usage for tailings rehandle is primarily used for dust management of the operating areas. This includes haul roads, active placement areas, exposed tailings and ancillary areas. It assumes that the tailings are produced at the design moisture content and therefore compaction does not require any additional water addition.

Based on the average operational areas, the water requirement was estimated at 123 m^3/day, which translates to 40,900 m^3/a.

11.1.4 Personnel

Personnel water demand is based on an allowance of 250 L/day/person, totalling 28,500 kL/a.

11.1.5 Evaporation and Storm Water

The project will harvest storm water to opportunistically supplement the project water demand and conversely water will evaporate from various open surfaces. The following permanent infrastructure is considered open plant that effectively captures all rainfall within its footprint, but also represents areas for evaporation loss:

- Raw water pond.
- Open topped tankage inclusive of thickeners, leach tanks, CIP tanks and the filter feed surge tanks.

In addition, drainage designed for the following areas will capture rainfall run off less efficiently than the open topped process infrastructure:

- Plant and NPI areas, which is estimated to have a run-off coefficient of 20%.
- Mining areas, which is estimated to have a run-off coefficient of 20%.
- Waste rock dumps and stockpiles, which is estimated to have a run-off coefficient of 5%.
- TSF, which is estimated to have a run off coefficient of 50%.

Accounting for average rainfall and evaporation the net positive supply of water from the project areas was estimated to average 80,000 m^3/a.

11.1.6 Overall Water Demand and Design Basis

The Project's peak water demand was estimated at 1.27 Mm^3/a. Although stormwater capture and return water recovered from various site facilities are expected to offset a portion of the freshwater requirement, their availability is inherently uncertain under the prevailing arid climatic conditions. A conservative design basis was therefore adopted which excludes these potential contributions and allows for a peak external water supply of 1.35 Mm^3/a.

11.2 Construction Water Demand

Water for construction is used across a number of areas, including:

- Bulk earthworks for moisture conditioning of bulk earthworks to allow compaction to the design requirements and dust management during construction.

- For the preparation of concrete.
- Personnel usage.
- Dust management and maintenance of roads and hardstands following construction.
- Dust management during pre-production mining.

A detailed estimate of the requirements based on geotechnical material classifications, material testwork, typical usage rates and a contingency factor was undertaken. This analysis showed the total requirement for water across the construction period is approximately 655,000 m^3, broken up as approximately 420,000 m^3 for bulk earthworks, 6,000 m^3 for concrete, 105,000 m^3 for personnel, 75,000 m^3 for dust management and 52,000 m^3 for pre-production mining. Additionally, the usage of this water was aligned with the construction schedule and the water usage histogram shown in Figure 11.2.

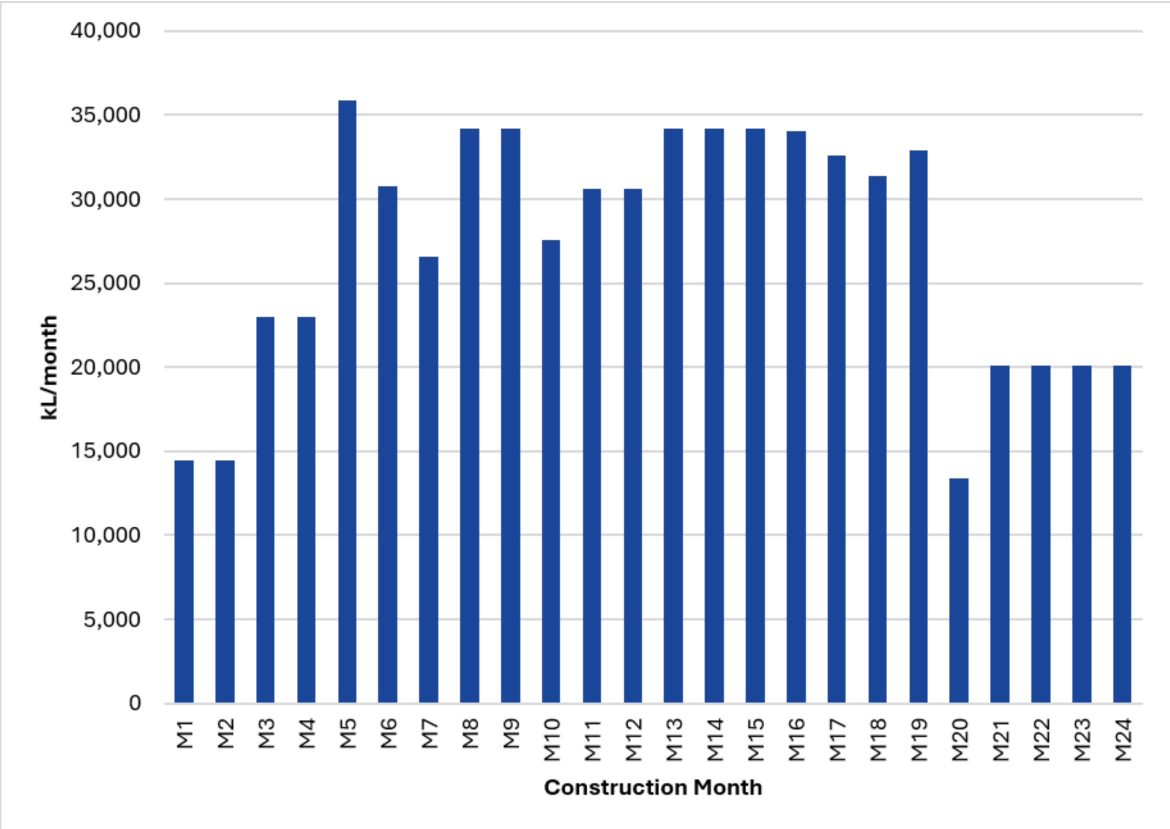


Figure 11.2: Construction Water Usage

11.3 Water Supply Options Investigated

Multiple water supply options for the Project were investigated as part of the DFS, including:

- Extension of the existing NamWater water supply schemes (WSS) at Okombahe, on the Omaruru River, Anixab, on the Ugab River, as well connection to Swakoppoort Dam via existing infrastructure with supply from Karibib.
- Groundwater abstraction from the Omaruru River delta's (OMDEL) northern and southern elevated channel alluvial aquifers.

- Groundwater abstraction from the palaeochannel beneath the Omaruru alluvial plain (OmAP).
- Groundwater supply from fractured Karibib marble aquifers on EPL 4818.
- Expansion of the Erongo Regional Council-owned Ozondati water supply scheme, targeting fractured aquifers in the surrounding area.

The locations of these various potential water sources are shown in Figure 11.3.

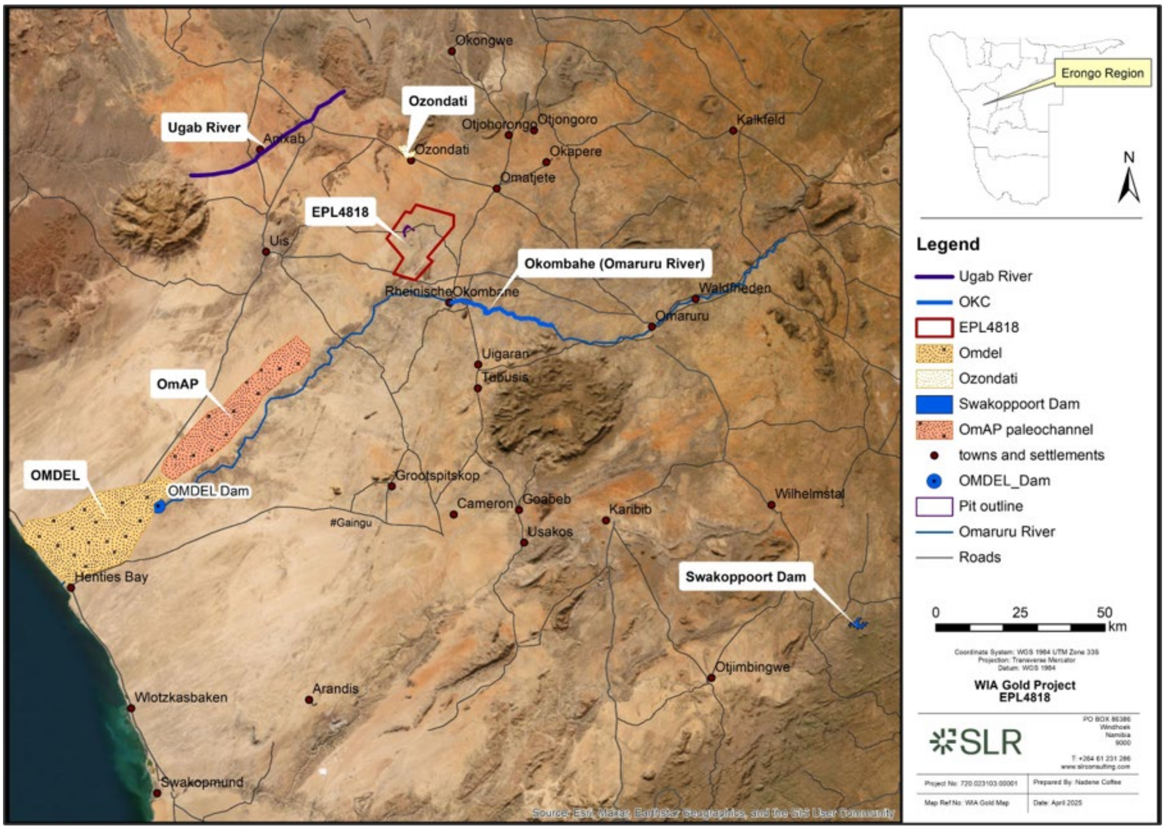


Figure 11.3: Water Supply Options Investigated

In addition to the options outlined above, investigations were also undertaken into the potential supply of seawater from Wlotzkasbaken and desalinated water from Orano's Erongo Desalination Plant (EDP), with supply of desalinated water from Trekkopje Mine, which is fed by existing infrastructure from the EDP.

A high level assessment of all identified water supply options was completed to evaluate their ability to meet the Project's construction and operational water requirements. Only those options considered to have a reasonable likelihood of providing a reliable and sustainable supply to the Project were advanced for detailed evaluation. The water supply options carried forward and assessed as part of the DFS, and discussed in the following sections, comprise:

- Groundwater abstraction from fractured Karibib marble aquifers on EPL 4818, primarily for construction water supply.
- Extension of the Okombahe WSS, to provide water for both construction and operational requirement.

- Groundwater abstraction from a dedicated OmAP borefield to meet operational water demand.
- Supply of desalinated water from Trekkopje Mine to satisfy operational water demand.

11.3.1 Karibib Marble Aquifers on EPL 4818

Groundwater occurs predominantly in secondary (fracture-controlled or karstic) aquifers hosted in crystalline and metamorphic rocks. Significant groundwater volumes mainly occur within fractured marble of the Karibib Formation, with some localised potential in zones where fracturing is more developed. The Kuiseb Formation schist, underlying much of the orebody area, is generally considered an aquitard but where densely fractured it can have some moderate groundwater potential.

A geophysical survey conducted identified eight sites targeting different geological units and structures. Average volumes of groundwater from marble aquifers on EPL 4818 were estimated by factoring rainfall, recharge and the geometry of the aquiferous formations. An estimated 215,775 m^3/a, or 591 m^3/day, was determined to be potentially available. Abstraction of this volume would require 10 to 15 production boreholes (see Figure 11.4), which could be pumped at up to 5 m^3/h and supply approximately 0.2 Mm^3/a of water at early stages of mine development and during construction. Drilling and test pumping of these groundwater targets is currently underway.

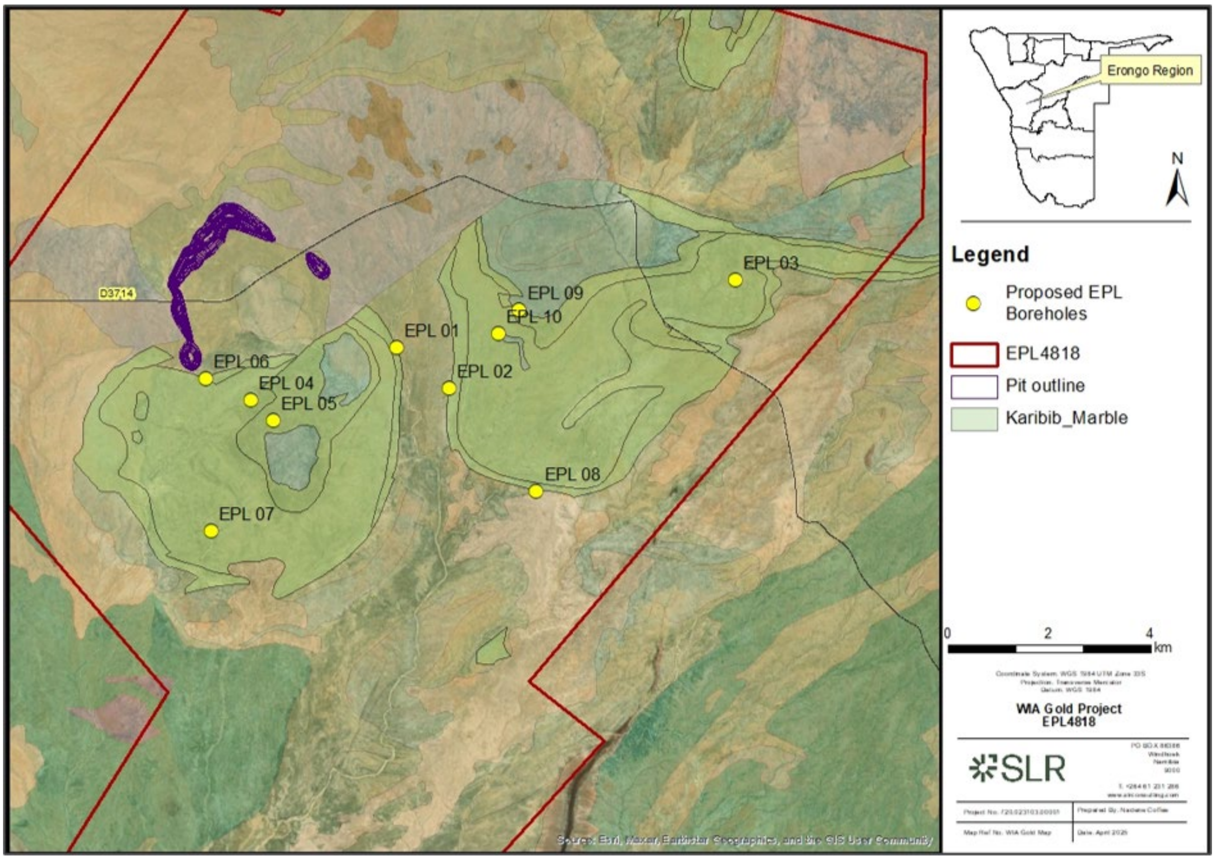


Figure 11.4: EPL 4818 Potential Bores

11.3.2 Okombahe Water Supply Scheme

The NamWater-operated Okombahe WSS is located approximately 8 km southeast from the southern boundary of the Project exploration licence. Groundwater is abstracted from alluvium of the Omaruru River and is the primary water source for domestic use, livestock and small-scale agriculture at the Okombahe Settlement.

The Omaruru River plays a critical role in the local hydrogeological and socio-economic setting. Although it is dry for much of the year, episodic flood flows contribute to groundwater recharge in the underlying alluvial aquifer systems. Overall, the alluvial aquifer is characterised by shallow groundwater levels, good hydraulic connectivity, and a dynamic recharge regime typical of ephemeral river systems, which have groundwater quality for potable use.

The Okombahe WSS, commissioned in 1986, comprises of three shallow production boreholes reaching depths of up to around 30 m before encountering bedrock, reflecting their dependence on the alluvial aquifer system. Production data indicates a total recommended abstraction capacity of the current WSS of 459,360 m^3/a, while actual abstraction in 2023 was only 161,393 m^3/a, equating to approximately 40% utilisation of scheme capacity. Long-term records (2004–2024) show an average abstraction of only 67,190 m^3/a, with potential individual production borehole yields of approximately 150,000 m^3/a.

The hydrogeological conceptual model for the Okombahe compartment is based on the regional hydrogeological model of the Omaruru River basin, developed by SLR in 2016, updated with recent geophysical survey, drilling, and test pumping data (Figure 11.5). The recharge-based sustainable groundwater abstraction calculation above shows that there is an estimated groundwater volume of 1.1 Mm^3/a potentially available from the alluvial aquifer in the Okombahe compartment.

Based on the drilling program results carried out in 2025 as part of the DFS, the Okombahe compartment, an approximately 35 km long section of the Omaruru River, comprises of two alluvial aquifer sub-compartments, separated by bedrock high.

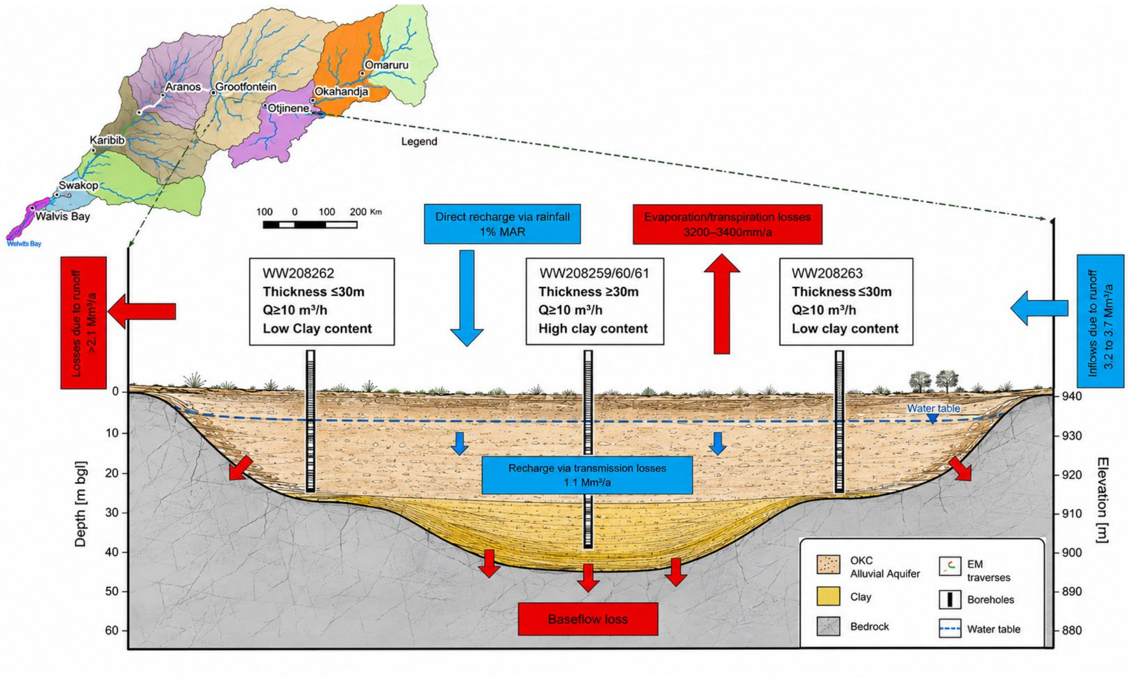


Figure 11.5: Conceptual Model of the Okombahe Compartment Alluvial Aquifer

A numerical groundwater model was developed, simulating abstraction scenarios, with a short- and long-term view of abstracting from a proposed wellfield distributed along more than 30 km of the compartment. Under steady-state conditions, the model estimated a total groundwater volume of approximately 35 Mm^3. Assuming a specific yield of 10%, an estimated extractable groundwater reserve of approximately 3.5 Mm^3, which was adopted as the reference storage volume available for groundwater development. This value represents the maximum extractable groundwater volume against which all model scenario results are to be assessed.

Abstraction within the model domain included the existing NamWater boreholes and 10 proposed production boreholes spread out over the 30 km compartment (see Figure 11.6), which could be pumped at between 5 and 10 m^3/h and supply potentially up to 0.5 Mm^3/a to the Project.

Simulation of groundwater abstraction considers indicators such as drawdown at production boreholes, maintenance of agreed minimum groundwater levels or river baseflow, capture fractions and post-drought recovery behaviour. The transient model outcomes demonstrate that the aquifer can sustain moderate abstraction rates under normal recharge conditions and remain operationally manageable under increased abstraction.

Pumping 0.67 Mm^3/a, which includes NamWater's current demand, is regarded as sustainable under average annual runoff conditions and even stays sustainable after drought periods not lasting longer than three years. However, at this rate of abstraction, significant drawdown of stored water volumes would occur should six years of extended low rainfall be experienced.

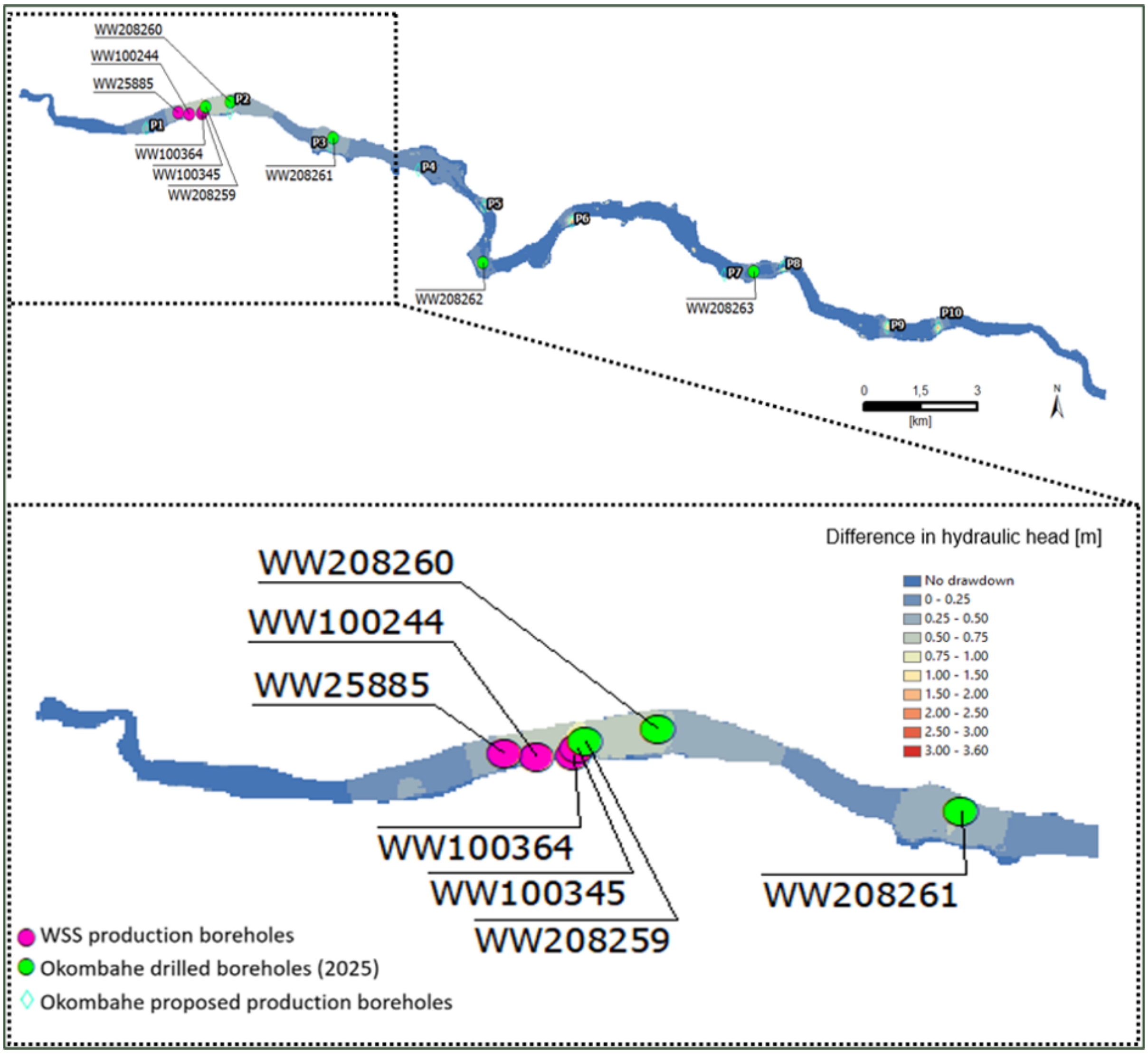


Figure 11.6: Okombahe Compartment Borefield and Impact of Abstraction

11.3.3 Omaruru Alluvial Plains

OmAP is located along the central western coast of Namibia, forming part of the extensive arid landscape of the Namib Desert. The plains extend over an area of approximately 2,600 km^2 and are characterised by an extremely flat and laterally extensive terrain. The environment is highly arid, supporting only sparse fauna and flora, with vegetation largely confined to major drainage lines where species are adapted to survive under limited water availability.

Geologically, the OmAP represent relict alluvial deposits associated with earlier stages in the evolution of the Omaruru River. These plains are inferred to overlie a palaeochannel formed by the historical southward migration of the river system. Over time, tectonic and geomorphological processes resulted in the abandonment of these channels, leaving behind an extensive alluvium-filled valley beneath the present-day plains.

Previous geophysical investigations within the OmAP have primarily aimed to identify groundwater-bearing palaeochannels associated with the historical evolution of the Omaruru River system. OmAP has been investigated since 1993 with helicopter-borne geophysical surveying comprising electromagnetic (EM), magnetic, and radiometric methods, followed by a limited drilling programme, which failed to intersect the proposed saturated palaeochannel aquifer. The results provided regional-scale delineation of potential palaeochannel systems, and also highlighted the inherent complexity of hydrogeological setting at OmAP, reinforcing the need for integrated geophysical and hydrogeological interpretation.

As part of the DFS, natural source audio magnetotelluric (NSAMT) surveying along the existing horizontal loop electromagnetic (HLEM) survey lines were undertaken to improve depth-to-basement resolution, provide accurate determination of the location of the deepest section of the presumed saturated palaeochannel and the subsequent siting of boreholes.

The first phase of NSAMT investigations was completed in early 2026 by Terratec Geophysical Services Namibia cc over five lines (lines 2 to 6 in Figure 11.7) coinciding with the earlier HLEM profiles. Two drilling sites were selected on line 4 and 6, targeting the deepest incision of the palaeochannel with successful results (see Figure 11.8and Figure 11.9).

These findings support a better understanding of the palaeochannel system and provide clearer guidance for drilling, with the deepest northeastern sector identified as the most prospective target for groundwater development.

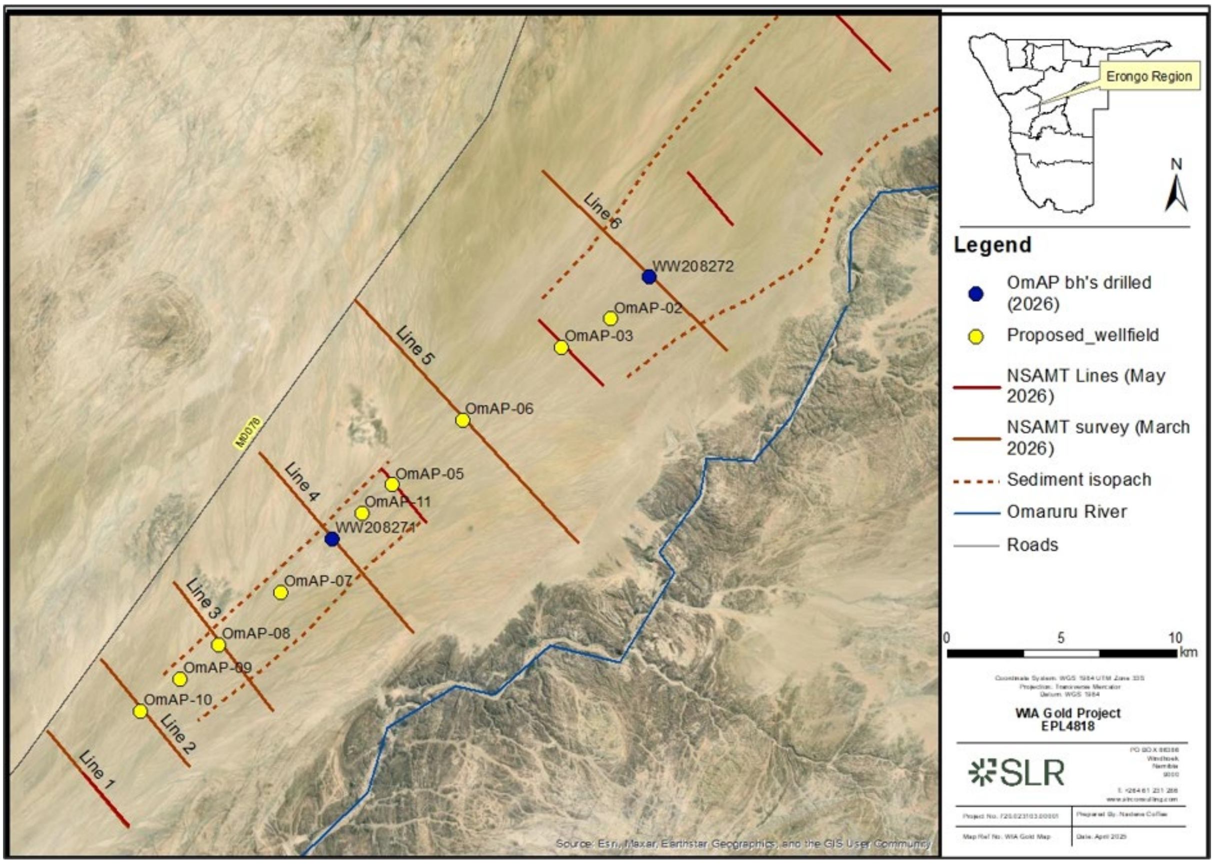


Figure 11.7: OmAP Site Map

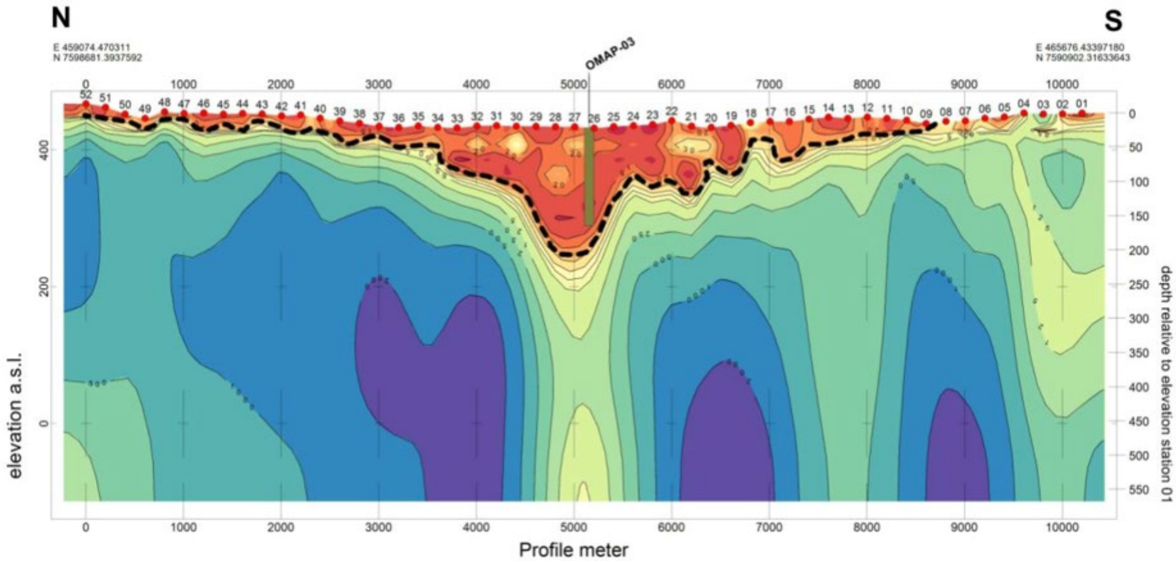


Figure 11.8: NSAMT Profile 4

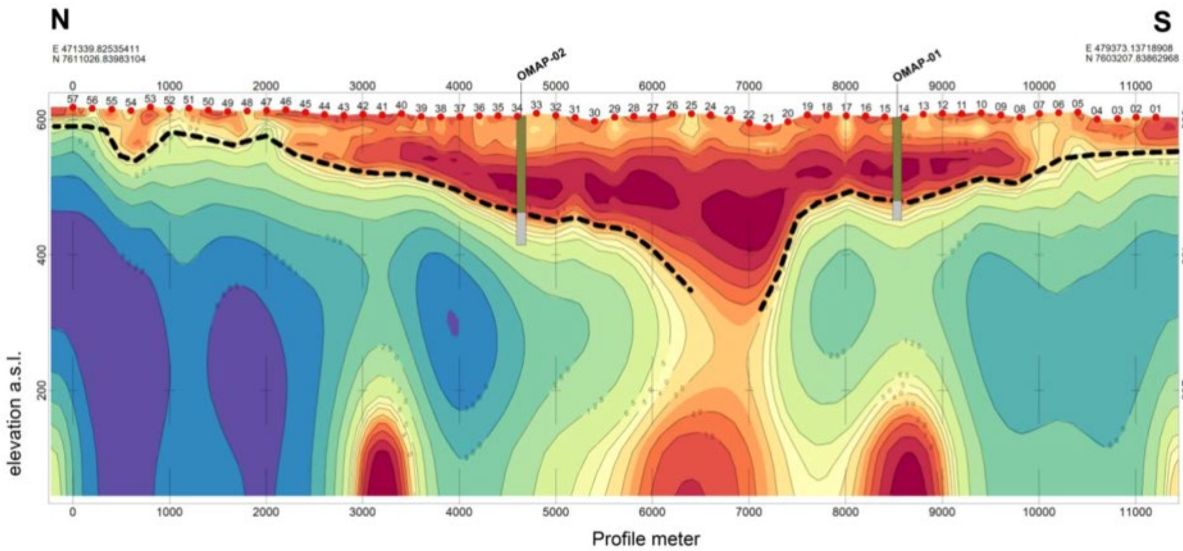


Figure 11.9: NSAMT Profile 6

Two exploration boreholes drilled as part of the DFS in 2026 provided new information on the OmAP palaeochannel aquifer system and confirmed that a large groundwater source is available to meet demand of the Project. Based on these preliminary results the following conclusions can be drawn:

- Groundwater levels rest between 150 and 207 mbgl with basement intercepted as deep as 318 mbgl.
- Pumping test results indicate that bore yields in the order of up to 15 m^3/h, under optimised borehole design conditions, are reasonable.
- Groundwater quality is brackish with TDS concentrations of between 2,800 mg/L and 4,000 mg/L observed.

The palaeochannel aquifer dimensions between the two exploration boreholes, which are approximately 20 km apart, as well as the aquifer parameters determined by test pumping, were used to estimate the abstractable volume of 85 Mm^3 (Figure 11.10 and Figure 11.11). This is considered to be a conservative figure as the actual length of the saturated palaeochannel is estimated to be more than 40 km.

Approximately 10 to 15 production boreholes, which could be pumped at 15 m^3/h and supply approximately 1.5 Mm^3/a of brackish groundwater to the Project, are proposed for the final development, as indicated in Figure 11.7.

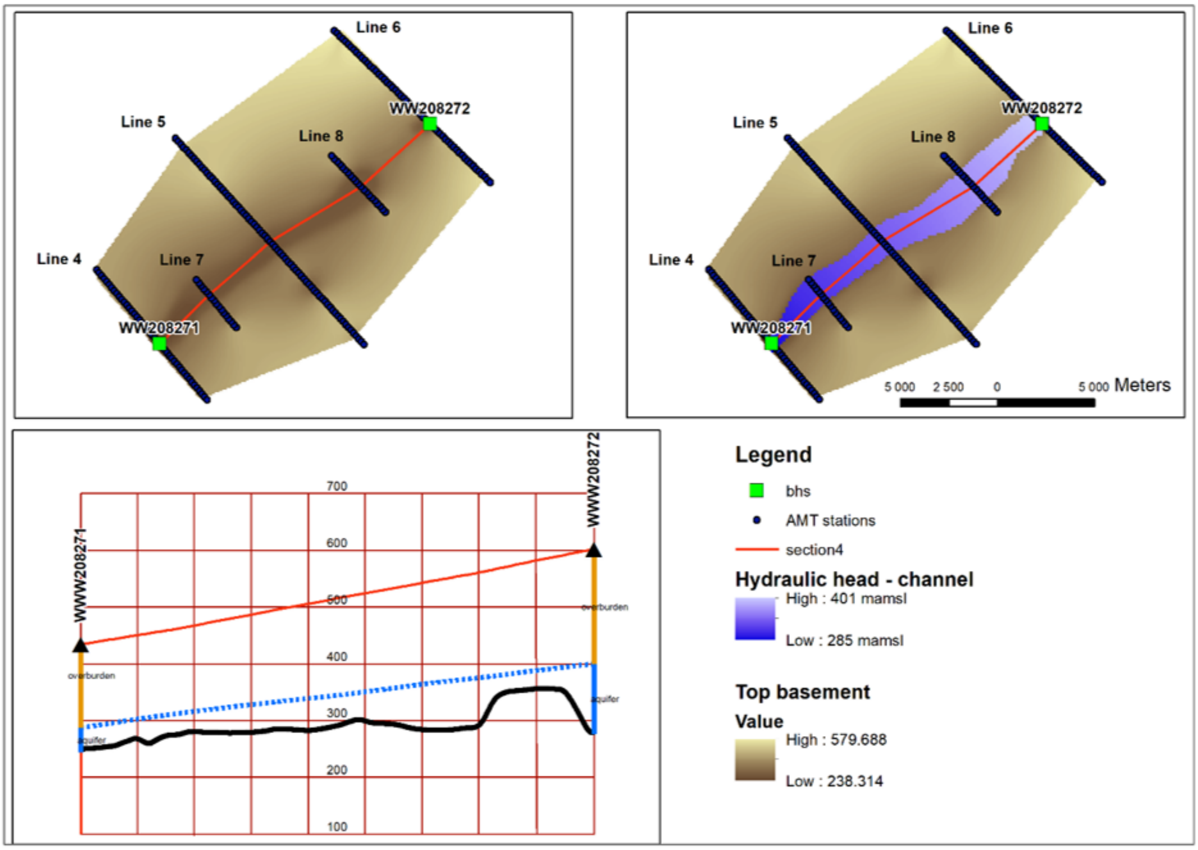


Figure 11.10: Conceptual OmAP Groundwater Model

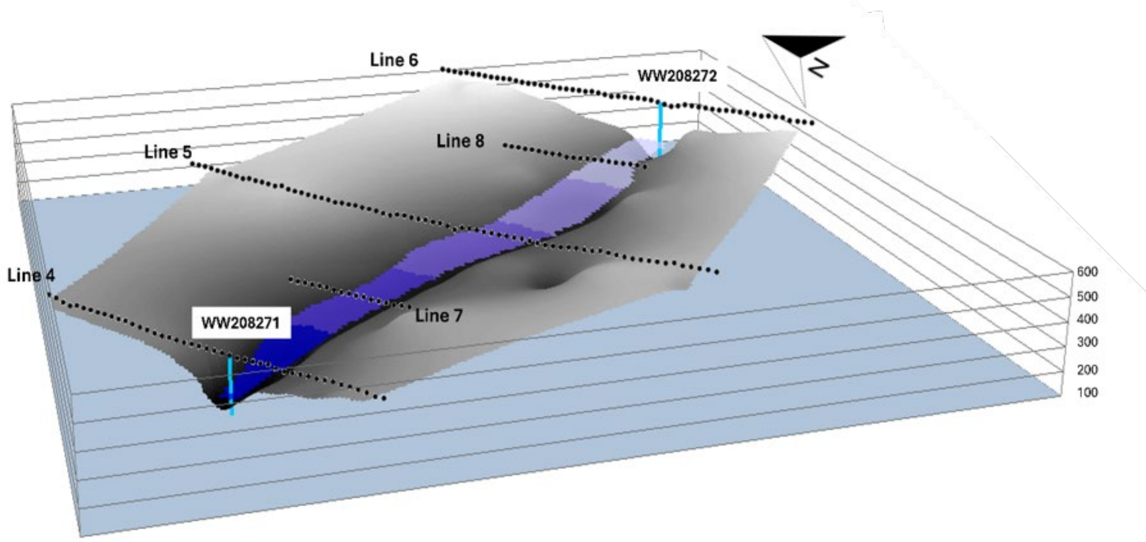


Figure 11.11: 3D OmAP Elevation

Drilling of a further six exploration boreholes at OmAP is currently underway and expected to be completed in the third quarter of 2026. Four of these will be larger 250 mm cased boreholes which will allow pump testing with a larger pump to assess higher abstraction rates. The other two smaller diameter boreholes, and the two existing exploration boreholes that are also smaller diameter, will be used as monitoring boreholes during the pump testing of the larger boreholes to assess the wider ground water drawdown and provide confidence in the aquifer characteristics and allow the development of a numerical groundwater model for OmAP.

11.3.4 Desalinated Water

Desalinated water can be sourced from Orano's EDP and provided at Orano's Trekkopje Mine some 47 km inland from the EDP. The EDP, located near Wlotzkasbaken, approximately 35 km north of Swakopmund, is currently operated by Nafasi Water. Orano (formerly Areva), at the time of construction of the EDP between 2007 and 2011, also constructed an 800 mm diameter above-ground pipeline 47 km inland to the Trekkopje Mine. This pipeline features a base pump station and two booster pump stations and was designed for a maximum flow rate of 2,740 m^3/h, equivalent to 20 Mm^3/a.

Discussions were held during the DFS with Orano regarding supply of desalinated water and a budget proposal was received outlining supply constraints, battery limits and pricing.

11.3.5 Conclusions

Following a detailed assessment of the costs, risks, technical considerations and implementation requirements associated with the various water supply options, the Project has adopted the following water supply strategy for the DFS:

- Construction water will be sourced from a combination of the local borefield on the EPL and water purchased from NamWater via an expanded Okombahe WSS. Water from the local borefield will be prioritised due to the higher cost associated with water supplied from Okombahe, which will be trucked to site.

- Operational water demand will be supplied from the OmAP borefield.

While these sources form the basis of the DFS, engineering designs and cost estimates were completed for all water supply options to preserve Project optionality, while additional confirmatory hydrogeological work is being undertaken at site and OmAP and environmental studies are underway.

Details of the proposed borefields, associated infrastructure and water supply systems incorporated into the DFS design are outlined below.

11.4 Borefields and Infrastructure

11.4.1 Omaruru Alluvial Plains

The OmAP borefield will compromise 13 boreholes, though it is anticipated that only 11 operating boreholes will be required at any time. The location and layout of the OmAP borefield is provided in Figure 11.12.

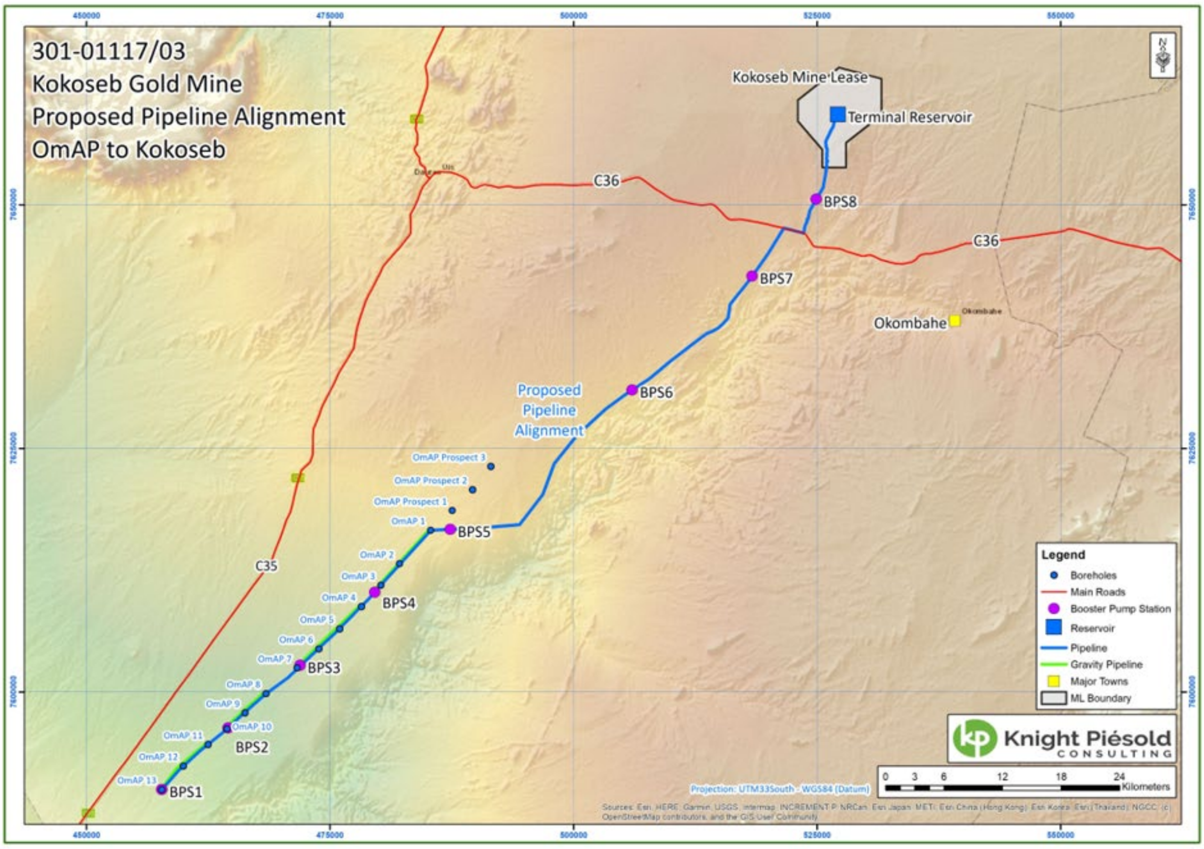


Figure 11.12: OmAP Borefield

Each borehole was designed for an average abstraction rate of 15 m³/h, providing a total supply capacity of 1.4 Mm³/a, sufficient for all operational needs of the Project. The groundwater is brackish (up to 3,500 TDS) and although not suitable for human or livestock consumption without treatment, is of adequate quality for gold processing and can be utilised for raw water supply without any specific treatment or process design requirements.

Eight pump stations will be required to transfer water to the Project site. The water from the various boreholes will be collected at four booster pump stations gravitating to the south-west after discharge at surface from the bore pumps into a gravity pipeline.

Between three and four boreholes will supply water to each booster pump station, which will be located adjacent to one of the bores such that the closest borehole can deliver water directly to the pump station reservoir and the bore and booster pump station can share power infrastructure.

Bulk water transfer will commence at pump station PS1 at the extreme south-west of the scheme from which water will be transferred to successive booster pump stations via increasing diameters of pipes (200 mm to 250 mm diameter) to PS4. From PS4 the water will be pumped by via four further pump stations in a 300 mm diameter above ground ductile iron pipeline mounted on concrete pedestals. The schematic of the borefield is show in Figure 11.13

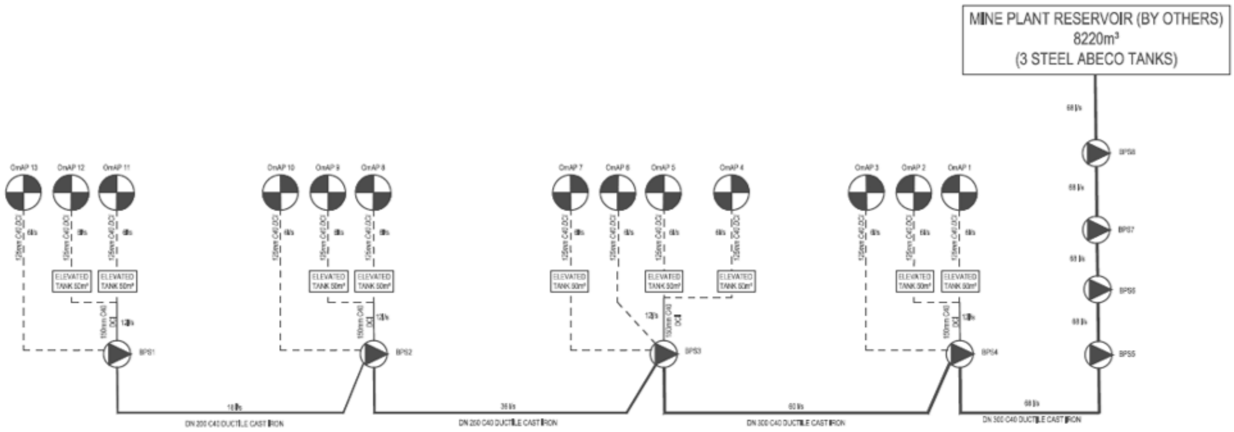


Figure 11.13: Schematic of OmAP Borefield

The power for the borefield will be supplied by a dedicated 33 kV OHPL fed from the Kokoseb HV Switchyard parallel to the pipeline. At each pump station or bore hole a pole or ground mounted transformer and metering will provide 400 V power for the facilities. The OHPL will also include fibre-optic communications will be provide monitoring and control of the borefield equipment.

11.4.2 Construction Water Supply

11.4.2.1 Near Mine Borefield

To support construction activities, a wellfield of 10 boreholes will be developed around the Project site, all situated on EPL 4148, although five of the boreholes are located outside the boundaries of the mining lease to the east and south-east. Each bore will be approximately 150 m deep, with a water depth of 20 mbgl and pumps installed at 145 mbgl. This borefield will supply approximately 212,430 m^3/a. Each borehole is expected to yield on average 5 m^3/h over 12 hours per day, allowing 12 hours for recharge. The layout of the EPL borefield is shown in Figure 11.14

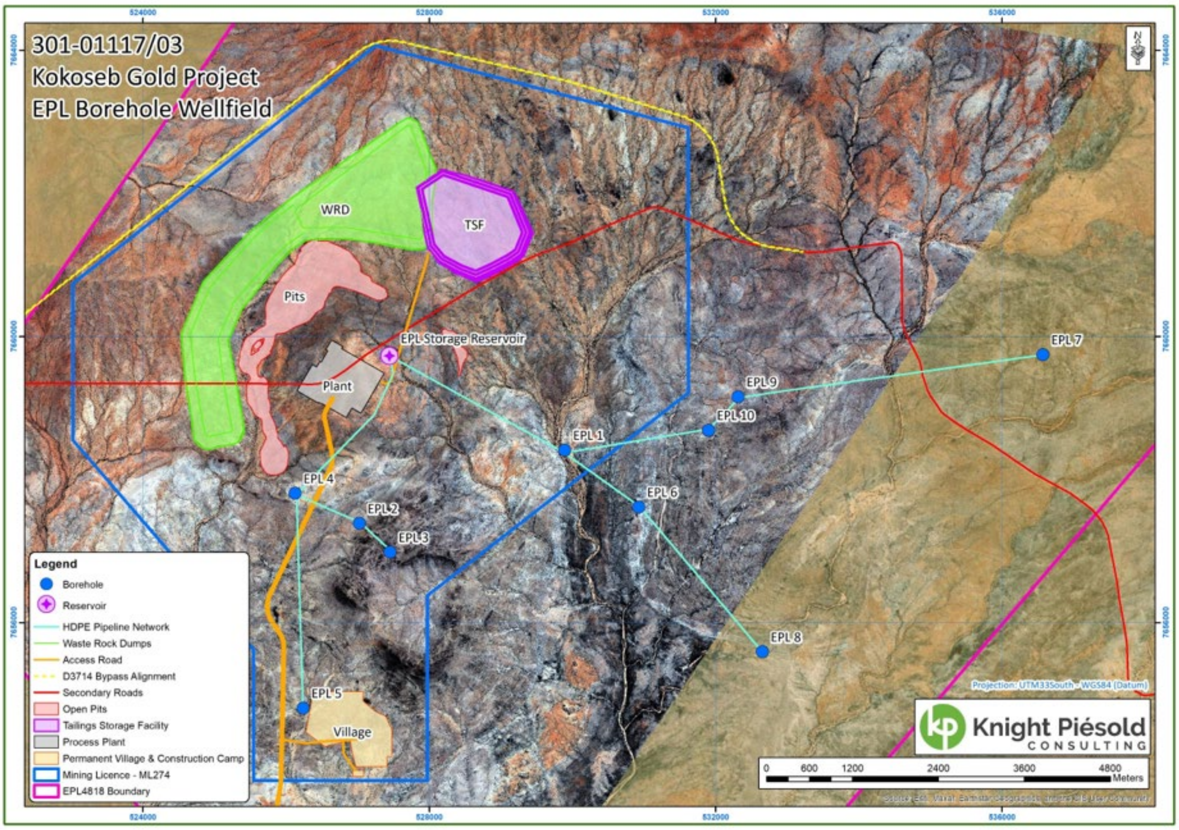


Figure 11.14: EPL Borefield Layout

The bores will be connected via two main pipeline “arms” such that alternating bores can be operated in tandem reducing the design flowrate in each of the arms and hence reducing the pipeline size. Since the borefield is only proposed for the construction phase of the Project, the pipelines will be run on surface in high density poly ethylene (HDPE) ranging in size from 50 mm to 90 mm and will feed to a 1,200 m^3 capacity reservoir close to the proposed bulk earthworks construction yard. Bores will be powered by local diesel driven generating sets and manually operated. Each bore installation will be fenced for security.

11.4.2.2 Okombahe Water Supply Scheme

Water supply for construction not available from the EPL borefield will be supplemented from an expansion of the Okombahe WSS. This will supplement the existing Okombahe WSS to provide capacity to provide 100% of the required water for construction if required, as outlined in Table 11.1.

Table 11.1: Okombahe WSS Construction Water Expansion

Water Source	Capacity/Usage kL/a
Water Supply	
Current NamWater Bores	459,000
Additional 3 Bores @ 7m^3/h, 20 h/d	153,000
Total Supply	**612,000**
Water Usage	
Current NamWater Usage	200,000
Construction Water Requirements	330,000
Total Usage	**550,000**
Surplus	62,000

Three new bores will be drilled close to the existing Okombahe WSS bores. The three new boreholes will be piped into a common 4.4 km buried uPVC header varying in size from 90 mm to 110 mm running to the existing NamWater reservoir located on the hill above the main part of Okombahe on the southern side of the Omaruru River. Each of the bores will be powered from the existing Erongo RED network installed at Okombahe.

Water supply for construction will be provided via a new above ground 160 mm HDPE gravity delivery pipeline from the NamWater reservoir which will be buried and run to the northern side of the Omaruru River adjacent to the D2306 road that runs to the C36 and site. The water will be fed into two temporary elevated high-flow storage tanks which will allow the rapid load-out of water into 38 kL capacity transport trucks for trucking to site.

12 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.
- Other infrastructure.

An overall site layout is provided in Figure 12.1, which shows the supporting infrastructure and services, mine, TSF and processing plant infrastructure.

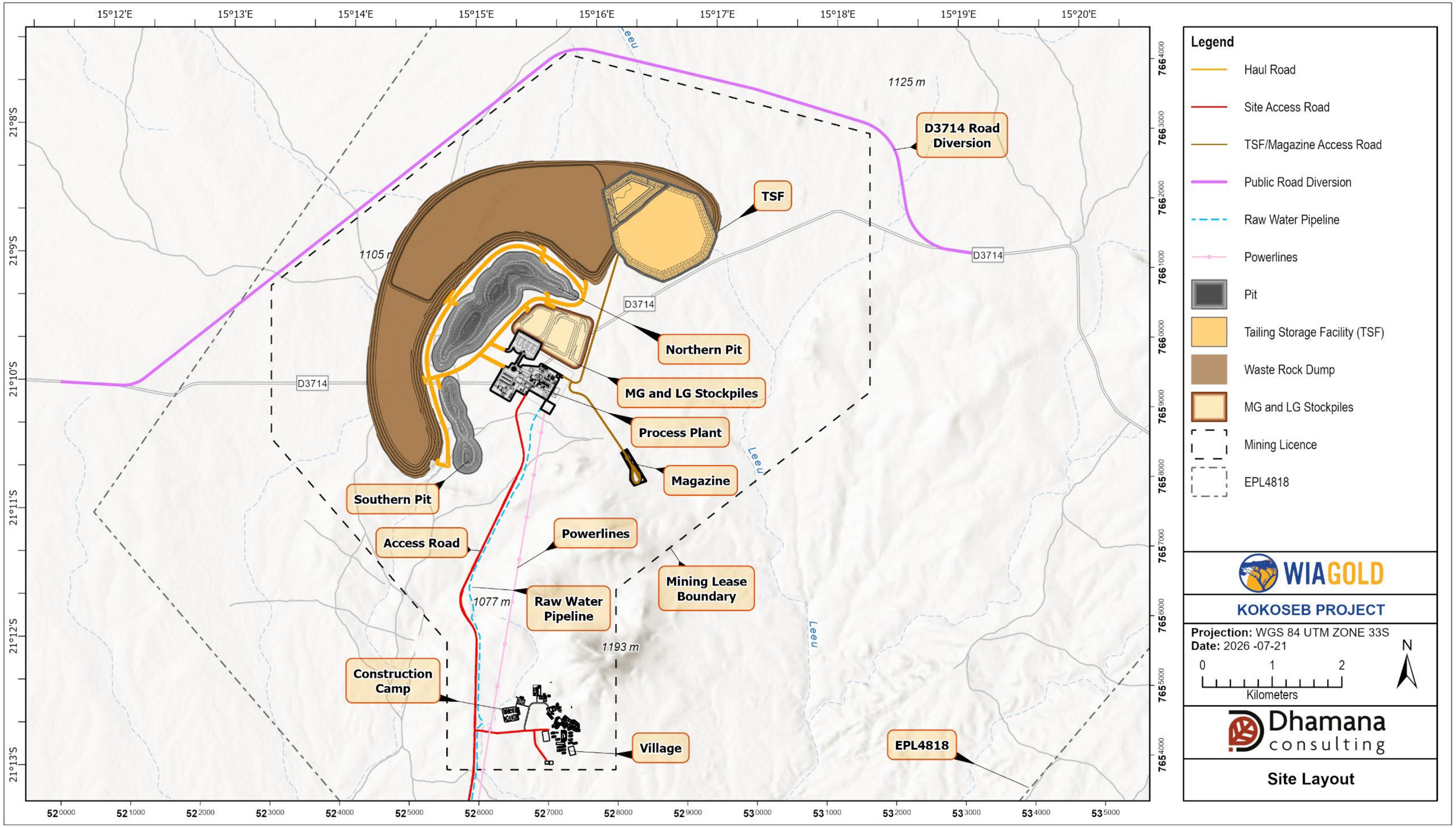


Figure 12.1: Overall Site Layout

12.1 Site Access

The all-weather unsealed C36 regional road between Omaruru and Uis runs through the southern portion of the exploration licence approximately 12 km south of the process plant site. In addition, the local D3714 road, which runs between Ni-Uis and Uis, traverses the process plant site from east to west.

Initial Project construction access will mirror exploration access via the D3714 until permanent access is established.

Permanent access to the Project will be enabled by construction of an all-weather unsealed road from the C36 approximately 18 km to the west of the Okombahe turn-off (C36 and D2306 intersection) from Omaruru. The access road, which will be approximately 13.5 km long, will pass by the NamPower metering substation and accommodation village before passing through the site boundary fence, with associated manned gate, and reach the process plant site main gatehouse.

The existing D3714 road will be diverted to the north of the proposed mining lease as part of the Project, maintaining public access at all times.

Access to the mine from the mining infrastructure area, which is co-located with the process plant, will be through a manned gate on the east of the security area to a network of mine roads providing access to the pits, stockpiles, waste rock dump and ROM pad.

Access to the TSF will be via a 2 km access road through a manned gate on the west of the security area before turning north to the TSF. Access to the explosives magazine will be off the TSF access road.

Transport for goods and personnel during construction and operations will be by road via Omaruru or Uis. The preferred route from Windhoek and South Africa will be through Okahandja, Karibib and Omaruru. Personnel transport from the northern parts of Namibia will also be via Omaruru. Seaborne freight will be imported through Walvis Bay, and any personnel from the coastal towns of Walvis Bay, Swakopmund or Henties Bay, will be transported through Henties Bay and Uis to site.

12.2 Offices, Workshops and Other Buildings

The key building infrastructure for the Project includes all non-mining facilities outside the processing plant that will be required to support the plant or processing functions. Processing plant infrastructure includes:

- Administration building, that will accommodate all management personnel including administration, security, and health, safety and environment. The building will consist of a combination of separate offices, open plan workstations and meeting rooms.
- Process plant office that will accommodate all process plant and maintenance office staff and consist of a combination of separate offices, open plan workstations and meeting rooms.
- Owners' mining office that will accommodate the owners mining team and will consist of a combination of separate offices, open plan workstations and meeting rooms.

- Plant control room.
- Clinic and emergency response building, which includes a consulting room, treatment room, offices and open plan along with a covered area for ambulance and emergency vehicle parking.
- Security buildings, including:
  - Site guardhouse, located on the site access road at the main Project fenceline.
  - Low-security gatehouse, located at the fenceline of the process plant area.
  - High-security gatehouse, located at the process plant security fenceline.
  - Mine guardhouse, located at the entrance to the mine area from the mining infrastructure area.
  - Tailings guardhouse, located on the TSF access road on the process plant areas fenceline.
- Two separate dining halls, one adjacent to the administration building and the other the process plant office.
- Fixed plant workshop including offices.
- Plant warehouse including offices.
- Separate reagent store buildings for cyanide, bagged lime and other general reagents.
- Tailings handling workshop and office building for maintenance of the tailings handling fleet.
- Laboratory.
- Male and female ablutions.

12.3 Accommodation Village

The accommodation village will be located approximately 1 km to the east of the site access road approximately 6 km from the process plant towards the C36 and will accommodate employees and contractors who are to be brought in from areas distant from the site.

The village will include management rooms, senior staff rooms, standard (junior staff) rooms and common catering and recreational facilities. The site is elevated to provide 270-degree views to the west and is separated from the mine and processing facilities by natural terrain. Key features of the proposed 536 bed accommodation village include:

- 15 management rooms (15 beds).
- 50 senior room buildings (100 beds).
- 104 dormitory accommodation rooms (416 beds).
- Kitchen and dining facility.
- Gymnasium.

- Wet mess facility.
- Laundry.
- Recreational areas.
- Maintenance workshop.
- Water and sewage treatment facilities.

Management rooms include a bedroom, ensuite, sitting room and small kitchenette. Rooms allocated to the most senior personnel will generally not be shared unless there is a back-to-back arrangement or a necessity to house consultant level visitors.

Senior rooms will be similar in size to management rooms but will have bedrooms at either end with a shared ensuite between them. Senior rooms may be motelled for back-to-back rosters and, hence, contain lockers for storing clothes and other items while away from the site.

Standard rooms will be dormitory style. Each room shall contain four beds and lockers. They will be arranged such that each building will share laundries and ablutions with facilities for male and female employees. The laundries will be used by junior workers to wash their clothes.

Central to the accommodation village will be a dry mess and central laundry facility. The central facilities will include:

- A 75-seat senior and 260-seat junior dining hall, complete with commercial kitchen, food preparation areas, larders and food storage areas (including a freezer, chiller room and dry store).
- Administration office including offices and a small shop.
- Medical clinic, incorporated within a dedicated accommodation unit for the resident nurse, with clinic and treatment space located at one end of the building.
- Wet mess facility, including alfresco area, male and female amenities, secure storage and refrigeration facilities, together with seating and entertainment areas.
- Gymnasium, including associated ablution facilities.
- Recreation building, providing informal seating areas, televisions and recreational facilities.
- Sports facilities, including a football pitch and a multi-purpose hardstand court.
- A significant laundry area for washing bedding, towels and other domestic fabrics, as well as a laundry service for senior personnel clothing. This also include storage rooms for linen and cleaning products.
- A covered docking bay for food and laundry deliveries.
- A carpark will also be located adjacent to the camp and will have provision for buses (including covered bus stops) and a parking bay dedicated to the clinic.
- Utility area in the camp that contains the fire and water distribution tanks, a step-down transformer and distribution board, a communication distribution building.

- Dedicated water services area will include potable water and a sewage treatment plant. Treated effluent from the plant will be used for irrigation or discharged into a leach drain.

The layout of the accommodation village is shown in Figure 12.2.

12.3.1 Construction Phase

During the construction phase aspects of the temporary accommodation facilities will be provided by the various construction contractors, which will include:

- Bulk earthworks contractor who will provide and install a fly camp complete with messing facilities.
- Process plant contractors, including civil contractor, structural, mechanical, platework and piping contractor, and electrical and instrumentation contractor who will install construction camps with various types of accommodation, recreation facilities and messing facilities.
- Permanent village contractor who will install a temporary fly camp complete with messing facilities for use until sufficient aspects of the permanent village are complete to cover the accommodation requirements to complete the permanent village construction.
- Owners' exploration team, along with the owners' project team and EPCM contractor representatives who will be accommodated in the relocated, and expanded, exploration camp adjacent to the exploration facilities. This will include accommodation, both tented and relocatable, ablution facilities and messing facilities. Following completion of the permanent village these facilities will be decommissioned, and the personnel will relocate into the permanent village.

12.3.2 Exploration Facilities

In addition to the accommodation facilities, facilities for the owner's exploration team's ongoing exploration will be relocated from the existing area at Okombahe or newly constructed. The facilities will include:

- Exploration offices, relocated.
- Ablutions, relocated.
- Core shed, new.
- Core cutting area, relocated equipment.
- Storage containers, relocated.

The costs for the establishment of the exploration facilities, other than earthworks and services, are not included in the Project cost estimate and are covered under the corporate exploration budget.

12.3.3 Temporary Services

For the construction period the central services will be expanded to accommodate the additional services required. This will include:

- Power generation through diesel generating sets (with one unit remaining following construction to provide emergency power).
- Potable water treatment.
- Waste water treatment.

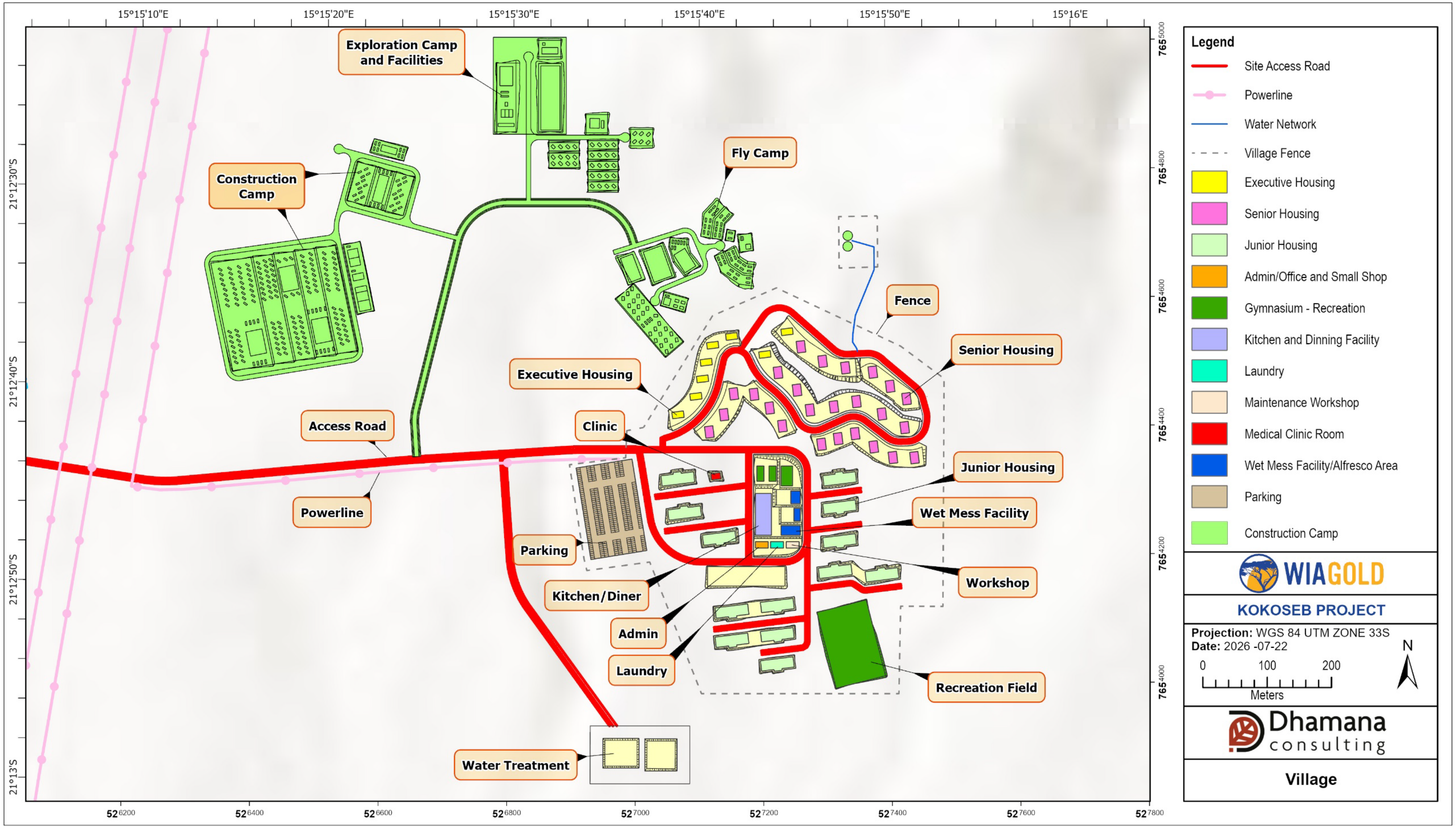


Figure 12.2: Accommodation Village Layout

12.4 Power Supply and Distribution

12.4.1 Estimated Load

The estimated electrical load for the Project is outlined in Table 12.1.

Table 12.1: Estimated Site Installed Power and Average Continuous Load

Area	Installed Power (MW)		Average Continuous Draw (MW)
	Total	Operating	
Primary Crushing	2.0	2.0	1.0
Milling and Leaching	32.5	31.3	23.3
Tails Treatment	5.5	5.4	3.4
Reagents and Services	3.8	2.9	1.0
NPI	0.6	0.6	0.4
Village	0.5	0.5	0.4
Mining	0.5	0.5	0.4
Total (excluding Borefields)	**45.4**	**43.2**	**29.8**

In addition to the site power requirements, the Project water supply system will also require electrical power for the operation of borefields, transfer pumps and booster pumps. The estimated power demand associated with the Project water supply system is summarised in Table 12.2.

Table 12.2: Water Supply Installed Power and Average Continuous Load

Water Source	Water Supply (Mm^3/a)	Installed Power (kW)		Average Continuous Draw (kW)
		Total	Operating	
OmAP Borefield	1.4	2,424	2,380	1,173

12.4.2 Options Assessment

An application was made for power supply for the Project to NamPower, the parastatal power utility in Namibia, and they investigated a number of options, including:

- 66 kV power supply via the existing Omburu-Uis OHPL.
- 66 kV power supply from the NamPower Omburu substation with a new 66 kV OHPL parallel to the existing OHPL to Uis.
- 132 kV power supply from the NamPower Omburu substation with a new 132 kV supply developed at Omburu and a 132 kV OHPL parallel to the existing OHPL to Uis.
- 132 kV power supply from the NamPower Trekkopje substation with a new 132 kV OHPL run to site.
- 220 kV power supply from the NamPower Omburu substation with a new 220 kV OHPL parallel to the existing OHPL to Uis.

Of these options, the 66 kV options were determined to be technically infeasible under NamPower operating guidelines. In addition, the 220 kV option was deemed excessively expensive.

Of the two 132 kV supply options, the Omburu supply necessitates the development of 132 kV supply at Omburu (which currently only operates 330 kV, 220 kV and 66 kV transmission voltages), however the Trekkopje supply is further from site, increasing the transmission distance. Both options were considered to be similar in cost and development timeline, however the risks associated with land servitude were considered less for the Omburu option due to the parallel route to the existing Omburu-Uis OHPL. Consequently, the option for 132 kV supply from Omburu was selected.

Consideration was also given to “behind the meter” renewables generation, however given the cost of grid power in Namibia the economic incentive is relatively low, as well as the additional project delivery complexity, it was decided to defer such implementation until post commencement of production.

12.4.3 Grid Connection

A 132 kV supply will be developed at the NamPower Omburu substation. This will involve the extension of the 220 kV busbars, extension of the substation yard terrace and construction of a new 132 kV transformer bay with a 40 MVA transformer and feeder bay. A new control building containing protection relays and fibre-optic SCADA interface will also form part of the new 132 kV development. An installed standby transformer is not required under NamPower guidelines as spare transformers with the same specification are held across the NamPower network. The terrace extension and new 132 kV yard development will make allowance for a future second 40 MVA transformer bay, to be developed by NamPower as and when required.

Expansion of the Omburu terrace, access road and switchyard will be required to accommodate the new 132 kV switchyard along with minor re-routing of the 66 kV OHPL to Karibib to allow this to be routed outside of the new 132 kV switchyard.

From the new 132 kV feeder bay the proposed grid-connection is a greenfield 132 kV monopole OHPL, designed to NamPower standards, planed parallel with the existing wood-pole 66 kV OHPL to Uis and easement corridor for approximately 90 km. The 132 kV OHPL will terminate at the NamPower Kokoseb metering substation, which will be located on the Project site access road, approximately 2.5 km from the intersection with the C36. Ownership of the facilities up to this metering substation will be granted to NamPower by means of a donation agreement for zero cost under the proposed power supply agreement.

A preliminary routing study and tower spotting was undertaken based on satellite topography with the proposed power line route shown in Figure 12.3.

From the NamPower metering station a 132 kV OHPL will be constructed to the site high voltage (HV) switchyard where the voltage will be stepped down to 11 kV for distribution around site.

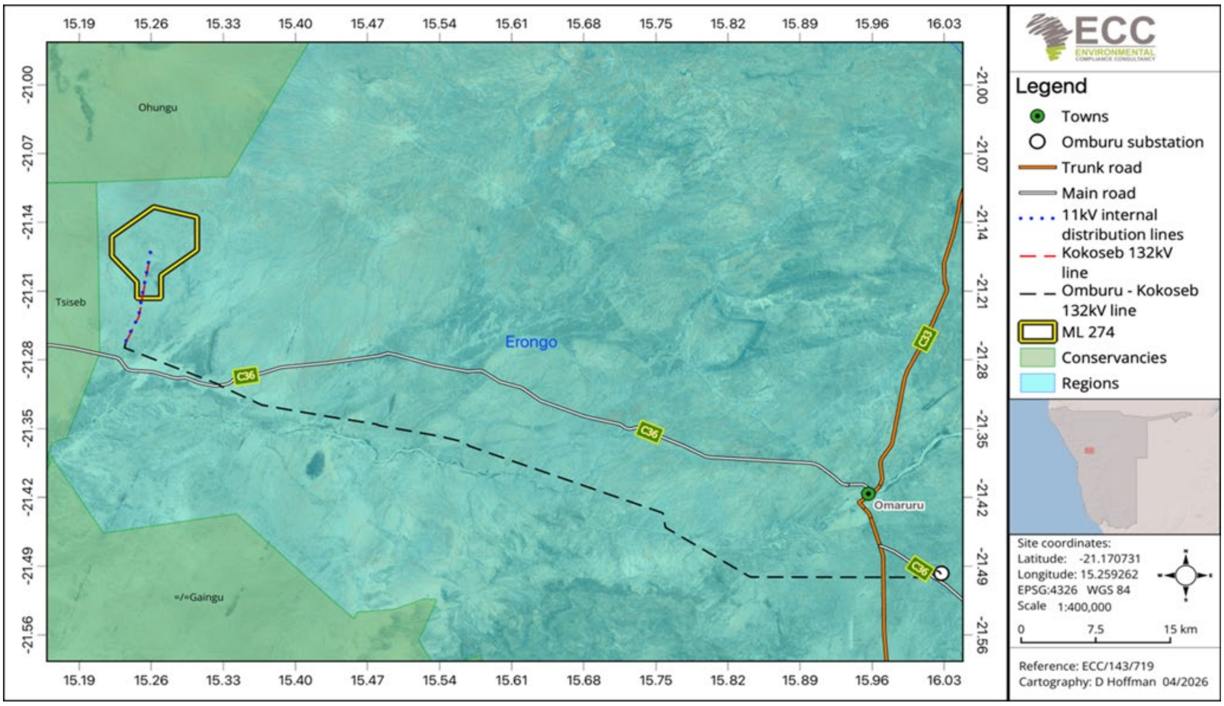


Figure 12.3: OHPL Route

Extensive engagement with NamPower was undertaken throughout the DFS process, resulting in the receipt of a formal power supply offer in May 2026. Wia has since requested amendments to the offer to incorporate a customer-build approach. An updated offer is anticipated in August 2026, after which negotiations towards a Power Supply Agreement will commence.

12.4.4 Power Distribution

Medium voltage power (11 kV) will be distributed from the main site HV switchyard by underground cable to the plant HV switch room and by 11 kV OHPL to the village. In addition, the site HV switchyard will also incorporate static synchronous compensators, a form of dynamic reactive power compensation, for power factor correction across the site. The site HV switchyard will also include spare feeder bays for additional feeds, if required at a later date.

11 kV power throughout the process plant will be distributed from the plant HV switch room to the process plant through underground and above ground cables. In addition, supply to the NPI will be from the plant HV substation, which will feed a series of ring main units to the various NPI facilities.

The 11 kV OHPL to the village will run parallel with the main 132 kV OHPL coming into site before diverting to the village for stepdown and distribution.

12.4.5 Emergency Power

Emergency power will be provided by diesel fired generating sets at both the plant site and the village. At the plant site a total of 6 MVA of capacity will be installed, feeding onto the plant HV switch room 11 kV busbar, which will allow critical plant emergency drives to remain operational to protect the integrity of the equipment and facilitate rapid restart. The emergency generation capacity will not be sufficient to continue production. Power will also be maintained to NPI facilities from these emergency generators, particularly security, emergency services, information technology and offices.

The village will have emergency power provided by a 1 MVA diesel generator in the event of an outage on the OHPL feed to the village or power supply to the site. This capacity will be suitable to provide for full operations of the village during the grid power outage.

12.4.6 Construction Phase Power

Power during the construction phase of the project, including for offices, workshops and construction water supply bores will be provided by mobile diesel generating sets.

Power for the village during the construction phase of the Project will be provided by the permanent emergency diesel generating sets which will be installed early in the construction sequence of the village. Additional temporary generating sets will be added as required to meet the load.

12.5 Mining Infrastructure

The bulk of the mining support infrastructure will be provided as part of the mining contract by the mining contractor. The mining infrastructure required for the Project, including delineation of supply, includes:

- Earthworks, including hardstand, drainage, fencing and surface water management, installed by Wia.
- Buildings provided by the mining contractor, including:
  - Contractors' offices.
  - Contractors' ablutions.
  - Contractors' crib building.
  - Heavy and light vehicle workshop.
  - Warehouse.
- Construction of any go lines and maintenance bays, including area lighting, by the mining contractor, in the area prepared by Wia.
- Heavy and light vehicle washdown bay and systems, including water cannons, hose reels, silt trap and oily water separator, supplied and installed by the mining contractor and for use by the owners mining team.
- High-flow and low-flow fuel dispensing bowsers installed by Wia for the use of both mining contractor and the owners mining team.
- Services provided and installed to a common point for the mining contractor by Wia, including:
  - Power supply.
  - Potable water supply.
  - Raw water supply (for washdown).

- Wastewater collection and treatment in the wastewater treatment plant.
- Communications via optical fibre for offices.
- Two-way radio repeater system, with radios provided by the mining contractor.
- Private cellular network for the use of the mining contractor for implementation of a fleet management system.
- Raw water delivered from the raw water dam to an offtake point in the mining area for mine dust suppression.

12.6 Fuel Storage and Distribution

Fuel storage for the Project will comprise bulk diesel storage to support the mining fleet, mobile equipment, process plant heaters, regeneration kiln and standby power generation facilities.

The diesel fuel storage facility will initially consist of five self-bunded diesel storage tanks, each with a capacity of 72,000 L, configured in a master-slave operating arrangement. The facility is designed as a modular system, allowing the number of tanks to be increased or decreased over the life of mine to match long-term fuel consumption requirement.

Diesel fuel will be held on consignment by the fuel supplier, who will replenish stocks through daily deliveries from their bulk fuel storage facilities in Walvis Bay. This arrangement minimises on-site inventory risk while ensuring a secure and reliable fuel supply.

Fuel from the storage will be pumped to the process plant day tank, back-up diesel generators day tank 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 washdown bay oily-water separator.

The diesel storage and distribution infrastructure will be delivered under a build-own-operate arrangement, with the capital and operating costs recovered by the fuel supplier through a service charge incorporated into the fuel supply contract.

12.7 Other

Other infrastructure, services and utilities required for the Project include:

- Surface water management infrastructure around the pits, 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 GridOnline, provided by NamPower, backbone network for data and voice communication, UHF radio repeater tower, portable UHF radios and a 4G/LTE tower for data communication with the mining fleet.
- 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 bore infrastructure).
- Double 1.8 m high fencing around the process plant high-security area.

- Access control and security monitoring systems.
- Waste disposal facilities (landfill) for putrescible and non-recyclable waste, with recyclable waste to be sent off site for recycling.

13 OPERATIONS

13.1 Operations Strategy

Operations across the Project will generally be conducted on a 24 hours per day basis, with most personal working 12-hour shifts.

To access specialist expertise and utilise dedicated equipment, the following operational activities will be undertaken by suitably qualified and experienced contractors:

- Open pit mining, including drill and blast activities, ROM pad operations and mine road maintenance.
- Filtered tails handling to the TSF, including tailings placement and maintenance of the filtered TSF.
- Laboratory services, including provision of equipment, consumables, systems and personnel.
- Camp management services, including catering and mid-shift meals for all personnel, camp housekeeping, site office cleaning and laundry services.
- Site medical services.

It is anticipated that all other Project activities will be carried out by Company employees.

13.2 Logistics

The majority of reagents, consumables and spare parts required for the Project will be sourced internationally and imported through the port of Walvis Bay or transported by road from South Africa.

A logistics contractor will be appointed to manage the delivery of general freight to site, including port handling management, customs clearance, inspection management, freight consolidation in Walvis Bay and transport to site.

The following exceptions will be managed separately:

- Fuel deliveries from bulk storage facilities in Walvis Bay, will be undertaken by the fuel supply contractor.
- Explosives will be delivered by the explosives supplier, or its nominated transport contractor.
- Lime will be transported from the suppliers facility in South Africa, either by the supplier or by a dedicated transport contractor.

This logistics strategy is intended to streamline freight management, optimise supply chain efficiency, and ensure the reliable delivery of Project materials and consumables.

13.3 Ramp-up

The ramp-up of gold processing plants, given their simplicity and well understood nature, is generally quite rapid. Based on industry data from several process plant engineering contractors, gold processing plants typically reach steady state operations within 10 to 60 days of the introduction of ore commissioning, with an average ramp-up period of 30 days.

However, the Project incorporates tailings filtration and stacking, which introduces additional operational complexity and may lead to a longer ramp up. Taking this into consideration and based on the experience and recommendations of the DFS consultants and other industry specialists, the ramp-up schedule presented in Table 13.1 was adopted for the DFS.

Table 13.1: Kokoseb 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 or reductions of instantaneous throughput, availability, utilisation and metal recovery.

13.4 Human Resources

13.4.1 Workforce Requirements

The high-level organisation chart for the Project is presented in Figure 13.1.

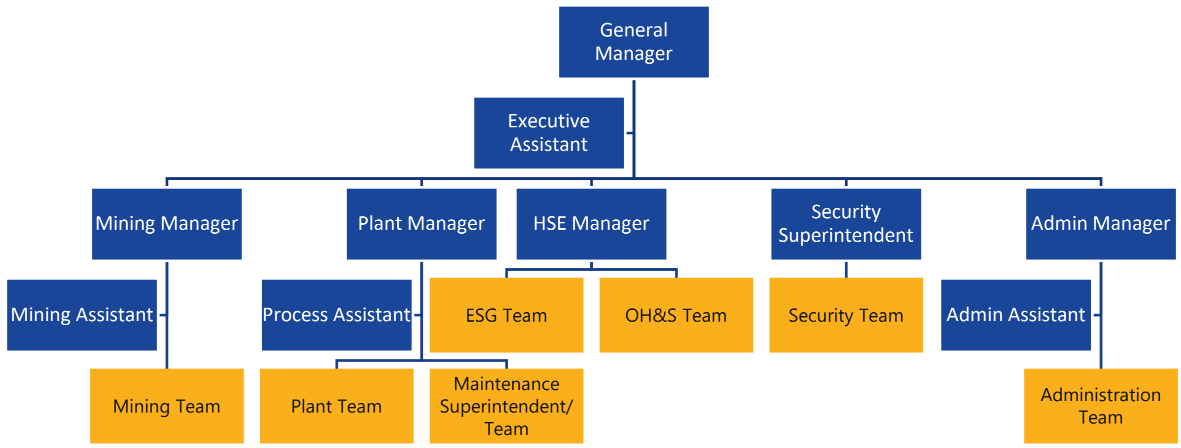


Figure 13.1: High-level Organisation Chart

This shows the operation split into five key departments being:

- Mining:
  - Technical Services, including geology, mine engineering, and surveying functions.
  - Open pit mining, responsible for managing the mining contractor and overseeing day-to-day mining activities.

- Process plant:
    - Metallurgical and engineering personnel.
    - Process plant operations, including shift operations teams, daytime operations support teams and a TSF operations team.
    - Process plant maintenance, including mechanical and electrical maintenance teams, maintenance planning, and workshop support services.
    - Laboratory operations.
- Health, safety and environment:
    - Environmental management and community relations teams.
    - Occupational health and safety teams.
    - Site medical personnel, including emergency paramedic and nurse on site at all times.
- Security, managing all aspects of site security.
- Administration:
    - Human resources.
    - Finance.
    - Warehouse and procurement.
    - Information technology.
    - Camp operations.

A summary of the anticipated Project workforce by department is provided in Table 13.2, including a breakdown of expatriate and Namibian personnel, as well as contract personnel involvement across the operation.

Table 13.2: Kokoseb Operations Staff

Department	Expatriate	Namibian	Total
Staff			
Management/Administration/Security	5	98	103
Processing	6	89	95
Mining[1]	6	20	26
Total Staff	**17**	**207**	**224**
Contractors			
Management/Administration		54	54
Processing		10	10
Mining[1]	1	233	234
Total Contractors	**1**	**297**	**298**
Total	**18**	**504**	**522**

Notes:

1. Mining labour numbers are based on requirements during the steady state period of mining operations.

Personnel and contractors will predominantly be accommodated in the accommodation village for the duration of their roster cycles and transit home when off roster.

The General Manager will work on a six week on, three week off roster, while regional expatriates will work on a six week on, two week off roster. Nambian employees residing outside the immediate Project area (i.e. not in Uis or Omaruru) will work on a four week on, two week off roster.

Where practical, employees from the immediate surroundings will be employed in day shift roles that allow them to work 10-hour shifts and commute daily between their homes and the operation using Project provided transport.

Medical services will be provided under a dedicated contract, with a minimum of one paramedic and one clinical nurse on site at all times. Medical personnel will operate on a six week on, six weeks off roster. While rostered on site, both personnel will be available during day shifts and will provide on call coverage for night shift operations.

13.4.2 Regulatory Context

Namibia's employment legal framework is governed primarily by the Labour Act, 2007, supported by constitutional worker protections. Compliance with legislative requirements relating to employment contracts, working hours, remuneration, social security, termination of employment, and dispute resolution, is a key consideration for the Project.

The principal legislation applicable to the Project includes:

- Labour Act, 2007, which is the principal statute regulating employment relationships.
- The Namibian Constitution. Enshrining Namibia's fundamental worker rights and underpinning labour protections.

- Supporting legislation and regulations, including those relating to social security, compensation for occupational injury or disease, and sector-specific health and safety requirements.

A continuous operation must be declared or authorised in terms of the Labour Act, and the applicable conditions of continuous shifts must be observed. Shift work within such arrangements must be distinguished from overtime. Employees work according to a structured shift system which enables continuous operations and overtime provisions still apply as normal.

Overtime will be worked in accordance with an agreement and may not ordinarily exceed 10 hours per week or three hours per day, unless otherwise authorised. To account for overtime for continuous shift operators, the project allowed for ordinary overtime to be paid at 1.5 times the hourly basic wage for personnel working more than 195 hours per month.

13.4.3 Workforce Sourcing

Namibia has a more established mining technical base than many greenfield mining jurisdictions within in the region and across Africa. The country supports several operating gold, uranium and other large-scale mines, and benefits from a local training pipeline through institutions such as Namibia University of Science and Technology, University of Namibia and Namibia Institute of Mining Technology, which collectively produce mining engineers, metallurgical engineers, geologists and artisans (tradespersons).

However, availability of skills varies by discipline and level of experience. In general, the local market provides a strong pipeline of operators, artisans and junior technical personnel, while there is relative scarcity of senior open pit mining professionals, advanced technical services personnel, metallurgical leadership, and higher-end maintenance planning, reliability engineering and instrumentation skills.

For the Project, this supports a workforce model that enables a significant proportion of operational and technical roles to be filled by Namibian employees over time, while continuing to rely on a combination of Namibian drive-in drive-out (DIDO), expatriates, and selected outsourced capability in those disciplines where local capability remains limited.

It is anticipated that more senior roles will be recruited where possible from within Southern Africa (Namibia, South Africa, Botswana, Zimbabwe, Zambia) or more broadly across the African continent.

This approach is expected to provide access to the skills and experience necessary to support the safe and efficient development and operation of the Project while maximising opportunities for local employment and skills transfer.

13.5 Security

A dedicated security team will form part of the owner's team workforce to provide security services during both the construction and operational phases. The security department will comprise approximately 46 personnel, including four-day shift positions and approximately 13 personnel on each 12-hour shift. Security personnel will be deployed across the following areas:

- Management and supervision of all security operations.
- Access control at key entry points including the site gatehouse, process plant gatehouse, gold room entry, village gatehouse, mine gate and tailings gate.
- Operation and monitoring of closed-circuit television (CCTV), including supervisory and operator roles.
- Security management of the gold room, including supervisory and access control personnel.
- Mobile patrol officers responsible for monitoring the process plant perimeter, administration and mining infrastructure areas, and the broader site boundary.

This security structure was designed to provide comprehensive site coverage, maintain controlled access to sensitive operational areas, and safeguard personnel, assets and product throughout the Project lifecycle.

14 ENVIRONMENTAL, SOCIAL, APPROVALS AND LAND ACCESS

This chapter outlines the regulatory and procedural requirements governing the Project, including environmental management, social engagement, statutory approvals, and land access. The environmental impact assessment (EIA) process in Namibia is regulated by the Environmental Management Act (2007) (EMA) and the EIA Regulations of 2012, providing a systematic framework to assess and mitigate potential environmental and social impacts. An environmental clearance certificate (ECC) from the Ministry of Environment, Forestry and Tourism (MEFT) is mandatory for mining and resource extraction activities listed in the EMA, with additional permits and approvals required from other authorities.

WIA engaged Environmental Compliance Consultancy (Pty) Ltd (ECC Ltd) as the environmental assessment practitioner to conduct an environmental and social impact assessment (ESIA) and develop an environmental and social management plan (ESMP) as prescribed in legislation. The ESIA was completed and submitted to MEFT for review in March 2026 and provides the detailed assessment of the environmental and social risks associated with the Project and provides management and monitoring actions for the management of these risks. The review by MEFT is ongoing and typically takes a period of nine months from submission.

In addition to the ESIA for the Project that was submitted, additional approvals related to off-site infrastructure, including the power supply and three water supply options, were commenced by ECC Ltd as part of the DFS activities.

14.1 Land Access and Permitting

The Project is located on communal land within the jurisdiction of the !Oe≠Gân Traditional Authority and is therefore subject to both customary and statutory land tenure arrangements in Namibia.

Access to communal land for mining activities is regulated by the Communal Land Reform Act, 2002 and the Minerals (Prospecting and Mining) Act, 1992. While minerals rights are vested in the State and administered by the MIME the occupation and use of communal land for mining purposes requires the consent of the relevant traditional authority and approval from the Communal Land Board. These legislative frameworks also establish obligations to compensate affected land users for loss of, or interference with, their rights and assets and, where necessary, to implement appropriate relocation measures.

In addition, given the Project's intention to align with international lender requirements, land access, compensation and resettlement activities will guided by IFC Performance Standard 5 (Land Acquisition and Involuntary Resettlement). This includes commitments to avoid or minimise displacement wherever practicable, provide compensation at full replacement cost, and restore or improve the livelihoods of affected persons.

Baseline investigations completed as part of the ESIA process, together with the homestead census undertaken in 2025 and a verification survey completed in June 2026, identified 18 homesteads comprising 43 project affected persons (PAPs) that may be subject to physical displacement, economic displacement, or a combination of both. Current Project planning indicates that ten homesteads (27 PAPs) may require physical relocation and associated livelihood restoration measures, while a further eight homesteads (18 PAPs) may experience economic displacement arising primarily from changes to access to communal grazing, livestock movement corridors, water sources and related infrastructure.

The number of affected homesteads identified through Project planning is consistent with the findings of the homestead census and verification survey. The outcomes of these studies, together with the agreed cut-off date of 18 May 2026, are documented in the standalone Verification Report and will form the basis for the ongoing resettlement planning and compensation process.

The current EPL, the proposed ML area and the locations of the homesteads situated within these areas are shown in Figure 14.1.

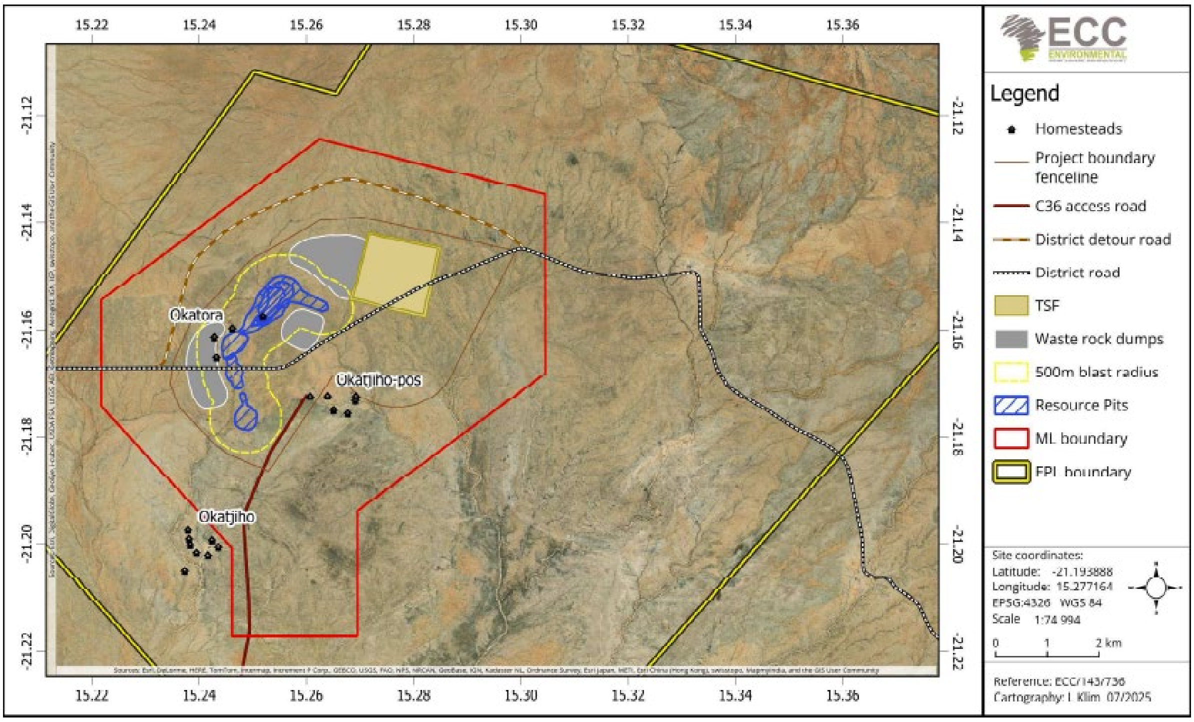


Figure 14.1: Kokoseb Project EPL and Proposed ML

Stakeholder engagement, compensation and relocation activities will be implemented through a relocation action plan (RAP), which will be developed based on, and expanded upon, the existing relocation planning framework completed in 2026.

The host community and its traditional leadership remain supportive of the Project. The !Oe≠Gân Traditional Authority recently issued a letter to the MIME supporting the grant of the Mining Licence (ML 274) for the Kokoseb Gold Project. The letter also reaffirmed the Traditional Authority's support for the exploration, environmental and social studies, and stakeholder engagement activities undertaken to date.

While this support provides an important basis for ongoing Project development, it does not in itself confer land access rights or tenure security. The remaining land access, compensation and relocation arrangements will be formalised through the applicable statutory approval processes, including obtaining the necessary approvals from the Erongo Communal Land Board and entering into individual agreements with affected households. These processes will be undertaken in accordance with the requirements of the Communal Land Reform Act, the

Minerals (Prospecting and Mining) Act, and IFC Performance Standard 5, with a focus on fair compensation, transparent engagement, and the restoration of affected livelihoods.

The Project area is also located within a designated groundwater-controlled area under the Water Resources Management Act, 2013 and associated Water Resources Management Regulations, 2023. Development and operation of water supply infrastructure within this area must therefore comply with regulatory requirements relating to borehole construction, aquifer protection, licensing and record-keeping.

A summary of the overall permitting and licensing requirements for the Project is outlined in Table 14.1.

Table 14.1: Permitting and Licencing Requirements

Permit/Licence	Act/Regulation	Related Activities Requiring Permit	Relevant Authority	Timeframe
Environmental clearance certificate	Environmental Management Act (EMA) 2007	Required for all listed activities shown in the EMA Annexure, including: • Energy generation, transmission and storage. • Waste management, treatment, handling and disposal. • Mining and quarrying. • Forestry activities. • Land use and development activities. • Water resources development. • Hazardous substance treatment, handling and storage.	Ministry of Environment, Forestry and Tourism (MEFT): Department of Environmental Affairs and Forestry (DEAF)	ESIA that forms part of the application submitted March 2026. MEFT require approx. 6 to 12 months for review and record of decision
Mining licence	Section 90 (2) (A) of the Minerals Act, 1992	All mining activities other than small scale mining, which is carried out under a Mining Claim.	Ministry of Industries, Mines and Energy (MIME): Directorate of Mines	Application submitted October 2025. MIME require approx. 6 to 12 months for processing and intention to grant, also requires submission of a project feasibility study (DFS)

Permit/Licence	Act/Regulation	Related Activities Requiring Permit	Relevant Authority	Timeframe
Surface rights agreements (mine, infrastructure corridors)	Section 52(1)(A) of the Minerals Act and Section 18(1) of the Communal Land Reform Act 2002	Traditional authority consent will be required. Additionally, several community residents residing on the Project area will need to be relocated. A relocation and compensation agreement will need to be negotiated between the Proponent and these residents.	Ministry of Industries, Mines and Energy (MIME): Directorate of Mines Ministry of Agriculture, Fisheries, Water and Land Reform: Department of Land Reform, Resettlement and Regional Programme Implementation	Typically 6 months, although it varies based on the nature and density of landholding.
Accessory works permit	Section 52 (1) (D) of the Minerals (Prospecting and Mining) Act 1992	Erection of any accessory works or supporting infrastructure such as roads, powerlines, pipelines, camps, or processing facilities necessary for mining (both within and outside the mining licence area)	Ministry of Industries, Mines and Energy (MIME): Directorate of Mines	Application submitted May 2026. MIME require approx. 2 to 3 months for grant.
Drilling or construction of borehole or well	Section 56 of the Water Resource Management Act 2013 and Section 44 (1) and (2) of the Water Resource Management Regulations 2023	Licence to drill, construct, deepen, enlarge, alter, clean or rehabilitate a borehole, or carry out a borehole drilling programme.	Ministry of Agriculture, Fisheries, Water and Land Reform (MAFWLR): Department of Water Affairs (DWA)	Permits in place for all exploration bores with new permits taking approximately 6 months
Abstraction licence	Section 44 (1) – (3) of the Water Resource Management Act and Section 44 (1) and (2) of the Water Resource Management Regulations	Licence to abstract groundwater from boreholes both on site and off site for water supply to mining activities.	Ministry of Agriculture, Fisheries, Water and Land Reform (MAFWLR): Department of Water Affairs (DWA)	2 - 3 months

Permit/Licence	Act/Regulation	Related Activities Requiring Permit	Relevant Authority	Timeframe
Licence to remove riparian species, fill or excavate riparian zone or erect structural development within riparian zone	Water Resources Management Act and the Water Resource Management Regulations	Construction of infrastructure in a water course requiring the removal of vegetation or excavation in a riparian zone.	Ministry of Agriculture, Fisheries, Water and Land Reform (MAFWLR): Department of Water Affairs (DWA)	3 – 9 months
Wastewater discharge licence	Section 72 of the Water Resources Management Act and the Water Resource Management Regulations	Required should there be any discharge of wastewater and/or excess industrial or mine wastewater (effluent).	Ministry of Agriculture, Fisheries, Water and Land Reform (MAFWLR): Department of Water Affairs (DWA)	6 – 9 months
Licence to construct a dam (or ponds)	Section 93 of the Water Resource Management Act and Section 47 (1) and (2) of the Water Resource Management Regulation	Construction of settlement, pollution control or stormwater ponds.	Ministry of Agriculture, Fisheries, Water and Land Reform (MAFWLR): Department of Water Affairs (DWA)	Unknown
Permit for the clearing of land	The Forest Act 2001	Removal of vegetation within 100 m of a water course, or removal of more than 15 ha of woody vegetation, or the removal of any protected plant species.	Ministry of Environment, Forestry and Tourism (MEFT): Department of Environmental Affairs and Forestry (DEAF)	2 – 3 weeks, valid for 3 months and can be renewed if required
Permit for the destruction of heritage objects and artefacts (heritage consent)	The Heritage Act 2004.	Interference with previously unidentified heritage artefacts located within the proposed mining licence footprint, or along proposed pipeline or powerline routes.	National Heritage Council (NHC)	3 – 4 months
Application for power connection	Electricity Act 2007	Connection of the Project to the national electricity grid.	Namibian Power Corporation (NamPower)	Submitted August 2025. Initial offer of supply received May 2026, negotiations of power supply agreement ongoing

Permit/Licence	Act/Regulation	Related Activities Requiring Permit	Relevant Authority	Timeframe
Consumer installation certificate for bulk fuel storage	Petroleum (Exploration and Production) Act 1991 and Petroleum Products Regulations 2000	Bulk fuel storage and dispensing.	Ministry of Industries, Mines and Energy (MIME): Directorate of Petroleum Affairs	2-3 months
Licence for explosives magazine	Minerals (Prospecting and Mining) Act, Mine Safety Regulations, Explosive Act 1956 and Explosive Regulations 2014	Operation of an explosives magazine to enable storage and handling of explosives.	Ministry of Industries, Mines and Energy (MIME): Directorate of Mines and Ministry of Home Affairs Immigration, Safety and Security (MHAISS): Namibian Police (NamPol)	Unknown
Permit for the storage and use of explosives, and the burning of packaging	Explosive Act 1956 and Explosive Regulations 2014	Explosive storage and blasting and disposal of explosives packaging by burning.	Ministry of Home Affairs, Immigration, Safety and Security (MHAISS): Namibian Police (NamPol)	Unknown
Amendment or adding of a route (change of route) (Road Diversion)	Road Act 2025	Diversion of the D3714 district gravel road.	Ministry of Works and Transport (MWT): Roads Authority (RA)	Unknown
High Value Mineral permit (HVM)	Section 105 of the Minerals (Prospecting and Mining) Act 1992	Export of minerals by air to international markets.	Ministry of Industries, Mines and Energy (MIME): Directorate of Mines	Unknown

14.2 Stakeholder Engagement

In accordance with Section 21 of the EMA, public participation is mandatory in the ESIA process to ensure transparency and stakeholder engagement. Stakeholder consultation for the Project commenced on 23 April 2024 through local newspaper notices, radio announcements, and site postings, inviting stakeholders to register for the Project. Key stakeholders consulted included local residents, businesses, community groups, traditional authorities (Daure Daman and !Oe≠Gân), conservancies, and government agencies (MIME, MEFT, Roads Authority, Erongo Regional Council).

Public meetings held in May and September 2024 allowed stakeholders to raise concerns. Key issues raised included employment, with a request to prioritise local youth for unskilled and semi-skilled roles; corporate social responsibility initiatives in education, healthcare, and infrastructure; water scarcity, noting the limited feasibility of new boreholes; and relocation and heritage protection, with the company committing to conduct a heritage study, comply with legislation, and consult affected families regarding burial sites and safe relocation.

Stakeholder engagement for the ESIA's for the off-site power and water infrastructure was carried out with public consultation meetings held between the 23rd and 25th of July 2026.

14.3 Environmental Baseline Studies

The environmental baseline for the Project was established using existing publications, prior research and targeted specialist investigations. As part of the ESIA, several specialist baseline and impact studies were carried out by subject-matter experts. The different specialist and baseline studies conducted for this ESIA are outlined in Table 14.2.

Table 14.2: Baseline and Specialist Studies Included in the Assessment

Study Area	Purpose
Terrestrial ecology	• Biodiversity, habitat and ecosystem services. • Identification of species of concern and sensitive ecological areas. • Impacts of mining construction and operations on habitats and biodiversity. • Recommend applicable, practicable mitigation and management measures for implementation.
Hydrology	• An assessment of possible flooding, design storm protection measures and diversion of natural drainage channels. • Recommend management systems for clean and dirty water.

Study Area	Purpose
Hydrogeology	• Investigate and assess feasible off-site bulk water sources for integration into the Project's water supply strategy. • Investigate and assess feasible on-site bulk water supply sources for the Project. • Assess the potential for contamination of aquifers from TSF and WRDs. • Provide a model to determine impacts of drawdown and plume mobility. • Assess the sustainability of groundwater supply for bulk water supply and alternative water supply schemes.
Air quality	• Provide emission standards and dust suppression requirements. • Assess prevailing wind directions and possible impacts of emissions on receptors (on-site and off-site).
Noise	• Identify possible noise sensitive receptors. • Model noise levels to understand the extent to receptors. • Recommend mitigation measures for noise impacts.
Visual and sense of place	• Assess the existing visual character and landscape quality of the area. • Identify sensitive visual receptors and evaluate visibility of proposed mine infrastructure. • Recommend mitigation measures to reduce visual impacts.
Soils and land use	• Assess existing land use and potential impacts on surrounding land users. • Assess the quality and quantity of material available and suitable for rehabilitation.
Traffic	• Review the potential traffic impacts and loading on routes associated with mining activities. • Estimate traffic volumes on district and mine routes during the construction and operational phase. • Assess the capacity of infrastructure and safety at the mine entrance. • Assess the need for an intersection upgrade at the mine entrance and provide a concept layout if necessary.
Heritage and culture	• To identify, assess and protect cultural, historical and archaeological resources that may be affected by the Project. • To comply with Namibian legislation and gain approval (consent) from the National Heritage Council.
Visual and tourism	• Assess the Project's visual impacts on the aesthetic character and quality of the receiving environment. • To determine how changes to the landscape may affect visual receptors (e.g. local communities and tourists).

Study Area	Purpose
Social and economic	• Assess the social and economic status quo of communities in the immediate Project area and other regional centres (Uis, Okombahe and Omaruru). • Identify communities located within or adjacent to the proposed mine footprint that may require strategic relocation prior to project development.
Geochemical sampling and analysis	• Undertake geochemical analysis of waste rock, tailings, and overburden to assess the mineralogical composition, acid mine drainage potential, and metal concentration of the leachate of waste rock and tailings.
Climate change	• To assess and understand climate-related risks on the Project. • To predict the Project's greenhouse gas emissions throughout the LOM. • Provide a basis for understanding the Project's contribution to climate change and identify opportunities for emission reduction.
Blast vibration impact	• Assess the impact of blasting on receptors in the area. • To determine the safe blast zone.

14.4 Environmental and Social Risks

The ESIA process is shown in Figure 14.2. All stages of the process are now complete with the final ESIA submitted to MEFT on 18 March 2026 and now under their review. Approval of the ESIA and grant of the ECC is anticipated to be in the second half of 2026. Project design in the DFS was informed and refined from the ESIA outcomes.

Figure 14.2: ESIA Process

14.4.1 Impacts Identified

The key impacts identified in the ESIA include:

- Heritage and cultural sites with potential for disturbance or destruction of archaeological artefacts and burial sites.

- Water resources impacts including groundwater reduction, contamination, and acid mine drainage.
- Environmental pollution through contamination of soil and emissions to air (including dust, $PM_{2.5}$, PM_{10}, NO_x and SO_x).
- Biodiversity and land use impacts including habitat destruction, vegetation clearing, and loss of agricultural land.
- Visual and landscape impacts which may include altered sense of place, tourism reduction, and impact from increased artificial lighting.
- Noise and vibration impacts from blasting, mining operations and vehicle movement.
- Climate change and emissions relating to the Project's greenhouse gas emissions and potential vulnerability to climate change.
- Socioeconomic impacts, which include positive impacts such as job creation, local economic development and growth of the gold mining sector but can also include negative impacts such as population influx and impacts on health and safety.
- Infrastructure and traffic impacts due to increased load on C36 road and the diversion of the D3714 around the mine.
- Rehabilitation and closure requirements including long-term land management and environmental restoration requirements.

14.5 Environmental and Social Management Plans

Table 14.3 provides a list of the environmental and social management plans for the Project, addressing potential impacts across the construction, operational and decommissioning phases. Each plan specifies mitigation and monitoring measures, performance indicators and the roles and responsibilities for implementation.

Table 14.3: Environmental and Social Impacts, Management and Monitoring Measures

Potential Impacts	Management and Mitigation Measures
Clearing of vegetation during construction.	• Identify and avoid protected flora species before disturbance, and where disturbance cannot be avoided, relocate them. • Prevent and discourage fires. • Limit clearing to the Project footprint and communicate no-go areas. • Prevent and discourage the collecting of firewood. • Incorporate large/old trees and native species into the overall landscaping. • Rehabilitate disturbed sites when possible.
Increased spread of alien invasive species.	• Conduct weed and seed inspections of vehicles entering the site. • Monitor the distribution of weeds throughout construction and operations and implement and eradication programme. • Limit clearing to the Project footprint. • Avoid planting invasive plant species as part of landscaping.
Loss of habitat during construction.	• Complete ecological walkdowns prior to clearing to identify and relocate fauna. • Phase vegetation clearance to allow fauna to evacuate the area. • Limit clearing to the Project footprint. • Avoid removal of trees with raptor nests. • No-go zones and protected species are to be communicated to all workers. • Avoid locating access routes through sensitive areas. • Ensure vehicles remain on demarcated roads.

Potential Impacts	Management and Mitigation Measures
Accidental injury and death of fauna.	• Relocate any species found on site. • Prevent domestic pets on site. • Use focused and wildlife friendly fencing. • Initiate a suitable waste removal system to avoid human-wildlife conflict. • Educate/inform contractors and staff on protected species to avoid illegal collection of such species. • Ensure appropriate speed limits are posted and adhered to.
Increased traffic and potential safety risks at intersections.	• Ensure road and intersection designs meet safety standards, including adequate sight distance and geometry. • Manage vegetation at intersection of the access road with public roads to maintain clear visibility. • Implement appropriate signage and speed controls near Project access points. • Apply traffic management and safety measures. • Coordinate with authorities to ensure maintenance of roads to manage increased traffic impacts. • Monitor impacts and communicate with stakeholders and the community.
Physical disturbance and destruction of soil structure and compaction.	• Minimise disturbance by optimising footprint and limiting excavation to necessary areas. • Manage soil stability by controlling erosion, stabilising slopes, and protecting exposed surfaces. • Restrict and manage machinery use outside operating areas to prevent soil compaction, particularly in wet conditions. • Implement effective stormwater and sediment control measures. • Train personnel in soil management and rehabilitation and restore disturbed areas where feasible.

Potential Impacts	Management and Mitigation Measures
Nutrient depletion, reduced microbial activity and organic matter loss in soils.	• Soils to be stored for longer than three years to be stockpiled in stockpiles less than 1.5 m in height. • Reuse topsoil as soon as possible to maintain fertility and microbial viability. • Topsoil stockpiles should be vegetated as soon as practicable to minimise wind and water erosion, preserve the soil resource, and maintain its biological functions. • Apply fertiliser to soil stockpiles where required to maintain soil fertility and nutrient levels. • Disturbed or excavated areas should be rehabilitated with soil material that was removed from the area, where practicable, shaped to free draining slopes and planted with native grass/shrub/tree species.
Soil contamination.	• Manage topsoil through proper stripping, low stockpiling, protection from contamination, and timely reuse. • Prevent contamination through proper storage, handling and disposal of fuels, chemicals and waste. • Maintain good site practices, including equipment upkeep, spill prevention, and effective waste management.
Reduction in flow and surface water availability and reduced groundwater recharge.	• Monitor groundwater and surface water through construction, operations and closure. • Modify altered surface water flow paths to allow upstream areas to route back into the natural downstream flow paths. • Discharge excess water of acceptable quality back into the surface water environment. • Adhere to abstraction limits and monitoring requirements specified in the groundwater abstraction licence issued by the regulator.
Surface water and groundwater contamination.	• Monitor groundwater and surface water for any water quality changes through construction, operations and closure. • Securely store chemicals and fuel in bunded areas with impervious surfaces, and conduct refuelling and major servicing in designated, standards-compliant areas. • Use drip trays for stationary vehicles and machinery during repairs and ensure spill kits are always available. • Regularly service vehicles and machinery to minimise leaks.
Increase in dust and gaseous emissions	• Regularly service vehicles according to manufacturers' specifications to operate efficiently as close to design emissions as possible. • Limit unnecessary travelling/idling of vehicles on untreated roads and applying dust suppressants on regularly travelled unpaved sections.

Potential Impacts	Management and Mitigation Measures
Increased noise and vibration disturbance	• Use low-noise equipment and keep all machinery well maintained. • Notify nearby communities in advance and restrict noisy works to daytime (no night-time construction work unless approved). • Train staff on noise control and proper equipment use. • Handle materials and operate equipment carefully (e.g., avoid dropping materials from height, minimise reversing and unnecessary idling). • Position and enclose equipment to reduce noise at sensitive locations; keep covers and doors closed. • Maintain roads to minimise vehicle-related noise and vibration. • Monitor and respond to noise complaints promptly, including measurement and corrective actions. • Control vibration and blasting impacts within specified limits through careful design and monitoring.
Disturbance and destruction of cultural heritage	• Follow the archaeological chance find procedure and all conditions of the approved heritage consent. • Establish and maintain buffer zones and no-go areas around sensitive heritage sites. • Ensure all workers are aware of the location and significance of heritage features. • Protect and clearly mark heritage sites (e.g., fencing, clearing vegetation) to prevent disturbance. • Maintain compliance through regular reporting and annual consent renewal. • If impacts to gravesites cannot be avoided, follow legal requirements for protection, or manage exhumation and relocation with specialist oversight and regulatory approval.
Loss of sense of place and visual disturbance.	• Limit clearing to essential areas and retain vegetation near sensitive boundaries. • Undertake dust suppression. • Undertake progressive rehabilitation.
Land acquisition and physical displacement.	• Engage with affected communities and community representatives during the Project planning phase. • Respect traditional decision-making processes and customary governance structures. • Implement and appropriate resettlement plan and livelihood restoration measures.

Potential Impacts	Management and Mitigation Measures
Worker health and safety	• Comply with legislated safety and health requirements and good industry practice. • Ensure availability and proper use of PPE. • Document injury statistics, corrective actions initiated and lessons learned to improve safety practices on site. • Deploy trained first aid teams and ensure that emergency response plans are in place, including access to medical facilities and clear reporting procedures. • Strict access control to be enforced at all entry and exit points of the mine.
Increased employment opportunities	• Maximise the employment of local communities over employees from elsewhere in Namibia and expats. • Set minimum targets for the inclusion of women and youth in both skilled and semi-skilled roles

14.6 Mine Closure

A national mine closure framework, developed by the Namibian Chamber of Environment, Chamber of Mines, MIME and international partners was incorporated into the draft revised Minerals Act. Consistent with this framework and international best practice, a conceptual mine rehabilitation and closure plan was outlined for the Project to support full regulatory compliance and alignment with recognised closure principles and objectives.

The key aspects of the mine closure plan, including post-closure monitoring include:

- Final closure of the TSF through capping with mine waste, reprofiling to final design landform, replacement of topsoil and revegetation.
- Plant and infrastructure removal and demolition along with remediation of the process plant and infrastructure sites.
- Mine closure including:
  - Construction of abandonment bund around the pits.
  - Reprofiling WRDs to final landforms, replacement of topsoil and revegetation.
  - Reprofiling of any unprocessed mineralised waste stockpiles to final landforms, replacement of topsoil and revegetation.
  - Removal and rehabilitation of mine roads and other surface infrastructure.
- Post-closure monitoring of revegetated areas, surface water, groundwater, and flora and fauna, with specific attention to pests and weeds for a period of up to five years.

14.6.1 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 was estimated at US$37.4m, excluding mine WRD reprofiling and progressive TSF capping but excluding the final stage of the TSF, which is carried out as part of the mining costs.

In addition, based on the objectives outlined above, an estimate was made for the post-closure monitoring that will be undertaken. This cost was estimated at US$2.7m and will take place over a period of up to five years post closure.

15 PROJECT IMPLEMENTATION

A project implementation plan was developed to define the proposed methodology that will be employed to successfully deliver the Project in accordance with the planned budget and schedule, while minimising the risk of adverse cost or schedule outcomes.

This implementation strategy was also used as a basis for the development of capital cost estimates for the Project.

15.1 Project Phases

The Project will be developed through a series of interrelated phases, some of which will overlap to optimise schedule efficiency. These phases include:

- Execution readiness, commencing upon completion of the DFS and continuing until project final investment decision (FID) and readiness to commence site activities. This phase will focus on establishment of the owners' team, engagement of key engineering consultants and completion of front-end engineering, long lead procurement activities and tendering of early-works construction contracts.
- Project delivery, commencing following FID and will involve the procurement, contracting and construction of all the facilities along with pre-production mining.
- Operational readiness, involving the staged build-up of the operational workforce, implementation of operational systems, developing operational and maintenance procedures, workforce training, and establishment of all functions required to support a successful transition into operations. This phase will proceed in parallel with the Project delivery.
- Completions and commissioning, incorporating a staged approach to construction completion, testing, commissioning, start-up and handover to operations. This phase will commence during the final stages of construction and culminate in the commencement of production.

15.2 Project Management Approach

The Project will be delivered through a combination of Wia's internal resources, an experienced project management consultant (PMC), EPCM contractors, and specialist external consultants. An integrated project management team (IPMT), comprising Wia personnel and the PMC, will be established to oversee project execution. This delivery model allows Wia to access proven project management systems, procedures, and expertise while maintaining a manageable size owner's team.

The Wia corporate and operational teams 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.

Under the leadership of Wia's Project Director, the IMPT will be responsible for the execution of all capital aspects of the Project. Key responsibilities will include:

- Overall project management and governance.
- Procurement and administration of engineering, supply, and construction contracts.
- Cost control, forecasting, and schedule management.
- Construction quality assurance and safety oversight.
- Engineering management, integration, and coordination.
- Standardisation and design governance.
- Design review and approval on behalf of Wia.
- Management of interfaces between contractors and work packages to ensure effective and efficient Project delivery

A project steering committee will be formed with the purpose of advising the Wia 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 Project Director and Project Manager will present to the project steering committee on safety, progress, expenditure and forecast, and risks.

The proposed Project organisational structure, including the owner's team and key delivery contractors, is shown in Figure 15.1.

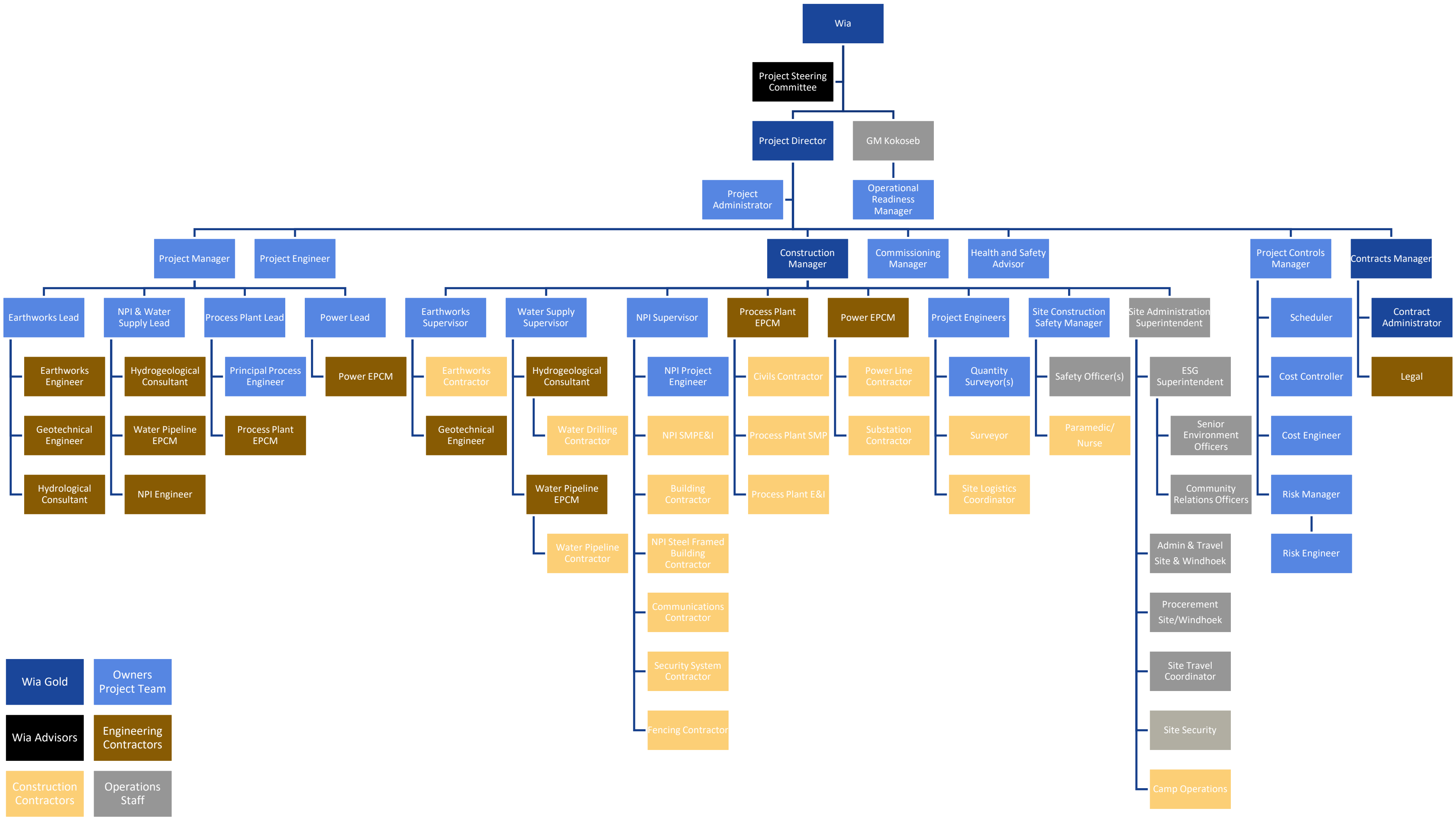


Figure 15.1: Project Owners Team Organisation Chart

15.3 Engineering Approach

The engineering services 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 Wia. There will also be a number of specialist engineering consultants who will provide specialist design input, with the procurement and construction management carried out by the IPMT in these areas. The EPCM contracts will include:

- Process plant EPCM, which includes the processing plant and process plant buildings.
- Power supply and distribution EPCM, responsible for the power supply to site and power distribution around site as well as the power supply for the bulk water supply.
- Bulk water supply EPCM, responsible for the development of any borefields (not bore drilling) and water delivery systems to 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.
- NPI engineering and procurement, which includes the administration and other support buildings, workshop, warehouse, accommodation village, site water management systems and Wia supplied mining infrastructure.
- Hydrogeological consultant, responsible for the design of any borefields, 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 mine designs, tendering of the mining contract.
- Mine geotechnical consultant, responsible for providing mining geotechnical advice to the mining engineers and operations.

This engineering delivery model combines specialist expertise with centralised project management and interface coordination through the IPMT, supporting the efficient execution of the Project while maintaining appropriate oversight of quality, cost, schedule, and risk.

15.4 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 were selected to be broken out of this strategy in an effort to identify works that can be allocated to local Namibian 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, preferably a local Namibian contractor, to carry out the construction works for the interfaces between construction contracts and installation of minor NPI supply equipment. Tendered and managed by the IPMT.
- Bulk water supply construction contractor covering the bore headworks, pumping stations and water pipelines. Tendered and managed by the bulk water supply EPCM.
- Building contractor, covering the design, fabrication and installation of the pre-fabricated buildings on site including the administration buildings and also the accommodation village. Tendered and managed by the IPMT.
- 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 IPMT.
- Power line contractor, supplying and installing the overhead powerlines to site and around site. Tendered and managed by the power station and distribution EPCM.
- HV substation contractor, for the construction of the HV substations including the modifications required at the Omburu substation, NamPower Kokoseb metering substation and site HV switchyard.
- 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.

15.5 Execution Readiness

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 six months is planned. The key workstreams in this period include:

- Formation of the IPMT through recruitment of personnel and engagement of the PMC.
- 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.
- Tendering and award of the EPCM and engineering contracts.
- Front-end engineering for process plant, including:
  - Finalisation of the plant layout, including a multi-disciplinary review.
  - Tendering of long lead equipment.
  - Completion of bulk earthworks design for the process plant to an approved for construction (AFC) status such that earthworks can commence immediately following FID.
  - Completion of sufficient design in various areas of the plant to allow design of civil concrete works for the leach tanks, thickeners, crushed ore stockpile and crusher such that civil construction (starting with the leach tank ring beams) can be commenced as soon as sufficient bulk earthworks is completed.
  - Preparation of the civil construction contract tender such that this can be tendered on FID allowing for award and mobilisation on completion of sufficient bulk earthworks.
- NPI front end engineering and design, including:
  - Review and finalisation of the village basis of design.
  - Design of the fly camp and construction camps, such that earthworks can be designed
  - Tendering of the village design and construction contract, adjudication and award.

- Power supply front-end engineering and design, including:
  - Power line route LIDAR survey and geotechnical survey.
  - Power line final routing.
  - Engineering design of the Omburu substation modifications, NamPower Kokoseb metering substation and site HV switchyard sufficient to specify long lead switchgear and transformers.
  - Tendering and procurement of long-lead electrical switchgear and transformers.
  - Preparation of power line construction and HV switchroom construction contracts ready to tender.
- Bulk water supply front-end engineering and design, including:
  - Detailed engineering for the construction water supply.
  - Tendering and procurement of the pumps, piping, valves and instruments for the construction water supply.
- Other front-end engineering and design, including:
  - Bulk earthworks design for the D3714 diversion, village earthworks, process plant area earthworks and site access road, such that the bulk earthworks can commence upon FID.
  - Preparation, tendering and adjudication of the bulk earthworks tender such that this can be executed on FID.
  - Mining engineering to allow preparation of the contract mining tender to be completed such that it can be issued for tender on FID.
- Site investigations and early works to support the final design, including:
  - Further process plant and TSF geotechnical site investigations based on the final layouts.
  - Geotechnical test pitting along linear infrastructure and identified sources of borrow.
  - Additional drilling in support of water abstraction permits for various water sources for the Project including the near mine bores, Okombahe WSS and OmAP borefield.
  - Site survey establishment.
- Placement of orders, and initial payments for, long-lead equipment including:
  - Process plant long-lead and major equipment (to secure certified vendor data).
  - Construction water supply equipment, tanks, electrical control panels and piping.
  - HV power transformers.
  - HV switchgear.
  - Water and wastewater treatment plants for the village to support construction.

- Electrical switchgear for the village to support construction.
- Construction support equipment including wastewater trailer, construction communications equipment, temporary facilities and water storage tanks.

- Construction readiness activities such as finalising the early works construction insurance, tendering of the site security contract, medical services contract and village operations contract.

15.6 Operational Readiness

Operational readiness and ramp-up of pre-production labour is key to successful start-up of the Project. On that basis a portion of the proposed operational workforce is engaged in advance of the commencement of operations. Figure 15.2 shows the build-up in workforce, excluding contractors, prior to the commencement of production.

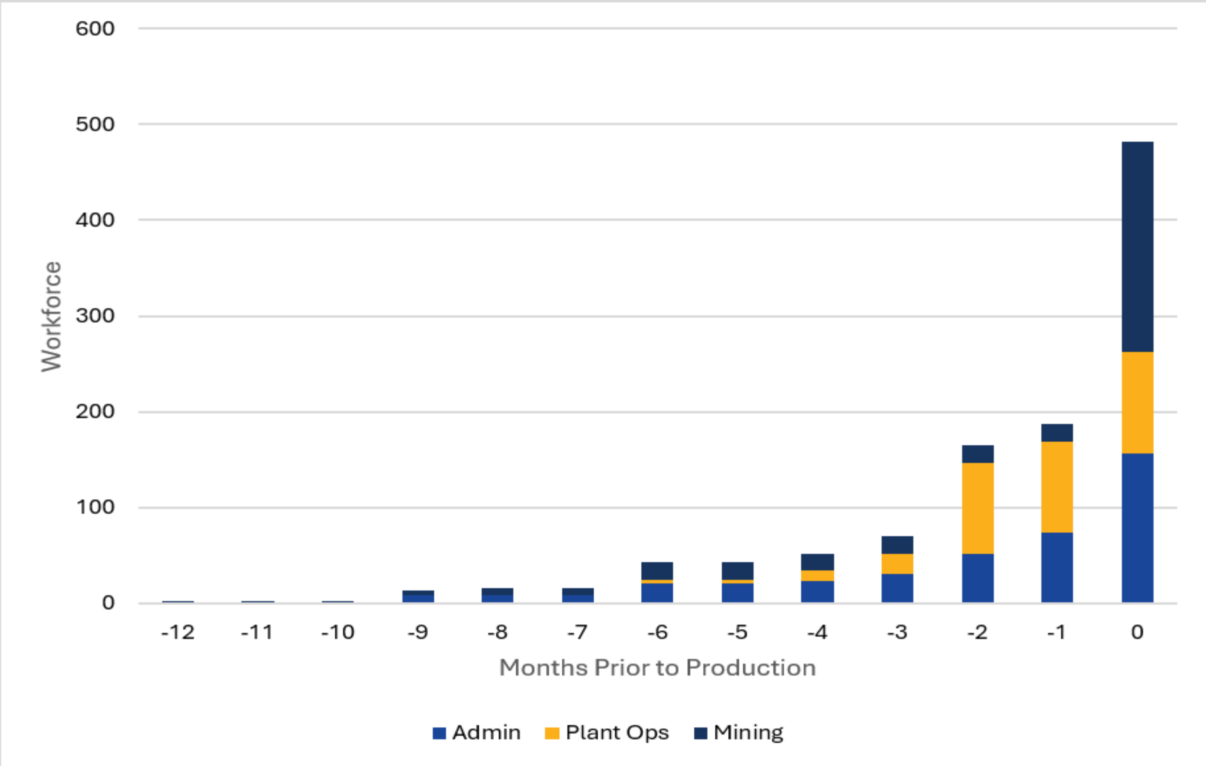


Figure 15.2: Pre-Production Workforce Ramp-up

In addition to the workforce build-up and the operational readiness tasks they will undertake, additional operational readiness activities that will be undertaken include:

- Mining engineering and mine geotechnical support for the early phases of mining.
- 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 operations specific aspects of the enterprise resources planning system will be undertaken by specialist consultants.

- Setting up the asset register, stockholding and warehouse management systems and bringing all the required spares and consumables into the warehouse ready for use.
- 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) with specialist contractors bought in to work with the permanent maintenance planners.
- Development of Project specific work practices and training modules across all areas of operations with specialist consultants bought in to assist the site process plant, administration and mining operations teams.
- General training courses for mine site operations for all personnel as required.

15.7 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 recover gold and achieving relatively continuous operation. Production of first product. Responsibility of the owner's 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.

15.8 Implementation Schedule

An integrated Project execution schedule was developed using Oracle Primavera P6 to support the planning and delivery of the Project. From the completion of the DFS, it is anticipated to take approximately six months for to reach FID and commence full Project development, with FID currently anticipated in October 2026. In parallel with this Wia will secure the Project's ECC and mining licence as well as completing relocation action plan.

The execution readiness activities detailed in Section 15.5 will commence in the third quarter of 2026 and be completed in parallel with the completion of funding.

The construction phase of the Project will commence at the beginning of 2027, with the breaking of first ground to commence site earthworks focussing on the completion of village earthworks, diversion of the D3714 and the process plant area, given site access is already available via the existing D3714.

Construction of infrastructure and services will proceed over the 22-month construction period with enabling infrastructure, such as the accommodation village, bulk earthworks and access roads, commencing immediately, followed by the commencement of the leach tanks in the process plant as soon as sufficient earthworks in this area are completed.

The TSF and associated infrastructure will be constructed following the wet season in 2028 to enable construction to follow on from other earthworks activities and complete construction with less risk of rain and delivering the facility ready for operations a minimum of two month prior to planned first ore milling.

Construction of the power supply grid connection and the OmAP borefield will commence following environmental approvals and granting of the servitudes in mid-2027 allowing power to be available in the third quarter of 2028.

Establishment of open pit operations will commence in the third quarter of 2028 to supply waste to construct the pit surface water management structures, ROM pad and prepare a stockpile of approximately three months of ore supply prior to commencement of milling.

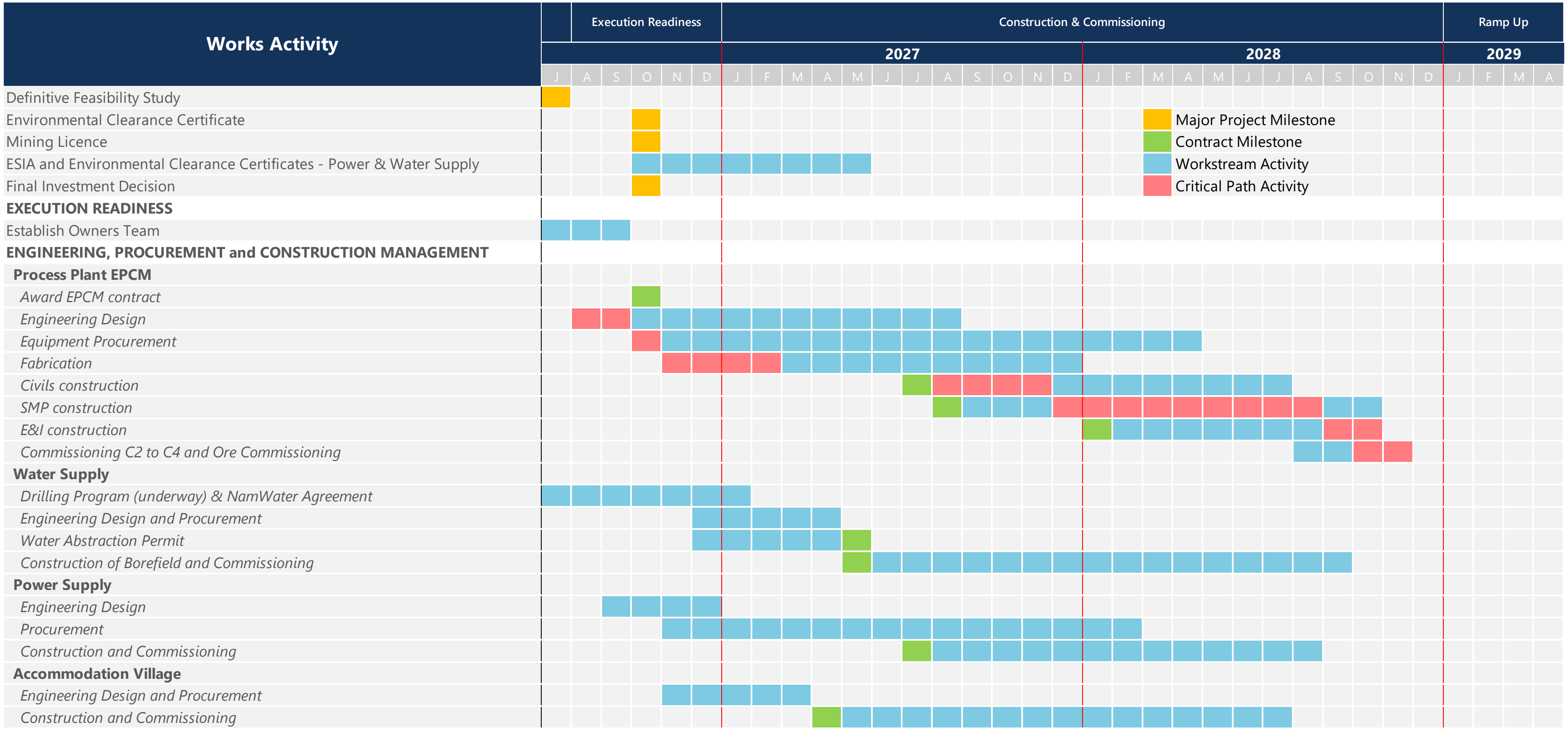
Commissioning will occur approximately three months prior to first production at the start of Year 1.

The critical path for the development of the Project from FID to commencement of production is the construction of the leach tanks in the process plant. Other areas of the Project have sufficient float, as shown in Table 15.1, to be considered to not be on the critical path.

Table 15.1: Schedule Float

Area	Days Float
Power Supply (required mid-point C2)	22
Water Supply (required C3)	13

Figure 15.3 provides a summary implementation schedule.



Works Activity
Execution Readiness
Construction & Commissioning
Ramp Up
2027
2028
2029
J A S O N D J F M A M J J A S O N D J F M A M J J A S O N D J F M A
Definitive Feasibility Study
Environmental Clearance Certificate
Mining Licence
ESIA and Environmental Clearance Certificates - Power & Water Supply
Final Investment Decision
EXECUTION READINESS
Establish Owners Team
ENGINEERING, PROCUREMENT and CONSTRUCTION MANAGEMENT
Process Plant EPCM
Award EPCM contract
Engineering Design
Equipment Procurement
Fabrication
Civils construction
SMP construction
E&I construction
Commissioning C2 to C4 and Ore Commissioning
Water Supply
Drilling Program (underway) & NamWater Agreement
Engineering Design and Procurement
Water Abstraction Permit
Construction of Borefield and Commissioning
Power Supply
Engineering Design
Procurement
Construction and Commissioning
Accommodation Village
Engineering Design and Procurement
Construction and Commissioning
Major Project Milestone
Contract Milestone
Workstream Activity
Critical Path Activity

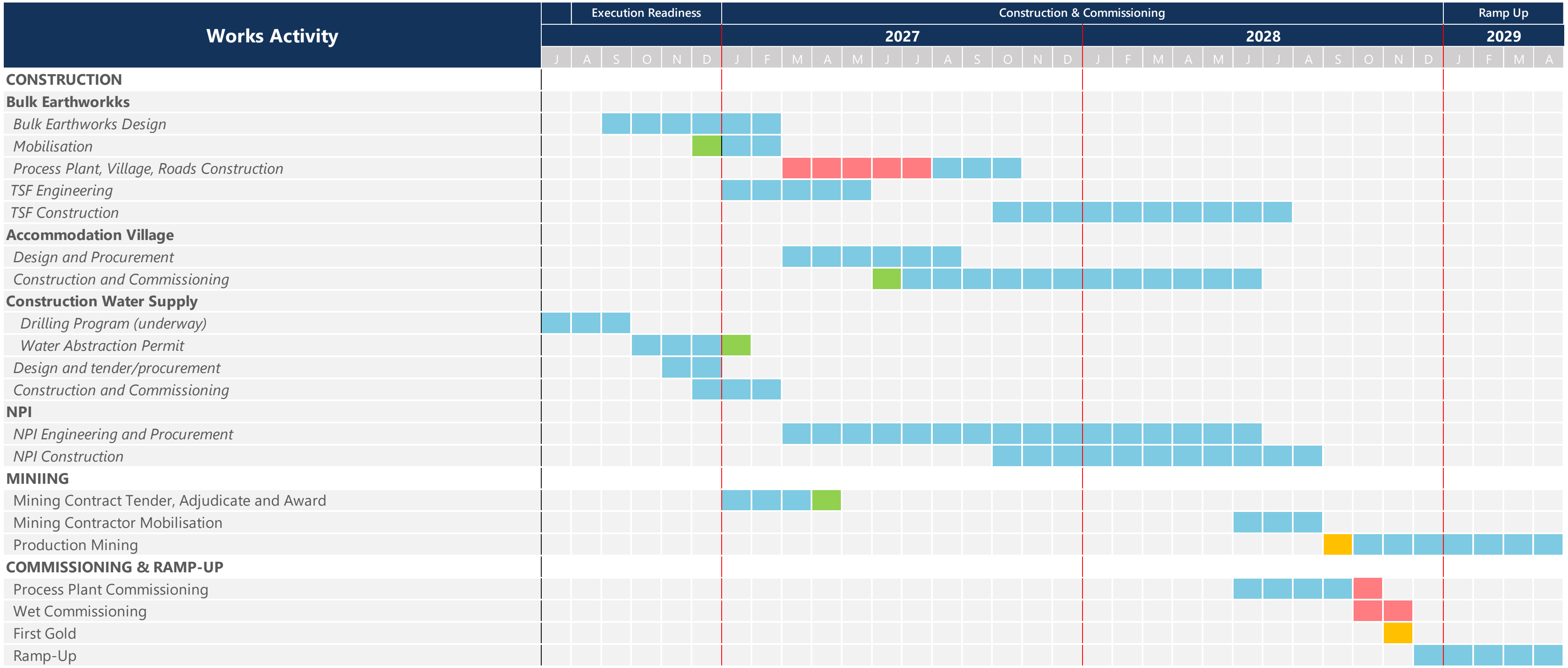


Figure 15.3: Summary Project Implementation Schedule

16 CAPITAL COST ESTIMATE

Capital costs for Project were developed based on the designs presented in this report and align with the implementation methodology and contracting strategy presented in Section 14.6. The estimate was generally derived from mechanical equipment list, specifications and material take offs (MTOs) produced for the DFS, and includes:

- Procurement and installation of permanent plant, equipment and materials.
- Construction labour.
- Construction plant and equipment.
- Contractors' preliminaries, overheads and profit.

The capital cost estimate was prepared to a standard generally consistent with an AACE Class 3 estimate, with an expected accuracy range of ±15%.

In addition, operational costs incurred prior to the commencement of ore processing, including pre-production mining, have been included in the capital cost estimate, however, these costs were estimated using the same methodologies and assumptions applied in the development of the corresponding operating cost estimate.

16.1 Basis of Estimate

The capital cost estimate was developed on the following basis:

- Earthworks quantities were derived from designs developed from 3D modelling of earthworks based on the requirements identified by specialist studies and pricing was developed based on detailed quotations from earthworks contractors active in Namibia.
- Bulk materials and equipment quantities were developed based on the engineering completed during this study including allowances for growth and wastage.
- Major equipment supply costs were based on budget quotations obtained during the study.
- Bulk materials supply and installation costs were developed from budget quotations received from fabricators and contractors based on preliminary scopes of work. These quotations were used to establish unit rates that were subsequently applied to the final quantities.
- Construction water costs were based on the methodology outlined for estimating water supply operating costs.
- Construction and installation costs were primarily based on submissions from construction contractors against preliminary scopes of work. These submissions were used to develop preliminary and general costs, installation productivity norms, and labour rates.
- The estimate was in accordance with the execution strategy outlined in Section 14.6 and assumes conventional field installation, with equipment and bulk materials

delivered to the site in the largest practical transportable modules utilising standard road transport.

- Project indirect costs were developed based on detailed estimates and align with the execution plan and project schedule.
- Owners' costs were included and comprise:
    - Owners' project management team and associated indirect costs.
    - Mining costs, including detailed mine design undertaken by consultants and owners' supplied mining equipment.
    - Environmental consulting services associated with construction approvals, compliance auditing and operational planning reviews.
    - First fills and spare parts inventory.
    - Project insurances.
    - Site security services during construction.
    - Site medical services during construction.
    - Environmental management and monitoring during construction.
    - Pre-production labour costs.
    - Operational readiness activities.
- Pre-production mining costs were estimated using the same methodology applied to the operating cost estimate and have been included in capital costs where incurred prior to the commencement of ore processing.
- A contingency allowance at the P_{80} confidence level was included based on the outcomes of the schedule risk assessment (SRA) and quantitative risk analysis (QRA).

16.2 Estimate Currency and Base Date

The capital cost estimate was prepared in United States dollars (US$), with a base date of the second quarter of 2026.

The exchange rates in Table 16.1 were applied in the development of the capital cost estimate. For financial evaluation purposes, all costs were maintained in their original currencies and carried through to the financial model, allowing foreign exchange movements and sensitivities to be appropriately modelled within the financial model.

Table 16.1: Foreign Exchange Rates

Currency	Exchange Rate	% of Capital Costs[1]
United States Dollar	1.000	30.4
Euro	0.847	0.4
Australian Dollar	1.452	1.0
South African Rand	17.30	25.4
Namibian Dollar	17.30	42.8

Notes:

1. Percentage of pre-production capital cost excluding pre-production mining

16.3 Contingency Estimate

In accordance with industry best practice a QRA and SRA were completed to evaluate the capital cost and schedule risk profiles for the Project.

The QRA and SRA assessed the potential variability in Project cost and schedule performance, together with associated risks, to determine the appropriate level of contingency for inclusion in the current cost estimate and project schedule at this stage of the Project development.

Contingency represents an allowance for known project costs that are expected to be incurred but cannot be defined with sufficient accuracy for detailed estimating purposes due to the lack of complete, accurate and detailed information, as well as the level of engineering performed to date. The inclusion of contingency is therefore necessary to establish the most likely Project cost outcome. Contingency does not provide for scope changes, project exclusions or changes to the proposed execution strategy.

A detailed Monte Carlo analysis was completed based on the outcomes of the QRA and SRA to quantify the potential cost variability. Contingency is included in the capital cost estimate at the P_{80} confidence level, equivalent to 9.9% of the base capital cost estimate. The contingency allowance excludes pre-production mining cost.

16.4 Capital Cost Estimate

As summarised in Table 16.2 the total pre-production costs from FID through to the first processing of ore, which will be shortly followed by first production of gold were estimated at US$474.8m, including a contingency of US$40.2m.

The estimate includes all the infrastructure, facilities, equipment, and services required to construct and commission the Project, together with pre-production mining, project management, first fills and spares, and owner's costs necessary to bring the Project into production.

Table 16.2: Capital Cost Estimate

Area	US$m[1]
Direct Costs	
Mining Infrastructure	1.0
Earthworks, Roads and Dams	16.1
Process Plant	163.0
Non-Process Infrastructure	26.1
Tailings Storage Facility	15.7
Water Supply	43.8
Power Supply and Distribution	36.7
Project Management	19.1
Construction Indirect Costs	40.0
Sub-Total	**361.7**
Owners Costs	
Owners Project Management	15.6
First Fills & Spares	7.2
Environmental & Social (Construction Phase)	1.1
Mobile Equipment	6.5
Project Office Costs	3.2
Insurances & Other Project Corporate Costs	3.0
Pre-Production Costs	9.5
Sub-Total	**46.1**
Mining	
Mobilisation & Site Establishment[1]	6.2
Pre-production Mining	20.7
Sub-Total	**26.8**
Total	
Contingency	40.2
Total	**474.8**

Notes:

1. Mining contractor mobilisation partially deferred over the initial years of mining with an allowance for contractor financing cost.
2. Totals may not compute due to rounding

The capital cost estimate excludes the following items:

- Sunk costs incurred prior the DFS completion.
- Escalation from the estimate base date through to Project completion.
- Government taxes, duties, levies and statutory charges.
- Working capital, sustaining capital and stay-in-business capital expenditures.
- Financing, including debt servicing, interest during construction, and other funding-related charges.

- Costs associated with schedule acceleration measures or schedule delays, including those arising from labour disputes, force majeure events, or other unforeseen disruptions.
- Exploration and resource delineation activities aimed at identifying additional Mineral Resources beyond the mining inventory considered in this study.

These exclusions are consistent with the basis of estimate adopted for the DFS capital cost evaluation and financial analysis.

16.5 Execution Readiness Costs

The execution readiness costs outlined in Table 16.3, which are fully included in the Project capital cost estimate, will be expended prior to FID and the commencement of construction from existing cash reserves. The activities planned for the execution readiness phase are outlined in Section 15.5.

Table 16.3: Execution Readiness Costs

Area	US$m
Owners team and tendering costs	2.0
Front-end engineering and design	1.4
Long lead and critical path procurement payments	6.5
Site Investigations	0.1
Pioneering site works	1.2
Total	**11.3**

16.6 Sustaining and Deferred Capital

Various sustaining capital expenditures will be incurred during the operations phase, including:

- TSF expansion and progressive closure comprising:
  - Stage 2 expansion and progressive closure of Stage 1 (Year 2).
  - Stage 3 expansion and progressive closure of Stage 2 (Year 7)
- 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 estimate, detailed in Table 16.4, is US$48.1m.

Table 16.4: Life of Mine Sustaining and Deferred Capital Estimate

Area	US$m
TSF Expansion	39.5
Progressive TSF closure (excluding mine waste placement)	2.1
General Sustaining Capital	6.5
Total	**48.1**

There is no deferred capital planned for the Project. Notably, while the construction of the final leach tanks in the process plant will occur following the commencement of milling, these costs were included in the pre-production capital estimate.

16.7 Closure Costs

Estimates were made of mine closure costs per the mine closure plan outlined in Section 14.6.1 and comprise the following:

- Final TSF closure, estimated at US$4.4m, noting that progressive rehabilitation activities are included in the sustaining capital estimate.
- Plant and infrastructure removal and demolition together with remediation of the process plant and infrastructure footprints, estimated at US$26.2m.
- Mine closure activities, including the construction of abandonment bunds rehabilitation of WRDs, roads, and other disturbed areas, estimated at US$2.8. Reprofiling of WRDs is included in the mining cost estimate and mining schedule.
- Closure planning, implementation management, and associated administration costs, estimated at US$1.9m.

A salvage value for the plant and infrastructure was included in the financial model at an amount equivalent to the estimated cost of plant and infrastructure closure, with the associated cash inflow assumed to occur upon completion of closure activities.

In addition, a program of post-closure monitoring over a period of up to five years following completion of closure, was estimated at US$2.7m.

17 OPERATING COSTS

The DFS operating cost estimate was developed from first principles by category and cost type. The methodology adopted for each major cost area is described in the subsequent sections.

The operating cost estimate was developed to a standard generally consistent with an AACE Class 3 estimate, with a target accuracy range of ±15%. The estimate is presented in US$ and is based on a pricing date of the second quarter of 2026. A summary of the life of mine operating costs is presented in Table 17.1.

Table 17.1: Life-of-Mine Operating Costs

Area	LOM Cost US$m	Unit Costs	Unit Cost US$/oz[1]
Open Pit Mining	1,438	US$3.10/t material mined	783
Processing	1,033	US$14.16/t ore milled	563
Tailings Handling	93	US$1.27/t milled	51
General and Administration	216	US$15.5m per annum	118
Transport & Refining	8	US$4.31/oz	4
C1 Cash Costs	**2,788**		**1,519**
Royalties	264	4 % of Revenue	144
Sustaining Capital & Closure Costs[2]	60	-	33
All-in Sustaining Costs	**3,112**		**1,696**

Notes:

1. Unit costs are per payable ounce of gold.
2. Sustaining capital & closure costs include salvage value of plant for calculation of AISC.

17.1 Estimate breakdown

The operating cost estimate was developed from first principles across the following cost categories:

- Mining.
- Labour.
- Power.
- Reagents.
- Consumables.
- Water.
- Mobile equipment.
- Maintenance.
- Transport and logistics.
- Laboratory.
- General and administration.

- Tailings handling.

The methodology and key assumptions applied in estimating each of these cost categories are described in the following sections.

17.2 Mining Costs

17.2.1 Basis of Estimate

Budget pricing was requested from five suitably qualified mining contractors for the provision of conventional open-pit truck-and-shovel mining services for the Project. Contractors were required to provide pricing to a budget level accuracy of approximately ±10% to ±15%.

The scope of work included:

- Mobilisation.
- Site establishment, including:
    - Clearing and topsoil stripping.
    - Haul-road establishment.
    - ROM pad bulk earthworks.
- Grade-control drilling.
- Production drill and blast.
- Ore and waste load and haul
- ROM feed and stockpile rehandle.
- Pit dewatering
- Dust suppression.
- Road maintenance
- Progressive rehabilitation, including:
    - Reshaping and ripping of WRDs.
    - Reclaim and spreading of topsoil.
- Demobilisation.

The scope also included the provision of mining infrastructure, as outlined in Section 12.5.

Pricing was requested on a commercial basis in US$, covering an initial contract term of seven years with a potential five-year extension. Unit rates excluded fuel costs, however, contractors were required to provided fuel-consumption rates, allowing the inclusion of fuel costs at a consistent Project rate.

Contractor submissions were normalised to ensure consistency in unit definitions, scope allocation, inclusions and exclusions, fuel treatment, fixed overheads, mobilisation and demobilisation, and mine-services infrastructure. Following this evaluation, a revised schedule of rates from the preferred contractor was applied to the final mining schedule and haulage model outputs to develop the mining cost estimate.

17.2.2 Open Pit Mining Costs

The open pit mining cost estimate comprises the following categories:

- Fixed unit rates applied to the physical mining 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 directly relating to mining quantities. These costs also include a number of ancillary mining activities, 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.
- Diesel consumption unit rates, which are fixed rates applied to the physical quantities over the LOM to calculate consumption with a fixed Project diesel price applied.

These costs are summarised in Table 17.2, which provides the open pit unit mining rates, Figure 17.1 showing the fixed contractor costs over the LOM. Diesel consumption rates for all mining activities are outlined in Table 17.3.

Table 17.2: Open Pit Mining Unit Rates

Cost	Units	Rate
Ore Re-handle to Crusher (FEL)	US$/bcm ore	0.95
Ore Stockpile Re-handle (Truck & FEL)	US$/bcm ore	2.51
Grade Control Drilling	US$/m	24.84
Grade Control Laboratory Costs[1]	US$/month	56,552
Additional Grade Control Assaying[1]	US$/assay	3.28
Drill & Blast – Waste	US$/t waste	0.66 – 0.93
Drill & Blast – Ore	US$/t ore	0.94 – 1.02
Load and Haul – Waste	US$/t waste	0.72 – 1.26
Load & Haul – Ore	US$/t ore	1.13 – 1.74

Notes:

1. Fixed monthly laboratory costs include assay of 6,080 grade control samples per month with additional samples charged at a rate per assay.

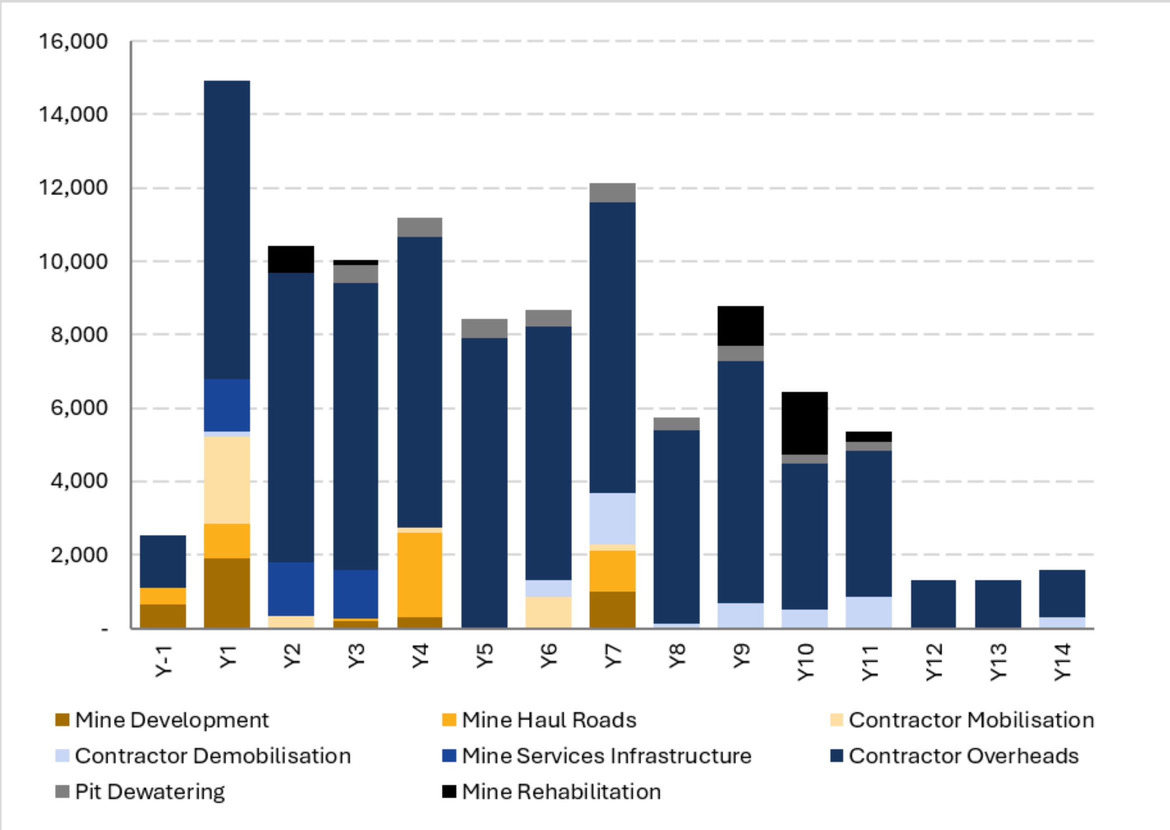


Figure 17.1: Fixed Open Pit Contractor Mining Costs over LOM

Table 17.3: Mining Diesel Consumption Rates

Cost	Units	Rate
Load & Haul		
Ore to ROM	L/bcm	0.81 – 3.02
Ore to Stockpile	L/bcm	0.63 – 2.64
Waste to WRD	L/bcm	0.44 – 1.76
Rehandle		
Rehandle with FEL	L/bcm	0.52
Rehandle with Truck & Loader	L/bcm	0.86
Drilling & Blasting		
Grade Control Drilling	L/lin.m	5.00
Drill Ore – Weathered/Fresh	L/bcm	2.71 / 2.88
Drill Waste – Weathered/Fresh	L/bcm	2.56 / 2.76
Blasting	L/bcm	0.04
Pre-Splits	L/lin.m	1.50
	L/hole	0.04
Mine Development & Rehabilitation		
Mine Development (LOM Total)	ML	1.79
Rehabilitation (LOM Total)	ML	1.56

The total mining diesel consumption over the life of mine was calculated at an average of 1.18 L/bcm for load and haul, or 1.51 L/bcm for all mining activities including load and haul, rehandle, drill and blast, mine development and rehabilitation.

17.3 Labour Costs

Organisational structures, employee roles, manning levels, Paterson grades and remuneration data were developed as part of the DFS and formed the basis of the labour operating cost estimate. Over the life of mine the labour is considered fixed in all areas, other than the owners mining team, which is varied based on the level of mining activity being undertaken with the personnel levels scaled based on the number of active digging units required for the mine production.

The gross cost to company comprises taxable gross annual salary and short-term incentive plan. Total cost to company is the gross cost, plus insurances, medical compliance checks, social security, allowances, and healthcare. Accommodation costs, transportation and flight costs are captured in the general and administration costs.

Expatriate employee costs were based on benchmark data from similar mining projects representing fixed annual net cash (i.e. gross cost to company), which includes guaranteed allowances but no additional benefits. The 50^{th} percentile rates were used for expatriate staff.

Namibian personnel annual gross salaries (i.e. total cost to company), including standard estimates for medical, pension and social security were sourced from a human resources consultant in Namibia and benchmarked against salaries at current operating mines in Namibia. Namibian tax tables were used to estimate the base taxable salary such that on costs and tax could be re-estimated to include bonuses, overtime and an allowance for unavailable labour that is specific to the Project's selected rosters. Overtime is calculated based on the number of hours exceeding the Namibian standard of 195 hours per month.

The following categories of employees were defined for the Project:

- Expatriate employees based outside of Africa working six weeks on three weeks off rosters.
- Regional expatriate employees based inside Southern Africa working six weeks on three weeks off rosters.
- Namibian drive-in drive-out (DIDO) employees based within Namibia working four weeks on two weeks off rosters when working day shift only roles.
- Continuous shift workers will work 14 days on seven days off rosters with a 12% shift multiplier applied to the total cost to company. Three panels are proposed for continuous shift rosters and two for day shift backfilled positions.
- Local employees based in the nearby towns of Okombahe, Uis or Omaruru will be employed in suitable day shift only, or continuous day shift roles, working:
    - For day shift only roles, ten hours per day, five days on two days off.

- For roles that require continuous day shift coverage, ten hours per day, four days on three days off with each week to include one cross-over day when both panels will be on site at the same time.

The total labour costs, excluding all contract labour, is provided in Table 17.4 based on operations with both open pit and underground mining underway.

Table 17.4: Labour Costs

Area	Units	Cost
Mining[1]	US$/a	2.9
Processing	US$/a	5.4
Administration and Environmental	US$/a	5.5

Notes:
1. Mining owners labour varies over LOM based on the level of mining activity determined by the number of digging units in use. The cost per annum presented represents the cost during the steady state period of mining operations.

17.4 Power

17.4.1 Power Demand

The fixed power demand for the operation was determined from the electrical load list, expected utilisation for electrical equipment items and plant area availability assumptions.

Weathered ore will require lower crushing and grinding energy than fresh ore. Power modelling indicates that the total processing power demand for weathered ore is approximately 20% less than for fresh ore.

Power consumption associated with the SAG and ball mill was excluded from the fixed power consumption estimate and was instead calculated separately based on energy requirements. Grinding power was estimated by applying the relevant specific energy consumption to the tonnes of each lithology processed during each period, based on achieving a grind product size of P_{80} 63 µm on average hardness ore.

The specific grinding energy assumptions adopted for each lithology are presented below:

- Fresh ore: 31.6 kWh/t.
- Weathered ore: 25.3 kWh/t.

These specific energy values were applied to the LOM processing schedule to estimate variable comminution power consumption.

17.4.2 Power Cost

Power supply cost from NamPower comprise both fixed and variable tariff components. With the exception of the NamPower network extension charge, which relates to the capital cost of the shallow connection infrastructure, these tariffs are reviewed and approved annually by the Namibian Electricity Control Board (ECB) of Namibia. The DFS operating cost estimate is based on the published 2025/26 electricity tariffs applicable at the time of the study. The ECB also responsible for reviewing and approving tariffs charged by regional electricity distributors, including Erongo RED, which services the Project region.

The tariffs used in the DFS are outlined in Table 17.5.

Table 17.5: Fixed and Variable Power Charges

Charge	Units	Rate
Variable Charges		
Charge per Unit	NAD/kWh	1.1526
Reliability Charge	NAD/kWh	0.1429
Losses Charge	NAD/kWh	0.1582
Levies		
ECB Levy	NAD/kWh	0.0212
National Energy Fund Levy	NAD/kWh	0.0160
Fixed Charges		
Maximum Demand	NAD/kW per month	275.21
Monthly Account Fee	NAD/account per month	28,844.17
NamPower Extension Charge	NAD/month	1,561,600

These charges, which are applied to the calculated power draw over the LOM, result in an average cost of power over the LOM of US$0.115 per kWh.

17.5 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. Reagent costs were based on quoted prices delivered Walvis Bay with transport costs estimated and added as appropriate.

The total reagent requirements for the processing plant are summarised in Table 17.6.

Table 17.6: Reagent Summary

Reagent	Packaging	US$/t	Consumption Rate	Annual Consumption	Source
Cyanide (Leach)	Briquettes, bulk box	2,675	0.62 kg/t	3,269	Escalated from historical pricing, CFR Walvis
Cyanide (Elution)			171 kg/strip	49	
Cyanide (Strip)			78.00 kg/t conc.	133	
Quicklime	Bulk tanker, local	359	0.91 kg/t	4,772	CFR Supplier South Africa
Activated Carbon	Bulka bag	3,503	30 g/t	158	Escalated from historical pricing, CFR Walvis
Copper Sulphate	Bulka bag	3,620	73 kg/h	586	CFR Walvis Bay quote
Sodium Metabisulfite	Bulka bag	840	234 kg/h	1,873	CFR Walvis Bay quote
Flocculant	Bulka bag	2,470	30 g/t	158	CFR Walvis Bay quote
Hydrochloric acid 32%	IBC	375	2,005 kg/strip	571	CFR Walvis Bay quote
Sodium hydroxide	Pearl	870	215 g/t	1,127	CFR Walvis Bay quote
Sulphamic acid	IBC	1,270	25 kg/week	1.30	CFR Walvis Bay quote
Antiscalant	IBC	2,010	681 kg/yr	0.68	CFR Walvis Bay quote
Borax	Bag, 25 kg	2,170	700 kg/yr	0.70	CFR Walvis Bay quote
Silica	Bag, 25 kg	770	356 kg/yr	0.36	CFR Walvis Bay quote
Soda ash	Bag, 25 kg	970	356 kg/yr	0.36	CFR Walvis Bay quote
Sodium nitrate	Bag, 25 kg	2,070	356 kg/yr	0.36	CFR Walvis Bay quote
Process Diesel (kL)	Bulk tanker	1.25/L		2,308	Project Pricing

17.6 Consumables

Specialised consumables for mechanical equipment were included as presented in Table 17.7. 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 from comminution modelling based on average ore characteristics with other usage rates provided by equipment vendors of from typical similar operations.

Table 17.7: Consumables

Consumable	$ / Unit	Unit	Annual Consumption	Basis of Estimate
SAG mill media	1,050	t	2,668	CFR Walvis Bay quote
Ball mill media	1,050	t	4,045	CFR Walvis Bay quote
Primary crusher mantle	59,957	set	2.1	Supplier quote, +15% freight
Primary crusher concave	86,600	set	1.4	Supplier quote, +15% freight
SAG mill liner & lifter set	2,185,961	set	1.6	Supplier quote, +15% freight
Ball mill liner & lifter set	695,164	set	1.6	Supplier quote, +15% freight
Pebble crusher liner set	6,760	set	5.3	Supplier quote, +15% freight
Tailings filter cloth set	219,269	set	3	Database, inclusive of delivery
Stainless steel wool	1,057	kg	110	Database, inclusive of deliver
Crucibles	4,514	each	9	Database, inclusive of delivery

17.7 Water

Water supply costs were estimated to include the following aspects for each of the water supply options:

- **Maintenance costs**: Included at various percentages of the capital cost using 0.5% of civils, 1% of pipelines, 4% of mechanical and electrical and 4% of power supply costs.
- **Operations labour**: Included based on each scheme having a dedicated workforce consisting of a superintendent and two technicians, one support vehicle and an allowance for fuel consumption. A 15% contingency was included on the operations labour to account for potential increases based on a NamWater operations approach.
- **Electricity costs**: Electricity consumption were estimated based on the design pumping rates and heads for each drive, pump efficiencies and power unit costs from Erongo RED.
- **NamWater mark-up**: Included at 15% on top of operational costs, based on discussions with NamWater.

For calculating the supply cost the operations, maintenance and labour costs were assumed to be fixed costs applicable regardless of water consumption. The electricity costs were assumed to be variable based on the total water usage from the scheme.

The water supply costs used in the estimate are shown in Table 17.8.

Table 17.8: Water Supply Costs

Aspect	Units	Cost
Capacity	kL/a	1,400,000
Fixed Costs		
Maintenance	NAD/a	12,034,751
Labour	NAD/a	1,867,500
NamWater Mark-up	NAD/a	2,085,338
Total Fixed Cost	**NAD/a**	**15,987,588**
Variable Costs		
Electricity	NAD/kL	20.3811
NamWater Mark-up	NAD/kL	3.0572
Total Variable Cost	**NAD/kL**	**23.4383**

17.8 Mobile Equipment

The maintenance and consumables required for mobile equipment were estimated and considered a fixed cost. Maintenance for this equipment are included within the maintenance costs, while diesel is included 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.

17.9 Maintenance

Maintenance costs are considered fixed annual costs and include the cost of spare parts (other than those included as consumables), maintenance consumables and maintenance contracts to maintain the processing plant 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 by 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.

17.10 Transport and Logistics

Transport charges from Walvis Bay to site were constructed to include the following for a twenty-foot container equivalent (TFE) transporting a 25-tonne payload:

- Cargo dues
- Terminal handling charges (Walvis Bay)
- Delivery release order fee
- Service fees
- Merchant haulage fees
- Container terminal operator fee
- Navis fee
- Container cleaning fee
- Port security charge
- Turn in fee
- Hazardous goods surcharges, if appropriate
- Transport from port to site via road, including the return of empty containers.

This was estimated at NAD 44,639.75 per TFE.

Freight costs for consumables (crusher jaws and teeth, mill liners, filter cloth sets, etc) were estimated at 15% of the supply cost as final container volumes are not known.

Where applicable, the estimated transport charge was applied to the goods supplied CFR to Walvis Bay.

17.11 Laboratory

Quotations were received to set up and provide contract laboratory services for the site. Laboratory costs and the estimated number of samples to be processed by business area, are summarised in Table 17.9.

The laboratory cost allows for the supply and maintenance of the laboratory equipment (amortised over the operating term), mobilisation and all ongoing costs (laboratory labour, equipment and consumables) comprising a fixed monthly cost and a variable cost related to the number of samples being processed. The laboratory building will be provided for the contractor to fit out. The building cost is included in the capital cost estimate.

Table 17.9: Laboratory Cost Summary

Item			US$/mo	US$m/a
Fixed Fee			99,060	1.19
Variable Fee (Based on the following samples)		Internal	29,321	0.35
Total			**128,381**	**1.54**
Source			**Sample Number**	
Mine Exploration & Grade Control		Internal	5,171	
Plant Solids (Assay, Moisture, Sizing)	Dry	Internal	1,369	
	Slurry	Internal	183	
Plant Solutions (Assay)		Internal	1,643	
Plant Carbon (Assay)		Internal	548	
Bullion (Au & Ag)		Internal	43	
Metallurgical Testwork Laboratory		Internal	78	
Environmental (CN, WAD, Total CN)		External	30	
Environmental (pH, TSS, TDS, E Coli, Assay)		External	4	

17.12 General and Administration

General and administration costs are considered fixed operating costs. The composition of the general and administration cost estimate, summarised in Table 17.10 is outlined below:

- Site office expenses, includes costs for communications and communication maintenance, office equipment and supplies, computer supplies and general office software licences.
- Insurance expenses, includes industrial special risks, third party liability, motor vehicles and operational shipping. Labour associated insurances (medical, death and disability) and workers liability insurances are included and reported in the labour total cost to company.
- Financial expenses, includes banking charges, legal fees, auditing costs and accounting consultants and bullion selling. Bullion transport, vaulting, refining and royalties are excluded, as they are included separately in the financial model.
- Government charges, includes surficial fees associated with permits and environmental inspection fees. All charges associated with revenue, including royalties and refining charges, are covered directly in the financial model.
- Personnel expenses, includes first aid and medical costs, safety supplies, business travel and accommodation for meetings/training, flights for expatriate employees, expatriate employee recruiting/relocation costs, training, recreational and local facilities costs, professional memberships and subscriptions, and entertainment allowances. The allowances for expatriate recruiting/relocation and other per person costs cater for all personnel (mining as well as administration and processing staff).

- Contracts, covers contract costs for personnel transport to site, camp management, catering and cleaning, site security, medical services, environmental compliance testing, OH&S and other consultants. The camp, catering and cleaning contract cost includes for all Wia staff personnel (mining as well as administration and processing).
- Road maintenance, covers maintenance of the site road network.
- Other expenses, includes general miscellaneous expenses provisions such as hired services.
- Consultants, includes environmental, mining, geotechnical, metallurgical, TSF Engineer of Record and equipment vendor support.
- Consumables, includes environmental, survey, geology and geotechnical consumables.

Table 17.10: Site General and Administration Cost Summary (excluding labour)

Area	Annual Cost US$m/a
Site Office	0.72
Insurances	1.60
Financial	0.09
Government Charges	0.01
Personnel Costs	1.36
Subtotal	**3.77**
Contracts	
Worker Transport (Bus) Contract	0.23
Camp, Catering and Cleaning Contract	2.85
Medical Services Contract	0.54
Road Maintenance Contract	0.37
Other Contracts & Services	0.56
Subtotal Contracts	**4.55**
Consultants	0.66
Consumables	0.14
Total G & A	**9.12**

The general and administration estimate is considered to represent the full range of support services and overhead costs required to sustain safe and efficient Project operations throughout the LOM.

17.13 Tailings Rehandle

Tailings rehandle will be carried out by a suitably qualified Namibian contractor. A request for budget proposal was issued to four potential contractors for the tailings rehandle services, including:

- Supply and maintenance of mobile fleet including FEL, trucks, grader, bulldozer, rollers and water cart.

- Pickup of tailings from bunkers under the filters, loading into trucks, transport to the TSF, placement, spreading and compacting, where required, in line with the TSF design.
- Dust management across the TSF and TSF haul roads.
- Management of the services on site.

The submissions from contractors were evaluated with the costs outlined in Table 17.11 selected from one contractor submission. Based on these rates, the average cost over LOM is US$1.27 per tonne of tailings.

Table 17.11: Tailings Rehandle Costs

Cost	Units	Rate	Annual Cost NAD m/a
Annual Throughput	t/a	5,250,000	
Mobilisation Cost (once off)	NAD	1,170,000	
Demobilisation Cost (once off)	NAD	705,000	
Fixed Costs			
Management and Supervision	NAD/month	940,761	11.29
Support Services	NAD/Month	190,322	2.28
Total Fixed Cost	**NAD/Month**	**1,131,083**	**13.57**
Variable Costs			
Compacted (50% of Tails)	NAD/t	20.68	54.29
Un-compacted (50% of Tails)	NAD/t	18.18	47.56
Total Variable Cost	**NAD/t**	**19.40**	**101.85**
Total Cost	**NAD/t**	**21.99**	**115.42**

18 FINANCIAL ANALYSIS

The financial evaluation of the Project was undertaken using a discounted cash flow (DCF) analysis in US$ (real Q2 2026 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 the Company.

The Project economics were assessed over the 14-year operating life, with the key valuation metrics comprising the net present value (NPV) and internal rate of return (IRR).

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 was calculated at the Project commitment date, or FID, currently anticipated to be 31 October 2026.
- Project funding entirely through equity.

While the base case assumes 100% equity funding for evaluation purposes, it is anticipated that the Project may ultimately incorporate a combination of debt and equity financing or other funding instruments.

18.1 Key Assumptions

Physical assumptions are based on the mine and production schedule presented Section 8.6.2. Capital, sustaining capital, operating and closure costs are as per Section 16 and Section 17 Other key financial assumptions are set out in Table 18.1.

Table 18.1: Key Financial Model Assumptions

	Units	Value
Gold Price	US$/oz	3,600
Discount Rate	%	5.0
Government Royalty	% of Revenue	3.0
Gross Revenue Export Tax	% of Revenue	1.0
Selling Costs	US$/oz	4.31
Corporate Tax Rate	%	37
Sunk Costs for Depreciation	US$m	30.0
Working Capital Assumptions		
Payment on Shipment of Gold	%	100
Capital Expenses	month	1
Operating Expenses (excl. Labour)	month	1
Labour and Royalties	month	0
Tax	month	1
Diesel Fuel Price	US$/L	1.25

The gold price of US$3,600/oz is based on the most recently published median long term real consensus pricing from more than 30 institutions which is notably lower than the July 2026 average spot price of approximately US$4,075/oz.

Prior spending on exploration and studies are assumed to be tax deductible at this stage and depreciation is calculated using a unit of production basis.

Foreign exchange rates in the DFS and the financial model are as per those provided in Table 16.1.

18.2 Key Financial Outcomes

The financial analysis was based on the mining inventory defined for the Kokoseb open pit operation. Gold production is front end loaded, averaging approximately 161 koz per annum during the first five years and 150 koz per annum over the first ten years. LOM production averages 131 koz annually, with an average AISC of US$1,696/oz.

The Project demonstrates robust financial performance at the consensus gold price assumption of US$3,600/oz, generating a post-tax $NPV_{5\%}$ of US$1,214m, an IRR of 41% and a payback period of 21 months.

Financial outcomes improve materially under the spot gold price scenario of US$4,075/oz with the Project delivering a post-tax $NPV_{5\%}$ of US$1,577m, an IRR of 49% and a payback period of 17 months.

A summary of the key Project outcomes is provided in Table 18.2.

Table 18.2: Key Project Outcomes

	Units	Base Case US$3,600/oz	Spot Price US$4,075/oz
Production			
Mine Life	years	14 years	
Total Gold Production	koz	1,836	
Average Gold Production			
Years 1 to 5	koz/a	161	
Years 1 to 10	koz/a	150	
Life of Mine	koz/a	131	
Proportion Inferred (contained gold)			
Years 1 to 5	%	Nil	
Life of Mine	%	3.8	
Costs			
Capital Costs	US$m	407.8	
Mining Mobilisation & Pre-production	US$m	26.8	
Contingency	US$m	40.2	
Sustaining Capital Costs	US$m	48.1	
Mine Closure Costs (excluding salvage)	US$m	37.9	
C1 Cash Costs	US$/oz	1,519	
All-in Sustaining Costs (AISC)	US$/oz	1,696	1,715
Financial			
Pre-Tax $NPV_{5\%}$	US$m	1,923	2,493
Pre-Tax IRR	%	51	61
Post-Tax $NPV_{5\%}$	US$m	1,214	1,577
Post-Tax IRR	%	41	49
Post-Tax Payback Period	Months	21	17

It should be noted that the mining inventory included in the financial analysis includes Inferred Mineral Resources and there is a low level of geological confidence associated with these Inferred Mineral Resources included in the mining inventory 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.

The financial outcomes presented are therefore subject to the risks associated with the inclusion of Inferred Mineral Resources and should be considered in the context of the level of confidence applicable to those resources.

Annual production (Figure 18.1), AISC (Figure 18.2) and project cashflows (Figure 18.3) are presented graphically below.

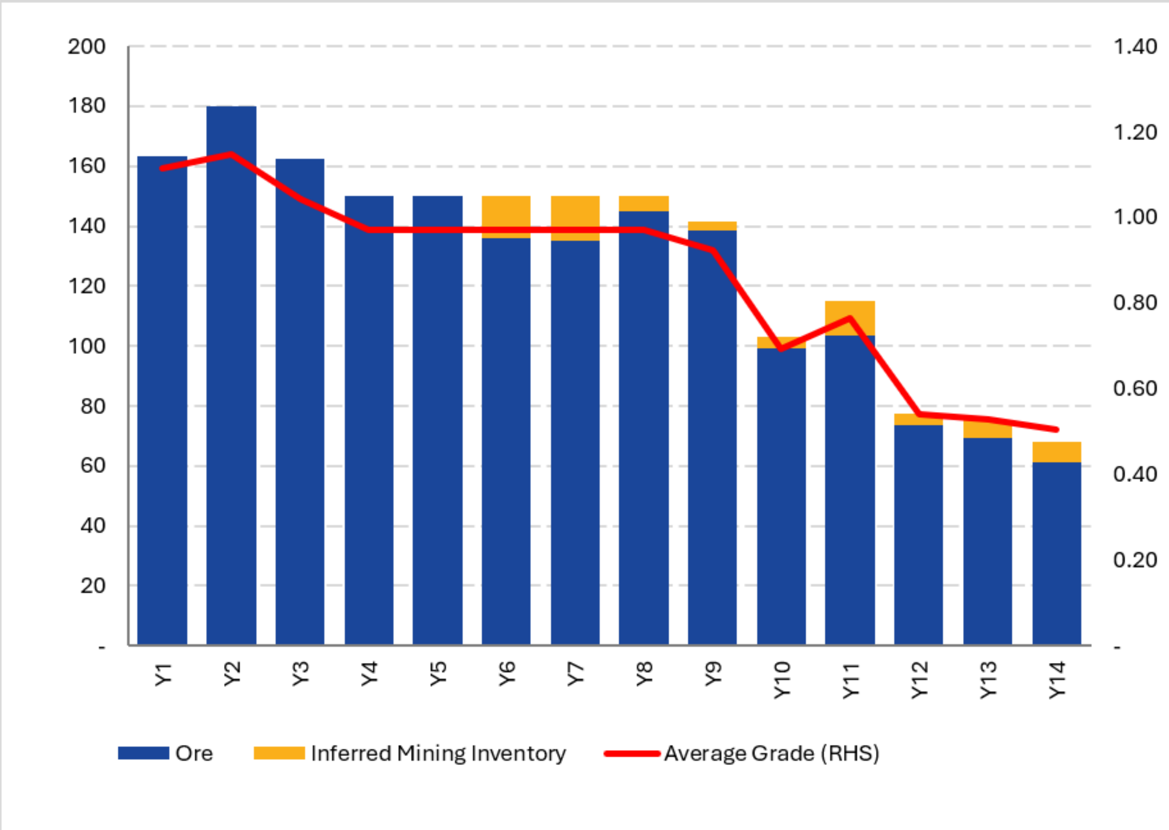


Figure 18.1: Gold Production and Grade

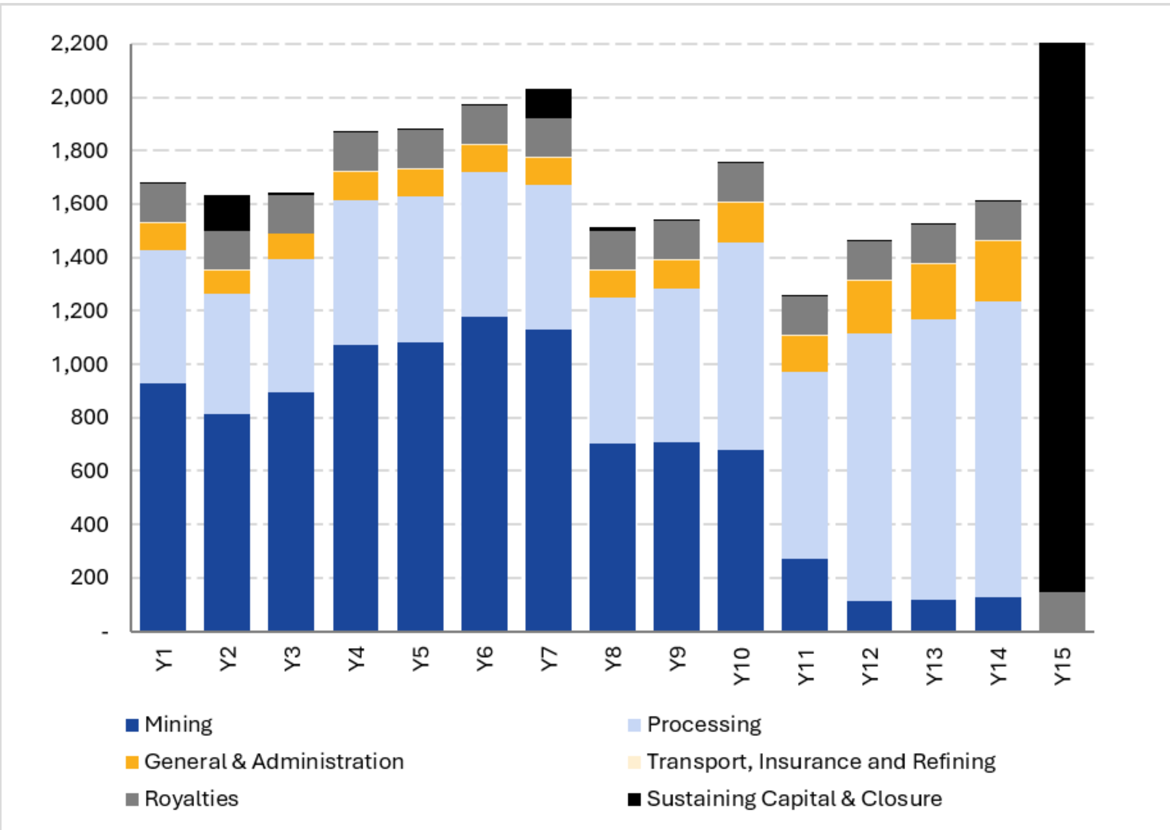


Figure 18.2: All-in Sustaining Cost per Oz

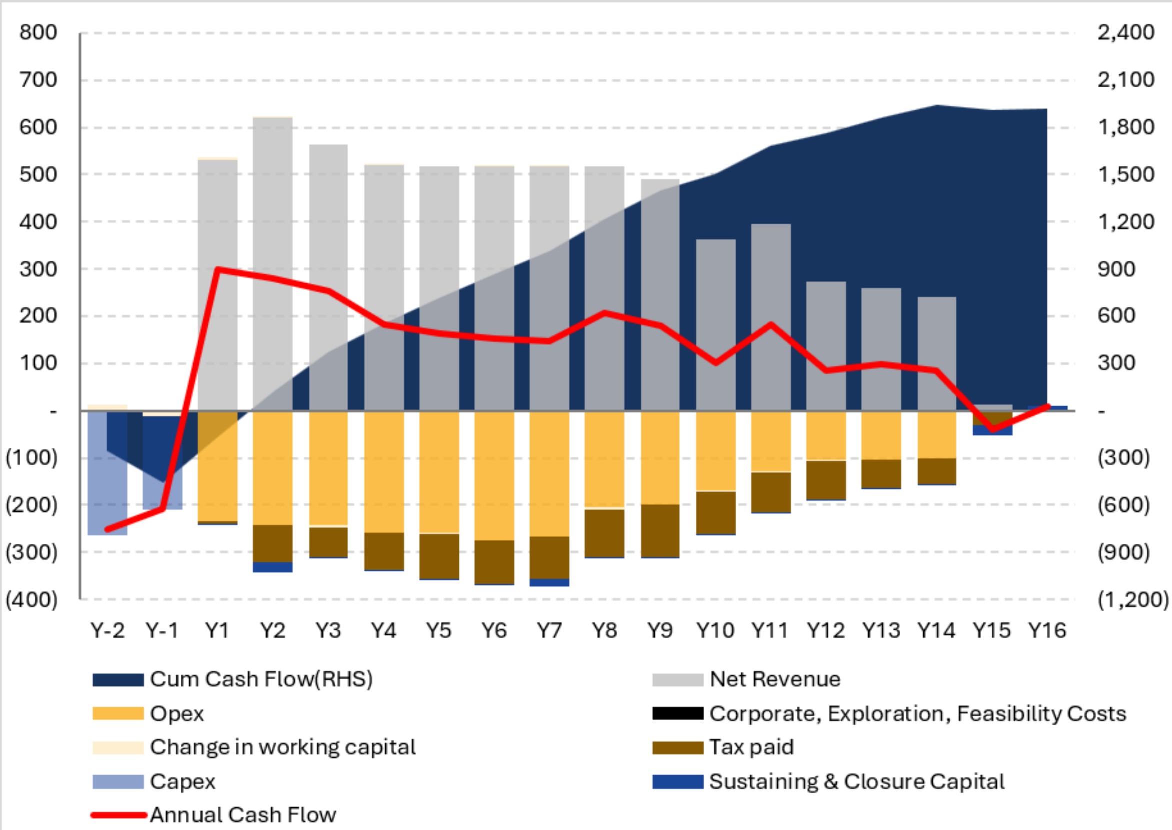


Figure 18.3: Project Post Tax Cash Flow

18.3 Sensitivity Analysis

18.3.1 General Sensitivity Analysis

The sensitivity of the post-tax $NPV_{5\%}$ to changes in key assumptions are shown in Figure 18.4.

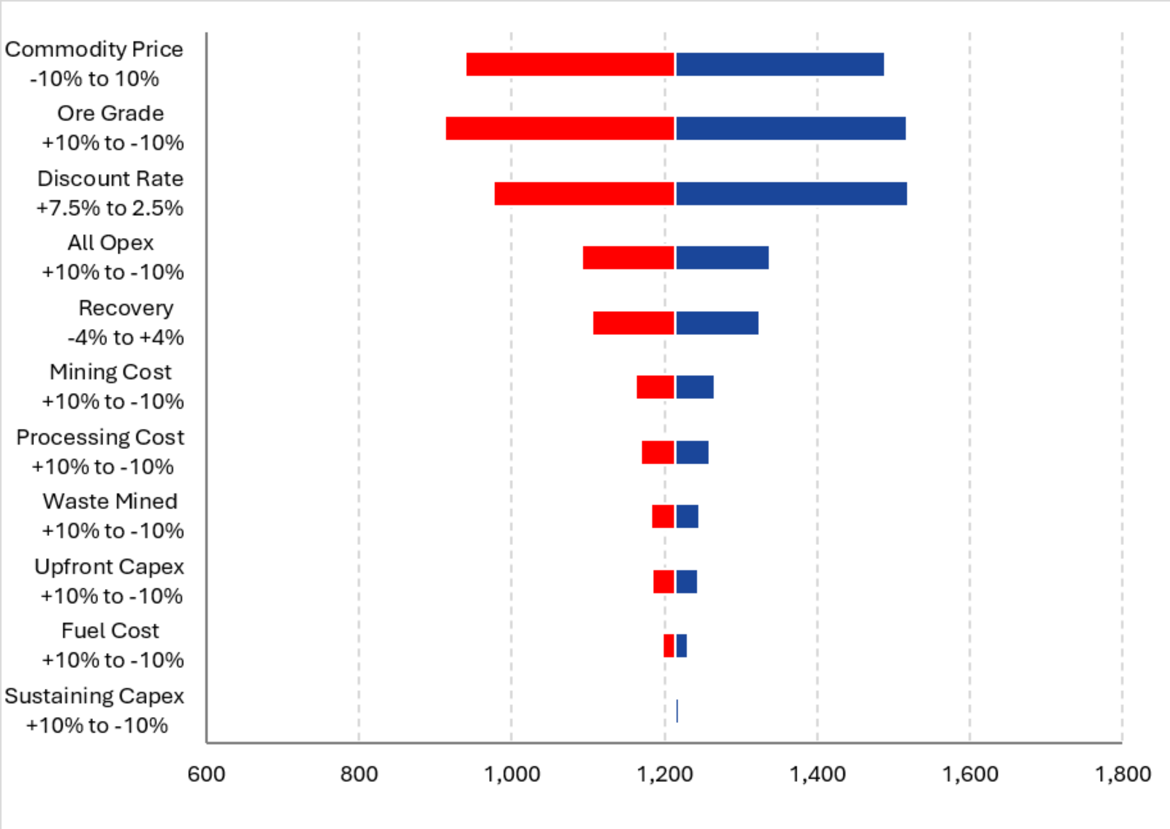


Figure 18.4: Post Tax $NPV_{5\%}$ Sensitivities

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, followed by operating costs and capital costs.

In addition, the NPV is also impacted by the assumed cost of capital, or discount rate.

Table 18.3: Discount Rate Sensitivity of NPV

Discount Rate	NPV US$m
2.5%	1,519
5%	**1,214**
7.5%	976
10%	788

18.3.2 Ore Reserves Only

As outlined above the mining inventory contains a small proportion of Inferred Mineral Resources. As part of the financial analysis a mining and production schedule was developed for processing Ore Reserves only. Under this scenario the key Project outcomes are:

- LOM mill feed of 69.8 Mt at 0.87 g/t containing 1,953 koz Au.
- LOM of 13 years 5 months of processing and total gold production of 1,767 koz.

- NPV$_{5\%}$ of US$1,160m and IRR of 41% at the consensus gold price of US$3,600/oz.
- AISC of US$1,728/oz.

The Ore Reserve case demonstrates that the Project continues to deliver strong economic returns without reliance on Inferred Mineral Resources, providing confidence in the overall robustness of the Project economics.

18.3.3 Processing of Mineralised Waste

As part of the mining and production scheduling a cut-off grade of 0.4 g/t Au was applied to mine scheduling with all materials below this cut-off grade being stockpiled separately in a mineralised waste stockpile. At the consensus gold price, much of this material is economic to process and an approximate production schedule was developed to process this stockpiled material at the end of the LOM. At the consensus gold price of US$3,600/oz the key Project outcomes are:

- LOM mill feed of 93.7 Mt at 0.87 g/t containing 2,255 koz Au.
- LOM of 17 years 11 months of processing and total gold production of 2,005 koz.
- NPV$_{5\%}$ of US$1,249m and IRR of 41% at the consensus gold price of US$3,600/oz.
- AISC of US$1,788/oz.
- Additional post-tax free-cashflow over base case of US$89m.

These results demonstrate the potential value of the mineralised stockpile inventory and highlight a future opportunity to extend mine life and increase Project cash generation under prevailing gold price assumptions.

18.3.4 Water Supply

While work to de-risk the OmAP aquifer is ongoing, the DFS also evaluated a desalinated water supply option to provide additional flexibility and optionality for the Project's long-term water supply strategy.

Under this option, 1.4 Mm^3/a of desalinated water would be sourced from Orano's EDP and supplied from Orano's Trekkopje Mine, located approximately 47 km inland from the EDP, to the Project site.

The proposed transfer scheme comprises a combination of pumped and gravity-fed delivery through the Omaruru River valley. Water would be transferred via a 300 mm ductile iron pipeline, supported by a total of seven pump stations along the route.

Power for the facilities would be supplied from both ends with the base pumping station and first three booster pumping stations powered by a dedicated 33 kV OHPL constructed alongside the pipeline and connected to the existing NamPower Trekkopje substation, which would require the development of a 33 kV feeder bay from the existing 132 kV system. The final three booster pumping stations will be supplied by a dedicated 33 kV OHPL, constructed alongside the pipeline and connected to the Kokoseb HV Switchyard via a step-up 11/33 kV transformer, similar to the power supply for the OmAP borefield. At each booster pump station, a pole or ground mounted transformer and metering would provide 400 V power for the facilities. The OHPL would also include fibre-optic communications will be provide monitoring and control of the borefield equipment.

Capital and operating cost estimates were developed for water sourced from Orano's EDP at Wlotzkasbaken and delivered to site from the Trekkopje Mine. The overall Project capital cost is reduced by US$4.5m versus the development of OmAP, due to the reduce complexity compared to a multi-bore borefield. Water supply cost for the desalinated water option is estimated at US$5.46/$m^3$, compared with US$2.31/m^3 for water supplied from OmAP

The financial impact of utilising desalinated water over the use of water from OmAP for operations is to increase average annual operating costs by US$3.25m, which results in the following key Project outcomes at the consensus gold price of US$3,600/oz:

- LOM AISC of US$1,721/oz.
- $NPV_{5\%}$ of US$1,198m and IRR of 41% with a payback period of 21 months.

These results demonstrate that the desalinated water option remains a technically and economically viable alternative, should additional water supply redundancy or diversification be required.

19 RISKS AND OPPORTUNITIES

19.1 Risks

As is typical for a project at the DFS stage, key risks include the inherent uncertainty associated with Mineral Resource and Ore Reserve estimation, open pit geotechnical performance, mining and construction execution, tailings filtration and filtered stack operation, water and power supply, and the timing of regulatory approvals.

These risks have been addressed through the completion of a DFS to an appropriate level of technical confidence, incorporating JORC Code (2012) compliant Mineral Resource and Ore Reserve estimates, extensive geotechnical investigations and testwork, groundwater modelling, mine planning and scheduling, metallurgical and tailings testwork, and the application of appropriate design criteria, operating assumptions and contingency allowances.

Infrastructure, water supply and permitting risks are being managed through ongoing engagement with Namibian authorities and key service providers, including NamPower and NamWater, completion of environmental and specialist studies, definitive water drilling and testing programmes, and the parallel evaluation of contingency water supply options.

While these matters remain subject to the normal approvals, execution and operational risks associated with project development, the DFS incorporates mitigation measures and execution strategies considered appropriate for a project of this scale and stage of advancement.

19.2 Opportunities

In addition to the risks identified during the DFS, several key opportunities were identified that have the potential to further enhance Project value and operating performance, including:

- **Increased processing throughput:** With greater confidence in the Project water supply than was available at the Scoping Study stage, an opportunity exists to increase plant throughput. Preliminary investigations indicate that processing rates could potentially increase to 7 Mtpa, requiring only an approximately 20% increase in water demand, principally through the earlier processing of stockpiled material. While a higher throughput scenario may result in a modest reduction in average mill feed grade, annual gold production could potentially increase to approximately 175 koz per annum, with total mining volumes remaining broadly consistent with the DFS mine schedule. Further investigations into potential constraints and associated costs, including processing infrastructure, power supply, water supply, and tailings handling capacity, are ongoing.
- **Underground mining opportunity:** The maiden underground Mineral Resource presents a potential opportunity to enhance Project production through the development of a concurrent underground mining operation alongside the proposed open pit operation. Higher grade underground material could displace lower grade open pit feed, thereby increasing gold production without requiring additional milling capacity.
- **Processing of mineralised stockpiles:** Material currently classified as mineralised waste, which is below the mining cut-off grade but above the economic breakeven grade could, represents a potential source of additional future value. Processing this

stockpiled material towards the end of the LOM could generate incremental free cash flow and extend the operating life of the Project. The potential economic benefits of this opportunity is outlined in Section 18.3.3.

- **TSF optimisation:** Opportunities may exist to further optimise the TSF design and reduce capital costs during the detailed design phase, including:
  - Construction of the initial embankment using material reclaimed from within the TSF basin.
  - Removal of the proposed HDPE liner, subject to the outcome of ongoing kinetic leach testing assessing the potential for neutral condition arsenic leaching from tailings materials.

These opportunities present potential opportunities for value enhancement and may be further evaluated as the Project advances through permitting, detailed engineering, and development.